The Cement Plant Operations Handbook

SEVENTH EDITION

The Concise Guide to Cement Manufacture

Philip A. Alsop, PhD

International Cementreview

The Cement Plant Operations Handbook

for Dry-Process Plants

Seventh Edition
January 2019

Philip A Alsop, PhD

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Preface

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

The scope attempted comprehends:

- a consideration only of cyclone preheater kiln technology, which comprises the majority of world production capacity and virtually all kilns installed since 1970
- 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
- an outline of plant assessment and plant valuation
- reference to ASTM and EN specifications for cements and for standard methods
- a collection of process formulae
- a selection of reference data and notes
- references to review articles

Philip Alsop

Florida, USA.

January 2019

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Section B – Process calculations and miscellaneous data

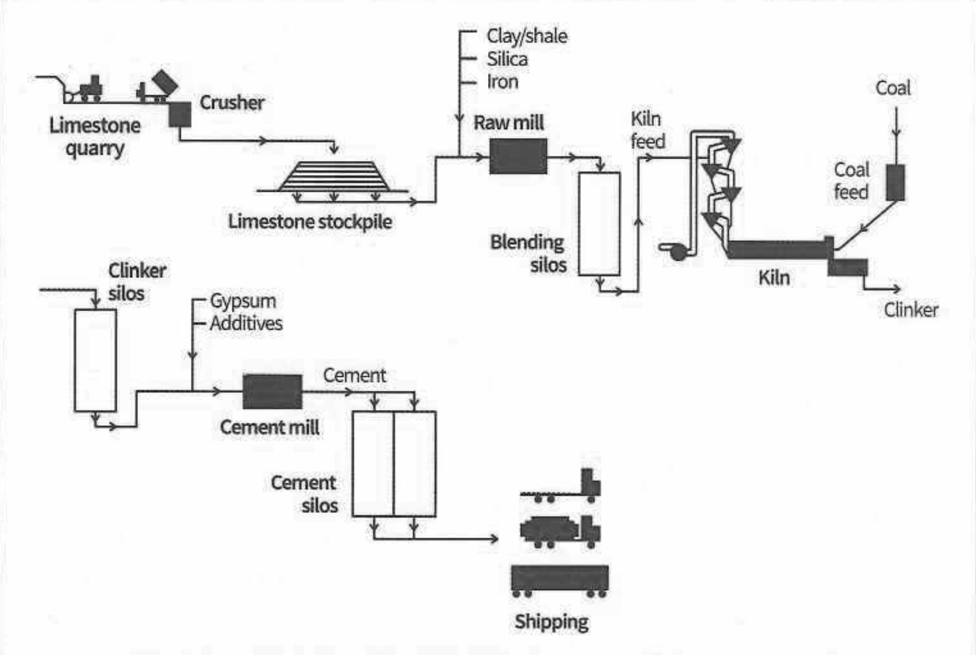
1. Introduction

Cement is "a powdered substance made of burned lime and clay, mixed with water and sand to make mortar or with water, sand and gravel to make concrete: the mixture hardens when it dries" (Collins English Dictionary). It must not be confused with concrete which is "a construction material made of a mixture of cement, sand, stone and water that hardens to a stonelike mass" (id).

1.1 The basics of cement manufacture

Portland cement is manufactured in a series of processes, as shown in Figure 1.1.

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 system, firstly to dissociate calcium carbonate to calcium oxide (lime) with the evolution of carbon dioxide, and then to react lime with the other components to form calcium silicates and aluminates that partially fuse at material burning temperatures up to 1450°C. The reaction products leave the kiln as dark grains of clinker. This is finally interground with a small proportion of gypsum (to control the rate of hydration), yielding a fine, powdered product: Portland cement. Other hydraulic or pozzolanic materials may also be interground to produce composite cements.

1.2 History of cement manufacture

The ancient history of hydraulic mortars is extensive, especially following the widespread use of mixtures of natural pozzolans and burned lime by Greeks and Romans. Constructed in 27BC and rebuilt in 117-125AD, the Pantheon in Rome is the only perfectly preserved building from this period, built of pozzolan-lime concrete with an unsupported dome spanning 45m. Portland cement was developed in the 19th century and is named after its resemblance in colour and character to the naturally occurring stone of Portland Bill, off the south coast of England.

Some of the more significant dates in the development of Portland cement manufacture are (extended from ICR Research, 2002):

1824	Joseph Aspdin patented Portland cement, contemporaneously with Louis Vicat's studies of hydraulic limes.
1845	Isaac Johnson recognised the significance of high temperature to produce tricalcium silicate, making the first modern cement.
1880s	Gypsum added for set control.
1885	Frederick Ransome patented the rotary kiln.
1891	The continuously-fed ball mill was patented.
1900s	Paper sacks introduced for cement.
1904	Cement standards BS12 and ASTM C9 established.
1911	First electrostatic precipitator installed in a cement plant.
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	Vertical roller mill first applied to cement manufacture with rapid development after 1960.
1930s	Introduction of the roll press – rapid development after 1980.
1932	Patent of cyclone preheater kiln with commercial development by KHD from 1951.
1937	Introduction of the grate cooler by Fuller.
1950s	Introduction of mechanical separators in mill circuits, with superior designs appearing from late 1970s.
1960	Introduction by KHD of the kiln bypass to allow use of raw materials with high volatile contents.
1960s	XRF methods for rapid chemical analysis introduced.
1966	Introduction of precalcination, initially as air-through riser-firing.
1973	Introduction by IHI of the flash calciner with tertiary air duct.
1980s	Introduction by IKN of fixed, inclined inlet cooler grates for clinker distribution and Coanda effect air injection.
1980s	Horizontal roll presses start to be applied in the cement industry.
1980s	Expert systems developed for kiln control, following fuzzy logic trials in 1975.
1990s	General introduction of multi-channel burners to cater for more demanding fuels.
2000s	Vertical roller mills used for grinding cement and slag products.
2010s	Industry 4.0 impacts cement manufacturing. The "Internet of Things" and Big Data combined with machine learning (artificial intelligence) increasingly offer new opportunities for plant automation and process optimisation.

1.3 Portland cement in today's world

Concrete plays a vital part in our daily lives, with immense benefits to society when building schools, hospitals, apartment blocks, bridges, tunnels, dams, sewerage systems, pavements, runways, roads and more. It is the world's most widely-used man-made material, with around 1tpa used for each inhabitant. This amount is twice as large as the total of all other building materials, including wood, steel, plastic and aluminium, none of which can replace concrete in terms of versatility, longevity, effectiveness, price and performance for most purposes.

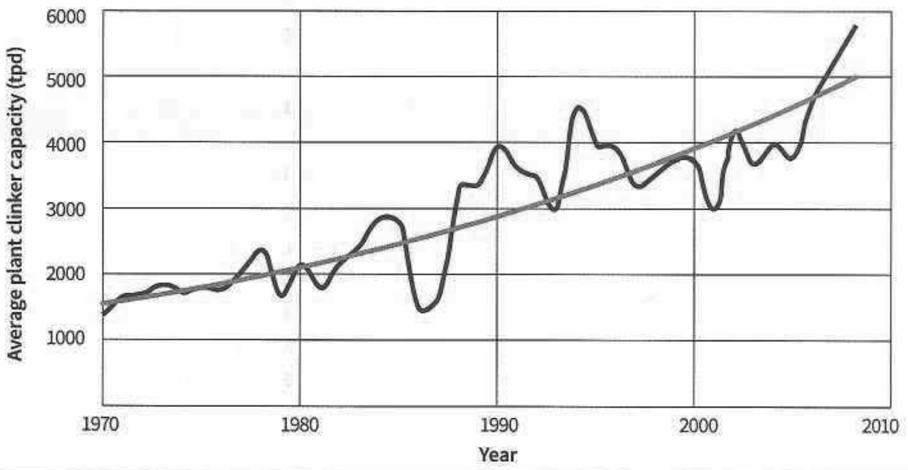
Table 1.1 Growth of world cement consumption

Year	Cement consumption (Mt)	World population (bn inhabitants)	Per capita cement consumption (kg)
1910	30	1.5	20
1925	150	2.0	75
1940	400	2.2	180
1955	600	2.7	220
1974	1000	4.0	250
2000	1500	6.0	250
2005	2300	6.5	350
2010	3300	6.9	478
2015	4056	7.4	548
2020F	4175	7.8	535

Cement is essential for many features of modern life and Table 1.1 shows a long-term growth rate in cement demand of 2-3 per cent per year, accelerating to a little under four per cent for the decade up to 2000 (Portland Bill, 2001). Consumption subsequently increased by over seven per cent annually from 2000 to 2011, reaching 3585Mt, then easing to around four per cent to reach 4056Mt in 2015 (Armstrong et al, 2017). Subsequent annual growth forecasts are relatively low and variable, strongly influenced by expectations for the Chinese market. The estimated total for world cement and clinker trade in 2016 was 189Mt (ibid).

The size of a typical modern cement plant has grown considerably, as illustrated by the trend of orders fulfilled by FLSmidth (Kristensen, 2009) (see Figure 1.2).

Figure 1.2 Cement kiln sizes continue to grow



More recently, this trends towards higher kiln sizes has been maintained, though at a slower rates, with capacities over 6000tpd commonly installed in the more populous countries, including India, Pakistan and Indonesia.

Assuming an average selling price of US$95/t, the cement industry worldwide has revenues of almost three-fifths of Walmart's, and Birshan et al (2015) report that McKinsey's has estimated that the global capital value of the industry is approaching US$500bn. The industry has some significant key features, being both capital- and energy- intensive and presenting difficult transport logistics (Cembureau, 2017):

- Until the more widespread acquisition of lower-cost equipment from China and India in recent years, the cost of a greenfield plant was typically above US$180m/Mta of capacity, equivalent to around three years of turnover, ranking cement manufacture among the most capital-intensive industries. Even with more Asian components, costs remain around the US$100m level.

- Producing 1t cement requires fuel equivalent 60-130kg of oil and some 110kWh of electric power.
- Cement has a low value-to-weight ratio, limiting the distance that cement can be economically transported, before the cost of transport exceeds the cost of the product. For land transportation, the maximum economic haul distance is nearer to 200km than 300km. Maritime transportation for over 4000km with 35,000t cargo can be as cheap as 300km of truck transport.

Cement consumption

Demographic trends, including population age, growth rate and urbanisation, are the main drivers of cement consumption. The pace of growth of demand is influenced by a country's stage of economic development and wealth, often measured by GDP per capita.

The world average of per capita cement consumption in 2016 was 560kg, while the median – which is perhaps a better indicator of central tendency, as it's less affected by outliers – reached 285kg.

The conventional wisdom has long been that, as GDP per capita increases, so does a nation's cement consumption, until a plateau is reached at around US$15,000 GDP per capita (eg, Betts, 2003). In reality, the relationship between GDP and cement consumption is more complex, with data showing a wide variation of trends, but clearing indicating a decline in consumption at high levels of GDP per capita as well as variation in regional trends.

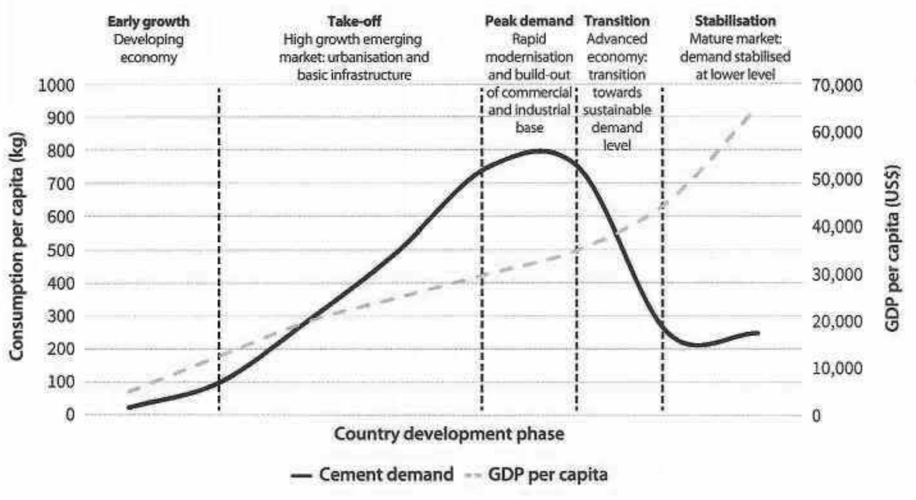
In general, cement consumption per capita tends to be lowest in emerging countries at an early stage of development, as they lack the capacity to invest in public infrastructure and housing.

As a country develops it moves into a phase of higher economic growth, enabling the population to invest in housing and basic infrastructure.

As per capita income increases further and a nation moves towards a higher income status, it continues to build up its infrastructure alongside its industrial and commercial base, thereby maintaining a high level of consumption.

In the later stages of development, per capita cement consumption stabilises at a lower level as the country will have satisfied much of its development and housing needs.

Figure 1.3 Evolution of per capita cement demand through phases of economic development (notional figures for illustration only)



A single new kiln may produce 13,000tpd of clinker, to be ground with other materials to produce over 15,000tpd of cement. This involves the quarrying or purchasing of over 24,000tpd of raw materials (including non-clinker cement components) and their blending, homogenisation and milling, plus the processing of about 2000tpd of fuel, and the milling and dispatch of the product. All of these materials have to be stored, extracted from storage, transported and fed to equipment at a controlled rate – and this is not to mention the high temperatures that are employed and process exhaust gases to be treated!

2. Raw materials

Portland cement composition varies from plant to plant due to both cement specifications and the mineralogy of available materials. A desirable raw mix is one that minimises both the heat input required for clinkering and total raw material cost, while producing a cement of acceptable performance.

2.1 Raw materials

Cement manufacture begins with the winning of raw materials and their transport to the plant. Quarrying involves either ripping or drilling and blasting. In either case, the recovered material needs to be of consistent quality.

Many questions arise in evaluating potential raw materials in relation to the following points:

- location and availability for exploitation
- area and thickness (hence available reserves)
- nature and volume of overburden
- geological structure and geotechnical development constraints
- chemical quality and variations, both laterally and in depth (including composition, mineralisation, cavitation, groundwater chemistry)
- physical properties
- appropriate quarrying and mineral processing techniques.

Concerns to protect biodiversity are emerging in relation to limestone quarry selection and management (Whitten, 2012). Much advice on avoiding sites is based on mammals and birds, and is not specific to limestone. Environmental and social impact assessments (ESIA) have tended to give poor consideration to the specific characteristics of limestone, and especially the smaller and severely range-restricted species which it contains. Very few species have been properly assessed for the authoritative IUCN Red List of Threatened Species and even fewer have been given any form of protection, so that even the best-planned quarry can cause global-level extinctions. Guidelines relevant to the cement sector have been produced (WBCSD, 2016). New European regulations require measures to prevent or eliminate colonisation of locations such as quarry sites by invasive alien species, ie, animals and plants arriving in natural environments where they are not normally found. Guidance for the industry has been produced (Cembureau, 2017).

Some major companies are using a mitigation hierarchy when selecting quarrying sites, summarised as 'avoid, mitigate, compensate'. Table 2.1 outlines a methodology to ensure that sites are selected on the basis of appropriate information, that 'offsets' are established only where mitigation is inadequate or impossible and that meaningful monitoring programmes are agreed.

Table 2.1 The mitigation hierarchy for biodiversity

Avoid at source	Redesign project to remove potential impacts due to the project's feature.
Reduce on-site	Redesign project to remove potential impacts due to the project's feature. Design control systems to abate impacts.
Reduce off-site	Implement off-site measures to reduce impacts that cannot be abated with end-of-process treatments.
Remedy	Repair any residual, unavoidable damage by restoration activities or appropriate interventions.
Offset	Compensate for residual, unavoidable impacts if other mitigation measures are not feasible, cost-effective or already fully implemented (eg, a like-for-like biological offset achieving 'no net loss' ecologically.
Net positive outcomes	Make a positive contribution to biodiversity conservation.

A structured "High Conservation Value" (HCV) approach can also be integrated to ESIAs, identifying areas with exceptionally-high social, cultural, environmental or biological values and developing management plans with stakeholder input to both permit exploitation and ensure maintenance of the values. A quarry-related HCV assessment would require the following:

1. team formation and briefing on project scope, with attention to the surrounding landscape and context
2. compilation of available data (vegetation, geology, soils, landforms, species), including preliminary stakeholder consultations
3. HCV pre-assessment based on existing data
4. planning for fieldwork and agreement on field methods for primary data collection
5. fieldwork by competent professionals
6. data analysis and interpretation, including identification of specimens by competent specialists
7. preparation of a draft report, with maps and management and monitoring recommendations
8. discussion with local government agencies and other stakeholders
9. adoption of a formal HCV management and monitoring plan.

Poor biodiversity performance can damage relations with regulators and potentially blight a company's reputation with its community and stakeholders, affecting access to reserves and permits to operate. The Integrated Biodiversity Assessment Tool (www.ibatforbusiness.org) provides a convenient way to access and use the most up-to-date biodiversity information in line with accepted standards and several cement companies are subscribers.

Quarry rehabilitation is an important consideration. Restoration 'to nature', without infilling, can cost around US\$10/m^2, but re-use for other purposes may be a more attractive proposition (eg, agriculture, fishery, nature reserve, lakes, sports, housing, shopping centre, light industry).

A good quarry management plan is essential, based upon a 3D model, established via a borehole survey throughout the prospective deposit (Ritzenthaler, 2011). Advances in computer power over the past 25 years have provided much more accessible models and plans for selective quarrying to maximise recovery and avoid low-grade material and contaminants.

Quarry management has been greatly facilitated in the current century by the introduction of Global Positioning System (GPS) technology (Mercy, 2001) and more recently – and increasingly – by the use of drones (Anon, Quarry Management, 2017). Full site surveys can be carried out more frequently, less dangerously and at a lower cost than was possible with a team of topographers.

2.2 Raw mix

Normally, cement plants stand upon deposits of limestone (the 'calcareous' raw component), while shale or clay (the 'argillaceous' component) is usually nearby. Limestone comprises 15 per cent of the earth's crust (Kolb, 2001).

A substantial proportion of the raw mix is $CaCO_3$: heating a raw mix evolves some 35 per cent by weight as CO_2 and 1.5t of raw materials are required to produce 1t of clinker. The basis of analytical data must be clearly distinguished between "raw" and "loss free" (or "ignited").

Raw cement mixes vary from a single component (rarely sufficiently uniform for modern quality demands) to four- or five-component mixes. In established economies many easily-accessible, good-quality reserves have already been consumed and cement makers are forced to use materials of relatively marginal quality, requiring sweetening by one or more additives and good blending to assure homogeneity. Kiln feed typically contains 78-80 per cent $CaCO_3$ so that acceptable limestone must 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.

Therefore, materials, as quarried, are typically in proportion as shown in Table 2.2.

Table 2.2 Typical proportions

Material	Share in mix (%)
Limestone (CaO)	85
Shale or clay (SiO_2, Al_2O_3 and Fe_2O_3)	13
Additives (SiO_2, Al_2O_3 and Fe_2O_3)	<1 each

MgO should not exceed 4-5 per cent or the resulting cement may be expansive. Excess alkalis (K, Na) affect both kiln operation (build-ups) and product quality (alkali-silica reactivity). Excess sulphur causes kiln build-ups and limits gypsum addition, which may cause setting problems. The stoichiometric ratio of alkalis to sulphur is best kept between 0.8-1.2 on a molar basis. Excess chloride in kiln feed (>0.015 per cent for preheater kilns, >0.02 per cent for air-separate precalciners) causes serious build-up problems in preheater cyclones.

Table 2.3 shows example ranges from chemical analyses of raw materials and raw meals for producing Portland cement clinker.

Table 2.3 Chemical analyses of raw materials and cement raw meal for the production of cement clinker

Components	Limestone, lime marl, chalk (mass %)	Clay (mass %)	Sand (mass %)	PFA (mass %)	Fe source (mass %)	Raw meal (mass %)
SiO_2	0.5-50	33-78	80-99	40-60	0.5-30	12-16
Al_2O_3	0.1-20	7-30	0.5-7	20-30	0.2-4	2-5
Fe_2O_3	0.2-5.9	4.0-15	0.0-4	5-15	50-93	1.5-2.5
Mn_2O_3	0.02-0.15	0.090	0.051	0.127	0.1-4	0.0-0.5
Fe_2O_3 and Mn_2O_3	0.1-10	2-15	0.5-2		19-95	≤2
CaO	20-55	0.2-25	0.1-3	2-10	0.1-34	40-45
MgO	0.2-6	0.3-5	0.3-0.5	1.0-3	0.5-7	0.3-5
K_2O	0-3.5	0.4-5	0.2-3	1-5	0.1-1	0.1-1.5
Na_2O	0.0-1.5	0.1-1.5	0.0-1	0.2-1.5	0.1-1	0.1-0.5
SO_3	0.0-0.7	0.0-4	0.0-0.5	0.0-1	0-3	0.0-1.5
Cl	0.0-0.6	0.0-1	Traces		0.0-0.5	0.0-0.3
TiO_2	0.0-0.7	0.2-1.8	0.0-0.5	0.5-1.5	0.0-3	0.0-0.5
P_2O_5	0.0-0.8	0.0-1.0	0.0-0.1	0.5-1.5	0.0-1	0.0-0.8
ZrO_2		0.02				
$CaCO_3$	96					
Loss on ignition	2-44	1-20	≤5	6.74	0.1-30	32-36

Source: Schorcht et al, 2013.

If the limestone is not homogeneous, rock from different areas may be blended to maximise recovery and selective quarrying may be needed to avoid low-grade material or problems such as alkalis. Frequent sampling and analysis of materials was traditionally used to control adjustments to mix proportions in cases of poor quarry homogeneity. Online analytical methods that assess the bulk composition avoid sampling errors and investment can be cost effective, whether applied to the blending process or at the raw mill. Prompt Gamma Neutron Activation Analysis (PGNAA) is an established example: material is transported on a belt conveyor through an analysis cabin and continuously irradiated with a neutron beam that stimulates emission of prompt gamma rays to identify the chemical elements present and their concentrations. Alternatives (avoiding the need to comply with atomic radiation regulations) include Laser Induced Breakdown Spectroscopy (LIBS) and Near Infra-Red analysis (NIR), which address the surface layers of bulk material (Müller et al, 2011 and Mühlen, 2017).

Quarrying and hauling are commonly monitored by the performance indicators shown in Table 2.4.

Table 2.4 Quarry performance indicators

Blasting	g explosive/t rock
Stripping ratio	t waste removed/t used rock
Loading	tph of equipment and loader availability (% of required hours)
Hauling	tph per truck and truck availability

The conventional limestone explosive is ANFO (ammonium nitrate activated with around five per cent fuel oil). Usage is typically between about 100-130g/t in France (Ciments Français, 1993), with considerable variation elsewhere, perhaps up to 200g/t. Staggered blasting delays help avoid undue vibrations beyond the quarry. Some North American locations use more explosive to produce a smaller-sized feed to crushers and so reduce the demand upon them, allowing higher throughput.

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

Haulage can often account for around 30 per cent of quarrying costs, and to keep that cost down, haul roads need to be maintained well, ie, to a standard at which a private car can drive at normal speeds.

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

In recent years, partly to address concerns over the long-term sustainability of using virgin quarried materials, kilns have increasingly used industrial by-products (eg, mill scale) and waste materials (eg, water treatment sludge), sometimes in return for disposal fees.

Table 2.5 Some industrial by-products used as alternative raw materials

Ca agents	industrial lime, carbide slurry, lime slurry, water treatment sludge, fertiliser industry sludge, low-grade limestone, paper sludge, sugar sludge
Si agents	foundry sand, silica fume, rice husk ash, catalyst fines
Fe agents	roasted pyrites, steel slag, synthetic hematite, converter flue dust, laterite waste, red mud, mill scale
Si, Al, Ca agents	coal fly ash, metallurgical slags, fluidised bed ash, stone working residues, phosphorus furnace slag, oil shale residues, bottom ash, basalt rock, waste bauxite, china clay waste, demolition waste, crusher fines, soil, tailings
S agents	desulphogypsum
F agents	CaF_2 filter slurry, ore tailings

Substances containing calcined calcium (CaO as opposed to $CaCO_3$) are particularly attractive in avoiding the heat required for dissociating carbonate and resultant CO_2 emissions (see Section B5.5.). Natural raw materials contain minor quantities of various elements such as P, Ti, Cr and Mn and the co-processing of wastes has led to incorporation of a much wider range of elements, with effects reviewed by Bhatty (1995). Different wastes are economically available in different regions: in some places the best supplies are already contracted to major cement companies: as with ownership of raw material reserves, assured and sustained access to supplies of alternative raw materials (ARMs) is of key strategic importance to a cement business. In some regions, it is not unusual for one or more minor components of a raw mix to itself be a blend of several materials, each delivery having been analysed to determine the concentration required.

Apart from raw materials, gypsum and fuel are required, together with various pozzolanic or hydraulic materials (both natural and by-product), when producing composite cements.

2.3 Reserves

A knowledge of limestone reserves is necessary and, to a lesser extent, of shale, particularly when justifying investment to increase 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 allows 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 assumptions.

Significant benefits arise from careful attention to initial selection and sizing of quarry equipment, and taking account of restrictions, such as permitted hours of operation or blasting.

2.4 Crushing

Primary crushers need to accept shot rock with the minimum of wastage or preliminary size reduction. Typically, feed size should be less than 1200mm and either the feed hopper should be protected by an appropriate screen, or a hydraulic breaker used to break oversized rock.

The choice of crusher is influenced by many factors, such as:

1. raw material characteristics (eg, compressive strength, abrasivity, breakage mode)
2. required capacity
3. required product size
4. expected feed size from quarry
5. capital and running costs
6. fixed, mobile or semi-mobile location.

There may be primary, secondary and, occasionally, tertiary crushers in series to achieve a final crushed product of suitable size for feeding to the raw mill. Most crushers are operated in open circuit, though frequently they are also preceded by a screen or grizzly to bypass fine material directly to the next stage. Crushed rock should ideally be -20mm for feed to raw ball mills. Vertical roller mills and roll presses generally require only a single stage of crushing, as they can accept larger feed, typically up to 100mm, the maximum being roughly determined by roller diameters (D_r):

Roller mills	Easy-grinding materials	$<4\% + 0.06D_r$; $<20\% + 0.025D_r$
	Hard-grinding materials	$0\% + 0.06D_r$; $<20\% + 0.015D_r$

Sometimes a simplified rule-of-thumb of not more than five per cent of table diameter is used for materials of average grindability.

Roll press Maximum feed should not exceed $0.05D_r$

Alternatively, a limit of twice the roll gap is used (Liedtke, 2000).

Primary crushers may be located at either the quarry or the plant, largely determined by comparing haulage and conveying costs (Heuer, 1997), whereas secondary crushers are almost always located at the plant. When the total tonnage to be carried between two fixed points justifies the investment, a belt conveyor system often gives the lowest overall transport cost. Mobile crushers are common in aggregate quarries but have been rare for cement (Anon, RP, 1994). Basically, as the quarry face moves, the distance to transport material to a fixed crusher becomes greater, and the expense of maintaining and operating a fleet of dumper trucks increases. Mobile in-pit-crushing is best suited to large, homogeneous, undisturbed, flat-lying deposits, where the total production is mined at one location, allowing point-to-point material transport. If deposit irregularities necessitate simultaneous mining and/or stripping at different locations, semi-mobile in-pit crushers fed by wheel-loaders or a (small) shove-truck system can be an economic solution, especially if the general haulage is uphill. If the truck fleet can travel downhill when loaded, longer hauls can be tolerated and semi-stationary crushers become an interesting alternative.

Horizontal or vertical impact crushers have been a fashionable choice because one machine can reduce run-of-quarry feed to mill feed size with the crushing path gap varied by hydraulic pressure. They are not the answer to all problems and sometimes entirely unsuitable. Some other types are illustrated in Figure 2.1.

Crusher selection depends on various factors, including the work index of the material, its abrasiveness and stickiness, the required throughput, feed size and required reduction ratios, and power consumption. Reich (2007) discusses these matters and provides Table 2.6 to assist in selecting ThyssenKrupp crushers.

Table 2.6 Crusher selection criteria

Crushing type	Maximum crushing ratio	Compressive strength (MPa)	Silica content (%)	Moisture content (%)
Single-shaft hammer crusher	1:100	<150	<8	<10
Double-shaft hammer crusher	1:100	<150	<8	<15-20
Impact crusher	1:20 1:60 Grinding path	<175	<12	<8
Double-roll crusher	1:6 primary	<150	>15	>20
Rollersizer	1:6 primary 1:5 secondary 1:4 sidesizer	<175	>15 <15 Secondary <15 sidesizer	>15 Primary
Jaw crusher	1:5	>150-400	>15	<5
Gyratory crusher	1:5	>150-400	>15	<5

Hammer mills (single and 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 discharge is partially or wholly screened by grates, which risk plugging in damp conditions.

The CenterSizer crushes material between teeth in the middle of a crushing chamber, with the required shear and tensile forces generated by high rotor torques at low circumferential speeds. Feeder fines can pass between the rolls without being further crushed, providing low power consumption, wear rate, and fines content. A SideSizer crushes material at the sides between crushing teeth and adjustable crushing combs.

Double-toggle jaw crushers have a simple compression motion with jaw angle of 15-20°, while single-toggle jaw crushers also have a measure of vertical jaw motion, adding attrition to compression. They can handle some sticky materials, but wear increases.

Gyratory crushers operate by pressure between a gyrating cone and a stationary or spring-loaded crushing ring. Cone crushers are another variant.

Double-roll crushers 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.

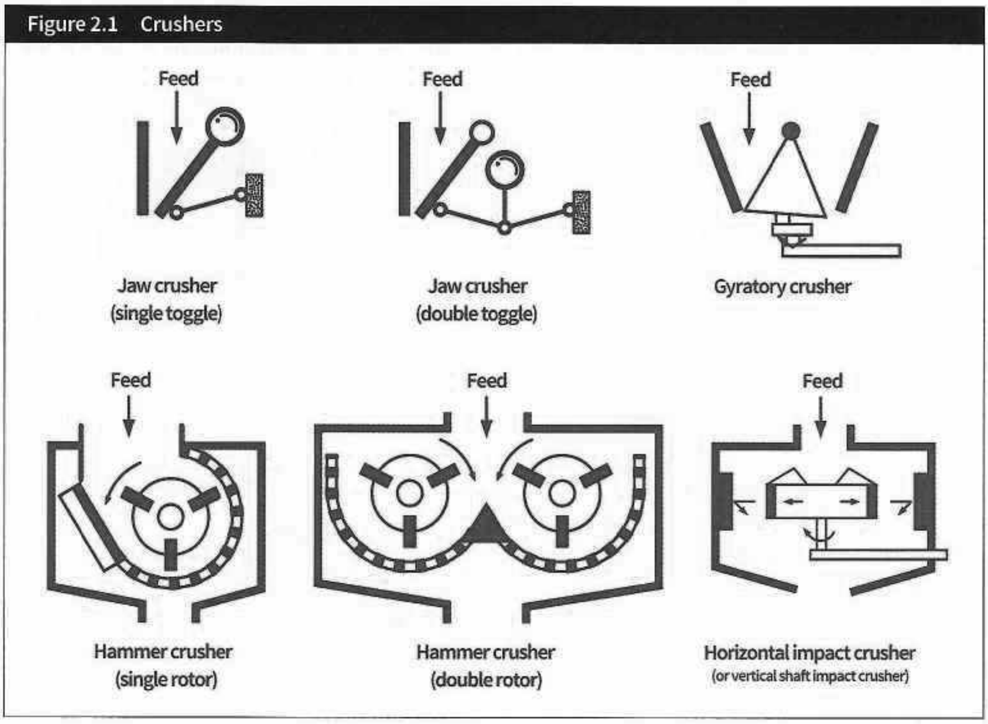
Monitoring of crusher operation requires the following data:

- production rate (tph) operating hours (h)
- involuntary downtime (h)
- feed moisture (%)
- product screen analysis.

2.5 Drying

The handling characteristics of materials relative to moisture content vary widely. In general, a raw mill can dry material of up to 15 per cent overall moisture, but pre-drying may be necessary for certain materials (or in certain seasons) to facilitate handling. On the other hand, pre-dried materials may also cause serious handling problems. Steam evolution from hot material causes build-up and can plug dust collectors, whilst the dry fine fraction is liable to flush from intermediate storage. There are advantages in both handling and heat consumption if raw materials can be taken to the raw mill and dried within it.

Figure 2.1 Crushers



Drying is commonly carried out in combined crusher-dryers or occasionally in autogenous mills, using either dedicated hot gas generators or waste heat from the kiln exhaust. The air flows require dust removal and their high humidity has favoured electrostatic precipitators over bag houses.

Monitoring includes:

- production rate (tph) and operating hours (h)
- involuntary downtime (h)
- hot gas temperature (°C) and discharge gas temperature (°C)
- feed moisture (%) and product moisture (%)
- heat input (for dedicated heat generators) (kcal/kg).

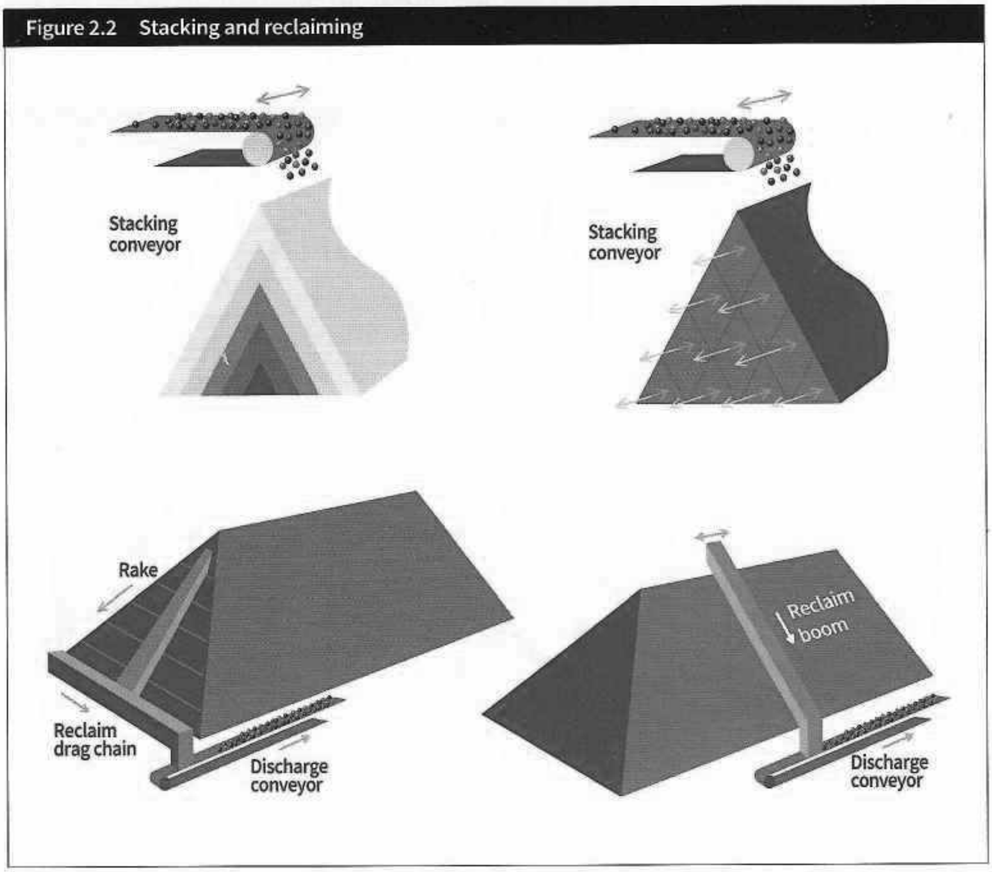
Heat consumption is most conveniently calculated on equivalent clinker basis so that it may be combined arithmetically with kiln heat to assess total process heat.

2.6 Pre-blending

If the limestone and clay/shale both have highly-consistent properties, or if 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, low-frequency, day-to-day material variation can be mitigated and buffer storage more economically provided, by a mixed pre-blend stockpile, either longitudinal or circular. Strictly speaking, 'homogenisation' refers to a single material and 'blending' to two or more. 'Pre-homogenisation' refers to homogenisation before the material is sent to the raw mill and is most frequently applied to the main raw mix component, limestone. Where two materials are to be blended, they are fed to the pile in the desired proportion.

The limestone (and clay/shale if premixed) are metered and fed simultaneously to a feed conveyor. There are two major stacking systems (see also Figure 2.2):

Figure 2.2 Stacking and reclaiming



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). Here, finer material is found at the centre of the pile and coarse in the lowest level and near the surface.
Windrow	stacking in longitudinal strips side-by-side and then in successive layers; avoiding the segregation of larger material (>100mm) falling to the foot of the pile, and managing the material reclaimed from the typically off-target end-cones which characterise chevron stacking (Petersen, 1994) 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), but shows advantages when treating a blend of materials with different handling characteristics (Paone et al, WC 2017).

Chemical analysis of material fed to storage is essential, especially if premixed. This may be either by continuous sampling followed by periodic conventional analysis (Andersen and Pedersen, 2004, and Narayanan and Koenig, 2004$^{1-2}$), or by neutron activation methods (Foster and Bond, 2006; Leetham, 2008, and Harder, 2009), continuously analysing a portion of the material flow diverted to fall though a chute or, better, the whole stream of material traveling on a belt. The latter method has the obvious advantage of rapidity and avoids a very difficult sampling problem.

The main objective of blending is to achieve target consistency of chemical analysis. At intermediate process stages it is convenient to consider the improvement ratio as a measure of effectiveness. This ratio relates to a chosen parameter, eg, LSF or carbonate content. Typically, the standard deviation of results for 24 samples taken hourly is used as a guide. In-pit blending can iron out relatively long-term or day-to-day fluctuations in material properties, whilst in-plant equipment compensates for short-term fluctuations that occur with an hour-to-hour frequency, or even more rapidly.

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

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

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 are ignored (which should be eliminated by normal handling) (Labahn, 1995).

Recovery from the pile takes place transversely to the longitudinal stacking, 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, 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 – probably the weekend shutdown of the quarry, or the time required for maintenance of crushing equipment. Total capacity of 7-10 days consumption is normal, ie, around 30,000-40,000t of material for a 3000tpd clinker plant. Pre-blend piles are usually covered, both to prevent rainfall on fine absorbent material, and to contain fugitive dust.

Circular pre-blends are sometimes employed to save space and avoid the difficulties caused by end-cones but do not allow subsequent expansion and present a challenge when a switch is made between different types of clinker, each with distinctive chemistry.

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

The effort to reduce variability starts with quarry planning and management. A model of the quarry in compositional terms is crucial and, with a quarrying plan, allows predictions for the general manner in which variations will occur as the reserves are exploited. Such work identifies general trends for changes that appear over periods of weeks, months or years and that can be evened out by use of appropriate equipment and working patterns. A stockpile constructed over a period of days has the potential to reduce variations which occur over a similar timescale. Residence time studies of conventional raw milling circuits suggest that these can even out variations occurring over periods of up to about 20 minutes. Later, after treatment in blending silos, raw meal variability undergoes a further small improvement, either due to layering into a selection of kiln feed silos from a single blending silo or as a result of internal layering and funneling. To achieve optimum performance a combination of blending components should be used to best match the full spectrum of periodicity of raw material variability.

By way of example, quarry face variability might be 40, which could be reduced by a factor of 1.2 by the time material arrives at the blending stockpile, which could improve matters by a further factor of 6. Moving from the raw mill hoppers to the blending silo could introduce a further ratio of 2:1 before a significant 6:1 improvement upon blending silo treatment, followed by a further 1.2:1 improvement during feeding to the kiln system. These improvements are cumulative and, multiplied together, indicate the overall system capability as 86.4 – more than sufficient to reduce the quarry face variation below a typical kiln feed target variability of <1. It is essential that all components in the system work together. If the stockpile equipment fails, then the cumulative improvement in variability drops to a meagre 14.4, insufficient to cope with the quarry face material.

Apparently insignificant changes to quarry operation – often by newly-arrived managers – can have significant impacts upon plant performance that are not always immediately apparent.

2.7 Storage and handling

Bulk material handling is a major activity and bins for raw materials, intermediate products and finished cement must reliably provide controlled flows of materials upon demand. The tonnages of material to be stored at various process stages are outlined by Deolalkar in Chapter 13 of his book (Deolalkar, 2009), while his Chapter 16 discusses the features of materials handling systems.

Hoppers and silos may be intermediate stores to compensate for differences in demand of subsequent processes, or feed bins installed for feed rate control to equipment. They 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 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.

Funnel-flow is the most common. Bins may either be flat-bottomed or equipped with discharge aids. It is typical for large-volume stores and is considered acceptable where:

- non-cohesive (free-flowing) products are involved
- product segregation is unimportant
- product degradation, including lump formation, poses no problems
- feed rate consistency is not of prime importance.

Hopper and silo flow processes are reviewed by Bresler (2003). Funnel-flow bins can typically not be completely emptied, resulting in reduced live capacity. The material discharge order is in a 'first-in, last-out' (FILO) sequence.

Bulk solids do not flow consistently in vertical or very steep channels and tend to form both rat-holes and temporary arches above voids, leading to erratic discharge rates.

The mass-flow concept is preferred for feed bins and is also recommended for storage purposes with cohesive products or where qualitative consistency of the reclaimed product is important.

Mass-flow bins typically have a slender shape, steep wall slopes and relatively large outlets. Flow out is uniform, independent of the material head. The flow sequence is 'first-in, first-out' (FIFO), avoiding product deterioration due to excessive storage in rest periods.

The bulk materials most difficult to extract are generally those containing a substantial quantity of both fine particles and moisture. Wet clay materials are particularly prone to difficulties and specially-designed wet-store feeders may be required (Maynard, 2001).

3. Raw milling and blending

Raw milling must produce sufficient feed to sustain the required kiln production, meeting targets for fineness, chemical composition and moisture. With multiple tasks to accomplish, process flow and control become quite complex. Operators need a clear understanding of both basic principles and interrelationships to get the best results.

3.1 Raw milling

Usually each raw component is stored and fed separately to the mill, from which product is sampled and chemically analysed to determine adjustment of feed proportions by addition of corrective components to maintain the target clinker composition. Older mills may have silos for raw material storage, but the occurrence of feed problems with moist material has led to a preference for hoppers.

Mill product is monitored either by continuous online analysis or by laboratory analysis of hourly grab or composite samples. Computer control effects feed corrections and maintains the desired optimum average composition.

Powder samples can be acquired more or less continuously by devices exposed to downward material flow, while rock samples require grinding and blending. Sample extraction, preparation and transport to a central laboratory mean that corrective steps are delayed. Modern online techniques, outlined in Section 2.2, allow rapid correction and greatly reduced variation of mill product chemistry. Any continuous analysis system requires reliable and uninterrupted material transport and feeding, which can be a serious challenge.

Operation of the various types of mill and separators is described in more detail under cement milling (Section 6.2). Grinding aids are rarely employed because savings via decreased power consumption are not cost effective. However, Whitehead and Kaya (2018) point out that in some circumstances the amounts of coarse quartz and calcite can be reduced, and improve kiln output rate and fuel consumption due to enhanced combinability of the kiln feed.

The main difference to cement mill operation is the need to dry raw feed and assure transport of slightly moist material within a mill. Also, many raw materials are softer than clinker, and operations must avoid coarse residues in kiln feed that are difficult to combine in the kiln and would result in high clinker free-lime, and/or excessive fuel consumption. Optimum kiln feed fineness must be determined empirically and should be as coarse as the kiln will tolerate.

Table 3.1 Recommended coarse particle limits for kiln feed (% by weight greater than specified size)

Silica	>200µm	Below 0.5
	90-200µm	Below 1.0
	>45µm	Below 2.0
Calcite	>125µm	Below 5.0

A narrow size distribution is optimal because fines tend to increase dust loss by entrainment in exhaust gas. The use of larger residues may be acceptable when they arise from natural rocks but requires checking by laboratory combinability tests. Lörke et al (2013[1-2]) describe a plant trial using an extremely-coarse raw mix, following laboratory tests in which calcareous raw material was ground more coarsely. A major gain in mill output was achieved, with no detrimental effect upon cement quality.

Materials are dried in the raw mill whenever possible, because:

- Less equipment is needed.
- Waste heat from preheater exhaust is used effectively.
- SO_2 from preheater exhaust is scrubbed.

Mill production is determined by the grinding power available and feed grindability, but for high-moisture feed, the system drying capacity may be the limiting factor. Typically, the target moisture level ex-mill is below one per cent to assure good flowability of raw meal for ease of handling.

Hammer/impact mills can be used for pregrinding/predrying purposes, ahead of a conventional mill, often with a type of flash drier.

Raw mill sweep with preheater exhaust is called compound operation. The raw grinding and clinker burning operations become interdependent and constant kiln operation is required to ensure control of mill temperature and pressure. When the kiln is down, the mill must also stop unless auxiliary heat is provided. When the mill is down, preheater exhaust must be cooled to maintain acceptable gas flow and temperature for dedusting. Gas conditioning by a water spray tower is usual, but if water is scarce, ambient air tempering can be used. Air-to-air heat exchangers were not commonly used for dust-laden exhaust gases due to the tendency to cause build-up and plugging, but examples are becoming more common.

A new raw mill classifier is now invariably of the 'high-efficiency', rotating cage-type: the rotational speed controls product fineness at constant gas flow.

Raw milling can consume 10-30kWh/t, depending upon the materials and equipment, and developments aim to reduce this figure. Around 90 per cent of new raw mills are vertical roller mills, though many ball mills (commonplace until the 1970s) are still in use.

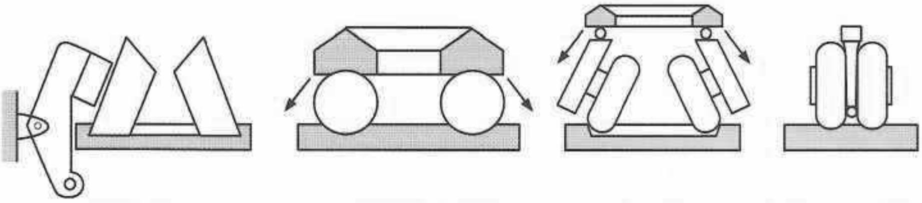
Roll presses are also used, particularly in upgrading existing ball mill circuits either to increase production or reduce power consumption. A press can accept material up to around 75mm in size and act as either a pregrinder for a ball mill (perhaps with a disagglomerator to strip fines from the pressed cake), or as an integral unit with its own gas circuit for drying. Such presses (with a limited number of suppliers) have only made a limited market penetration, a key element of success being improved separators, such as the KHD Humboldt Wedag V-Separator, introduced in 1994, although other designs tend to be preferred for new installations.

Ball mills equipped with a drying compartment and adequately swept with hot gas (1.5-3m/s above the ball charge), can handle feed with above 10 per cent moisture and a maximum size of around 25mm, preferably much less. With a tandem hammer mill, the maximum feed moisture content may be around 12 per cent if the mill exit diameter is large enough to keep the gas velocity low. Centre discharge mills ('double rotators') and fully air-swept mills (5-6m/s) can also dry up to 14 per cent moisture. One of the limitations of an air-swept ball mill is dependence upon the gas stream for material transport. The recirculating load is limited to about 75-150 per cent. Efficiency can be gained by using higher recirculating loads and conveying the material mechanically can achieve this. Typical bucket elevator installations are designed with a recirculating load capacity of up to 500 per cent, with the added advantage that the grinding efficiency within the mill tends to be greater due to improved control of material flow. Bucket elevator mills are not designed for gas sweeping and so are limited to raw materials with below about 4-5 per cent moisture.

For feed moisture of up to six per cent, drying is effected by part of the kiln exhaust at temperatures of up to 300 °C. Obviously a high drying requirement may be inconsistent with maximising kiln thermal efficiency – generally five- and six-stage preheaters are only employed where subsequent drying requirements are minimal. Alternatively, but more expensively, dedicated hot-gas generators can be used for drying in the mill or hotter gases from a lower cyclone stage or the cooler exhaust may be used. Drying is also aided by heat dissipation from mill drive power which equates to approximately 1t moisture per 1000kWh. Higher moisture burdens require larger mill bodies, increasing the cost.

Vertical roller mills have some 33 per cent lower specific power consumption than ball mills and have come to dominate the scene since their introduction during the 1970s. These mills (see Figure 6.3.) comprise two to six rollers which are hydraulically pressed on to a horizontal rotating grinding table. Feed material is directed to the centre of the table and thrown outwards by rotation under the rollers and into a rising air current at the periphery that is directed by means of louvre ring ports. The air passes through an integral rotary classifier and fines pass out with the air current while coarse material falls back to the table. Depending on different suppliers the rollers are, frusto-conical, cylindrical or spherical in shape, and in some cases may have an axis of rotation inclined at some 15 degrees to the horizontal, as indicated schematically in Figure 3.1.

Figure 3.1 Schematic illustrations of some roller and track designs



Loesche mill
Tapered-lever
mounted rollers

Peters ring-ball mill
Track-guided balls

Pfeiffer mill
Track-guided
crowned rollers

Polysius roller mill
Track-guided
crowned rollers

Source: Lynch and Rowland, 2005

Material dries in air suspension between table and classifier. The circulating load is typically 800 per cent. Roller mills are prone to vibration due to an unstable grinding bed, often caused by fine, dry mill feed. Spraying water directly to the bed is the usual solution, which also conditions the gas. Roller mills can dry and grind coarse moist feed (max ~18 per cent moisture with an auxiliary furnace) of up to 125mm in size, preferably 80-100mm. They are compact, with the classifier built into the mill body.

Operationally, roller mills are quite sensitive to changes in parameters. Abrasive raw materials such as sand can cause quite rapid wear of grinding surfaces. Material is retained on the table using a dam ring at its circumference. As the table wears, the depth of material on the table increases and mill power consumption increases. This is not normally a problem unless the mill motor is approaching full load, in which case the dam ring height must be reduced to compensate for table wear and avoid restricting feed rate. Detection of tramp metal in mill feed and its removal are important precautions.

Specific power consumption depends upon material hardness and mill efficiency. For ball mills the range is from approximately 10kWh/t (mill drive only) for soft, chalky limestone to 25kWh/t for hard materials. Compared to ball mills, energy savings for VRM circuits can be around 30 per cent. FLSmidth claims the world's largest VRM installation for raw materials, rated at 8700kW and capable of producing 750-1000tpd at around 26kWh/t.

Air emerging from a mill carries 50-500g/m^3 of fine material, requiring a velocity of some 15-20m/s to ensure entrainment, demanding the use of powerful fans. Raw meal is collected by a fabric filter or other dust collector – with its function being as much to collect product as to clean gas.

Poittier (2018) reviews the latest evolution of the Horomill, suggesting that the correct configuration for raw milling is closed circuit with a third generation TSV™ classifier and perhaps a flash dryer in the gas circuit.

Raw mills are monitored by:

- production rate (tph)
- operating hours (h)
- involuntary downtime (h)
- mill motor specific power consumption (kWh/t)
- mill circuit power consumption (kWh/t)
- maximum feed size (mm)
- product fineness (% finer than 300m and 125m)
- feed moisture (%)
- product moisture (%)
- pressure drop (mm WG)
- limestone (%); clay/shale (%); additives (%)
- inlet and outlet gas temperatures (°C)
- furnace fuel consumption (kJ/t of dry product)
- noise level (dB).

Additional operating parameters required periodically include:

- circulating load (%)
- steel usage (g/t)
- chemical analysis of +90µm fraction.

Analysis of the coarse (+90µm) fraction may be lime-rich or, more likely, silica-rich, relative to the total sample but should show constant bias.

Downtime analysis can usefully highlight reasons for lack of availability and identify target areas for improvement. For example, significant mill blockages would indicate an issue related to drying capacity. Factors external to the mill are highlighted, such as feeder starvation due to handling and drying problems in the crushing department.

3.2 Blending

This is another critical process stage. Deficiencies will impair product quality, reduce kiln output, increase both fuel consumption, refractory wear and, potentially, grinding costs with lower throughput for similar 28-day strengths. For optimum results a kiln feed with the correct chemistry and particle size distribution is essential - with, above all, a minimum of variability. A major cause of customer dissatisfaction and complaint is not the absolute concrete strength level achieved with the cement but its variability.

Because customers require similar properties for cements from different sources, and also to optimise clinker production, there has been a pragmatic trend to converge on a 'standard' clinker chemistry in a narrower range than often repeated from historical descriptions (Moir, 2003):

- LSF: 95-97%
- silica ratio (SR): 2.4-2.6
- alumina ratio (AR): 1.5-1.8.

Most plants have to use corrective materials such as sand and iron oxide to achieve this range, and the blending system has to produce a raw feed that uniformly yields such a clinker.

The two major types involve turbulence (in which the material is tumbled about by injection of high volume air through air-pads on the floor) and controlled flow (CF - where sequenced light aeration of segments of air-pads causes layers of material in the silo to blend by differential rates of descent). CF silos may have multiple discharge chutes, or an inverted cone over a centre discharge within which the meal is fluidised. Approximate compressor power consumption is shown in Table 3.2.

Table 3.2 Compressor power consumption

Blending silo design		Power consumption (kWh/t)
Turbulent mixing (airmerge)		1.5-2.5
Controlled flow	inverted cone	0.25-0.50
	multi outlet	0.10-0.13

Source: Bartholomew, 1995

Turbulent mixers 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 the kiln. Continuous blending involves simultaneous feeding of the silo, overflow to a second silo and final discharge to kiln feed.

Modern equipment generally uses continuous CF silos, each having capacity of more than 24-hours' kiln feed and yielding a blending ratio (or 'blending factor') of 4-8, or around 2-3 for older silos. Note that a given silo will show a lower blending efficiency if the feed is consistent. The retention time of raw meal in a blending silo affects blending ratio and may be easily monitored by the addition of zinc oxide or fluorescein to silo feed (see Section B4.7). Apart from power savings, the effective capacity of a CF silo is some 20 per cent greater due to the higher bulk density of meal which is not heavily aerated. The design of modern blending silos is described by Halbleib (2003).

Blending silos should be monitored by:

- blending ratio ($s_{feed}/s_{product}$)
- compressor power (kWh/t of throughput).

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

To ensure good blending action, it is preferable that material chemistry varies both above and below its target set-point on several occasions during the silo residence time.

With the availability of real-time, online analysis of mill feed or product, chemistry can be maintained within narrow limits and modern plant designs frequently dispense with kiln feed blending or cut down the size of prehomogenising stockpiles. However, such procedures do not work well when faced with certain origins and frequencies of variability in a major raw mix component, or with corrective components that are either difficult to extract from storage and accurately dose, or that are difficult to combine within the kiln (Harrisson, 2013). Rather than investing in equipment to correct the chemistry of the feed on its way to the kiln, an improved quarry management system to deliver a more consistent supply of limestone provides a superior solution.

3.3 Kiln feed

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

Suspension preheaters lose a fraction of kiln feed entrained in exhaust gas. As this fine fraction is usually of a typical composition, kiln feed analysis must be biased to yield the desired clinker composition. The dust loss, some 5-12 per cent of kiln feed, is not usually collected until after it has passed through a raw mill or dryer, so that dust catch is not the same quantity or composition as preheater dust loss. Even if the dust collector catch is returned directly to the kiln, it must still be compensated. Likewise, care must be exercised to minimise chemical disturbance due to dust return, particularly when the raw mill is down and the dust collector catch changes from mill discharge instead of kiln discharge. The least negative option is feeding to the blending silo or to a separate storage tank for subsequent controlled return.

If the kiln exhaust passes directly and continuously to dust collection, then the dust may be returned directly to the preheater - sometimes, if low in quantity, re-entrainment can be minimised by insufflation at the hood injection at the upper end of the kiln. Either way, the return rate should be controlled.

Kiln feed is monitored by chemical analysis on four- or eight-hourly grab samples - not cumulative samples - to determine standard deviation results (see Section 2.6). Analysis is conventional for major oxides with variation monitored statistically in terms of C_3S or LSF.

Kilns, particularly the larger ones, are intolerant of variations in feed chemistry. The general rules for chemistry control at the raw milling stage are similar, no matter how many mix components are available, and are basically simple:

- If the chemistry is not at the desired level: change it!
- If a parameter is below the desired level: increase it to above that level. (Known as 'crossing the target line' - as seen on a control chart - and failure to do this usually results in 'off-target' batches of raw meal.)
- When making a change, pay due regard to previous results and weighfeeder settings. (This is important when batch blending, where the concept of keeping a weighted average of all material chemistries entering a silo is often advantageous.)
- Where possible, use continuous samplers: spot samples can often be misleading and cause erroneous corrections.
- Residues are very important and their testing and control demands as much care as bulk chemistry checks.

Variations in fuel ash should not be overlooked and ash should be considered as part of the raw mix argillaceous component. If the quantity fed into the kiln varies, the chemical variability of the clinker will also vary. Plants with high fuel consumption and/or high fuel ash contents should minimise variations by blending the fuel(s).

It should be remembered 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 operators need to respect maximum tolerable limits upon variability – see Table 3.3.

Table 3.3 Kiln feed target variabilities for various factors

These are maxima and should be improved on where possible

Variable	Max standard deviation
LSF [or C_3S] (%)	1.0 [2.5]
$CaCO_3$ (%)*	0.1-0.2
Silica ratio	0.07
Alumina ratio	0.05
90µm residue (%)	1.0
Coal ash (%)	1.5
Feed rate (%)	±1.0

* *$CaCO_3$ is an alternative to LSF or C_3S (potential tricalcium silicate content – see Section B5.1), which are the preferred control variables.*

The importance of controlling variations is indicated in Table 3.4.

Table 3.4 Likely influence of chemical variations upon process and quality parameters

Effect on fuel consumption	
± 1 LSF %	10-15kcal/kg
± 0.1 SR	6-18kcal/kg
± 1% 90µm residue	5-10kcal/kg
- 1% free lime	10-20kcal/kg
Effect on output	
+ 1 LSF %	- 1% output
+0. 1% SR	-2 -3% output
Effect on quality	
±1 LSF %	±5% strength @ 1 day
	±1.5% strength @ 28 days
+1% free lime	- 2% strength @ 28 days

Kiln feed is normally conveyed by bucket elevator to the top of the preheater. Pneumatic conveying wastes energy, and feed de-aeration is desirable before injection as the entraining air adds to the kiln ID fan load and may reduce capacity.

Although about 1.55t raw materials are required to produce 1t of clinker, kiln feed-to-clinker ratio is typically 1.65-1.75 as weighed, due to the loss of dust with exhaust gas, later 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 a loss-free basis.

4. Flames and fuels

The kiln burning zone is at the heart of the manufacturing process and the supply of heat energy results from combustion of fuels. Combustion science involves thermodynamics, fluid mechanics, chemical kinetics and transport processes, and did not emerge until over 100 years ago. In recent years new insights have arisen through advances in computer capability, experimental techniques and asymptotic methods of applied mathematics.

Kiln operators can control the following factors when firing fuel in a kiln or calciner:

- fuel type
- fuel handling
- heat transfer
- burner momentum
- excess air
- emissions.

Other factors that are typically outside immediate operator control can have a dramatic effect on combustion:

- secondary air[1] momentum
- tertiary air momentum
- kiln aerodynamics
- calciner aerodynamics.

Unfortunately each factor is integral to one system and it is difficult to review one independently of another. This section attempts to address the broad subject of combustion and how it applies to making cement.

4.1 Chemistry of combustion

Combustion is a specific group of chemical reactions where a fuel and oxygen react at a sufficiently-high temperature to evolve heat and combustion products. Combustion can vary in rate from a very slow decay to an instantaneous explosion: a kiln requires steady heat release at a certain rate.

Oxidation of industrial hydrocarbon fuels can reasonably be simplified to four basic reactions:

- **The complete oxidation of carbon**

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

- **The complete oxidation of hydrogen**

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

  The difference in the physical states of the water produced in the reaction causes the complication of gross calorific value (GCV, or higher heating value) and net calorific value (NCV, or lower heating value) for fuels. Users face the conundrum of paying for gross heat but utilising only net heat.

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[1] ***Primary air** enters the kiln via the burner pipe, with the fuel.*

***Secondary air** is drawn in around the flame. In a cement kiln, this is hot air drawn from the clinker cooler via the kiln hood.*

***Tertiary air** in a cement kiln system is also hot air from the clinker cooler, in this case supplied to the precalciner by a duct.*

Table 4.1 Gross and net calorific value for selected fossil fuels

Fuel	Calorific value		
	Gross	Net	Difference (%)
Coal (kcal/kg)	5500-7100	5400-7000	2
Oil, #6 (kcal/kg)	10,200	9700	5
Natural gas (kcal/m^3)	6200	5600	10

- **The incomplete oxidation of carbon**

In the event of imperfect combustion, some carbon is only partially oxidised, forming carbon monoxide. The main effect is to reduce heat release.

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

Only just over half of the heat is released compared with complete combustion to carbon dioxide. Burners producing carbon monoxide as a result of bad fuel/air mixing show a significant drop in efficiency and significant levels of carbon monoxide should be avoided.

- **The oxidation of carbon monoxide**

Often, reactions proceed rapidly to the point where CO is formed and then slow greatly until CO burn-out is achieved, according to the following reversible reaction:

$$2CO + O_2 \rightarrow 2CO_2 + 173kJ/mole\ (41kcal/mole)$$

The combustion of dry carbon monoxide is extremely slow, but if H-containing radicals are present, it increases significantly.

4.2 Fuels and flames

Both natural and manufactured fuels vary widely in composition and physical characteristics, as considered below. It is important that procurement staff have a proper appreciation not only of the benefits of savings in fuel purchase price but also of the importance of assuring a consistent quality of supply and of costs that may be involved in storing and handling fuel from a new source, and in compensating for the adverse effects that it may produce during kin operation.

Gaseous fuels

The characteristics of some typical gases are given in Section B6.3. Natural gas has been used in kilns for many years. While the basic constituent is methane, which has narrow flammability limits, the presence of other gases affects calorific value and density. Higher hydrocarbons widen flammability limits and assist flame stability. Natural gas has a low carbon content and produces low-emissivity flames with a detrimental effect on radiant heat transfer and plant efficiency unless some dust particles are added. The high hydrogen content means that natural gas requires more combustion air per kJ of heat released than most other fuels. It also produces more exhaust gases, though with a smaller proportion of CO_2. For a kiln using 850kcal/kg clinker and two per cent excess O_2, flame temperatures and gas volumes are shown in Table 4.2.

Table 4.2 Flame temperatures and gas volumes

Fuel	Flame temperature (°C)	Combustion gas (Nm^3/Mcal)	Total exhaust gas (Nm^3/t clinker)
Coal	2250	1.23	1360
Oil	2350	1.31	1420
Gas	2400	1.45	1550

Oil fuels

Oil fuels are produced by refining crude oil or manufactured from coal. They are classified as distillate fuels (such as kerosene and diesel oil), or as residual fuels. The latter come in a range of viscosities and are classified differently in different countries. Typical characteristics are given in Section B6.2. Residual fuels must be heated to become pumpable and reduce viscosity to enable atomisation. The heavier the fuel, the more heat is needed. Owing to their tendency to solidify when cold, handling systems must avoid 'dead legs' as much as possible. Lighter 'white' oils yield a better profit than black fuel oils and refineries increasingly manufacture more of these. The black oils are heavier with different characteristics to the white products, having increasing quantities of asphaltenes. 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 form 'gels' in tanks and fuel handling systems with disastrous results. Proposed fuels should always be tested for compatibility with the existing fuel before purchase.

Oil atomisation is important because initial drop size determines the size of the cenosphere that is formed and hence the droplets' burn time. Oxygen diffusion is dependent on surface area, but oxygen demand depends on particle mass. Since surface area depends on diameter^2 and mass on diameter^3, larger drops take longer to burn.

Most atomisers produce a range of drop diameters, varying from a few micrometres to around 100-1000µm, or even more. A 100µm particle burns in about half a second in an industrial flame while a 500µm particle takes about five times as long and a 1000µm particle 10 times as long. Since residence time in a flame is typically one second or less, it follows that drops larger than about 200µm will not fully burn out before the end of the flame and will either drop on the product, unburnt, or end up in the dust collector.

For optimum performance, a range of drop sizes is ideal, fine drops to facilitate ignition and flame establishment and larger drops to maintain a controlled burning rate. However, for the above reasons, the maximum drop size must be below 100-250µm, depending on the exact application.

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

Most burners vary flow rate to operate over a range of heat liberation rates: atomiser performance must be satisfactory over the entire operating range, since cement plants do not always operate at full load. The drop size of many types of atomiser increases rapidly as the fuel flow-rate is turned down and this can present special problems for plant operation: turndown performance is an important selection criterion for atomisers.

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 with a high CV, and very low volatile and moisture content, to lignites with moisture and volatile contents of up to 60 per cent. 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 plant performance and on maintenance requirements. Relevant properties include:

- **Volatile content** – The higher the volatile content, the more rapidly the coal ignites and burns. Highly-volatile coals (above 35 per cent) tend to present significantly higher explosion risks than those with a volatility below 25 per cent. Coals with volatile contents above 45 per cent require special precautions.
- **Swelling properties** – Once the volatiles have been driven off, a coke particle is left. If this is larger than the original particle, it has a more open pore structure and will burn more rapidly than if it shrinks.
- **Moisture content** – Coals have two types of moisture: surface moisture and inherent moisture. Generally the higher the inherent water, the greater the coal reactivity and the higher the consequential fire and explosion risk. For pulverised coal firing, surface moisture has to be removed during 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 fire or explosion.
- **Ash content** – Cement manufacture demands consistent levels of ash quantity and composition, and coals have to be selected accordingly.
- **Hardness and abrasion indices** – Coal hardness affects coal mill capacity: the harder the coal, the less can be ground and/or the coarser the resulting pulverised coal. The abrasion index is mainly dependent on ash characteristics. Very abrasive coals with high silica ashes cause high wear rates in grinding elements.

Petroleum coke

Petroleum coke (petcoke), a by-product from oil refining, is the solid residue remaining after extraction of all valuable liquid and gaseous components from crude oil. The volatile content range is typically 5-15 per cent, depending on the coking process. The main difficulty in burning petcoke is its low reactivity due to this low volatile content. Petcoke has certain advantages, particularly its very high CV of around 8000kcal/kg (gross), but increasing price in certain market conditions sometimes reduces its attraction. The usually-high sulphur content (3-6 per cent) can also limit usage (Batra et al, 2005): its use has been banned in some Indian states (Jethmalani, 2017).

It exists in four basic forms, delayed ('green'), calcined, fluid coke and flexicoke. Delayed coke is by far the most common and 'green delayed coke' typically has 8-16 per cent volatiles, though higher temperature processing can yield less than one per cent volatiles. It may be 'sponge' or 'shot' and is used – with care – as up to 100 per cent of total kiln fuel.

Petcoke burning usually involves finer grinding than coal and higher excess oxygen to compensate for its low reactivity and achieve complete combustion, which can result in some de-rating of the kiln (Roy, 2001). High-momentum burner design and attention to calciner design can also assist.

4.3 Physics of combustion

No chemical reactions can take place until the oxygen in the air is brought into contact with the fuel.

Therefore, all combustion processes take place in the following stages:

mixing → ignition → chemical reaction → dispersal of products

Combustion rate depends on the slowest of these stages. In most industrial combustion systems, 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.

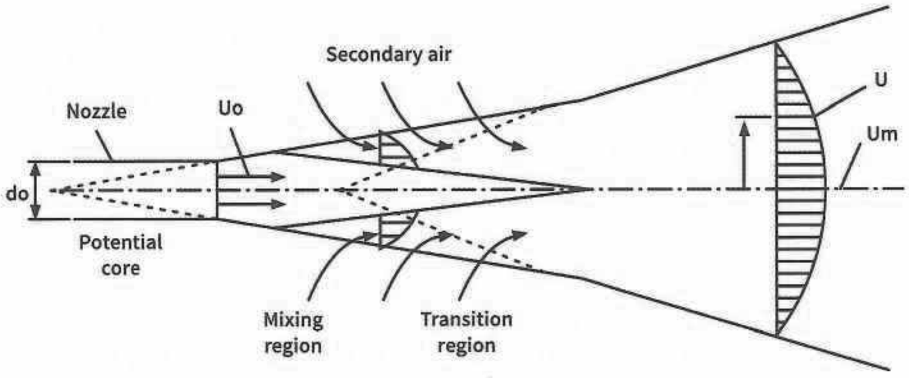
Insufficient fuel/air mixing produces unburned CO in flue gases, wasting fuel energy potential. For good combustion, adequate air must be supplied for complete mixing and the burner must be designed to mix the fuel and air streams effectively and efficiently. Hence, the saying of combustion engineers: 'If it's mixed, it's burnt.'

Fuel/air mixing

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

Figure 4.1 shows a free jet issuing from a nozzle in an ambient medium. Friction occurs between the jet boundary and its surroundings, locally accelerating the surrounding fluid to the jet velocity. The accelerated air is then pulled into the jet, expanding it. This process is momentum controlled and continues until the jet velocity is the same as that of its surroundings. The greater the jet momentum, the more surrounding fluid is entrained. A free jet that is able to expand unimpeded can entrain as much of its surrounding medium as it needs to satisfy its entrainment capacity.

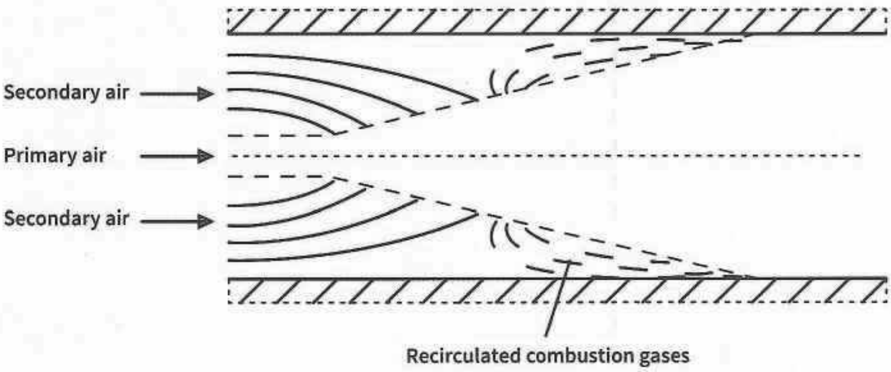
Figure 4.1 Entrainment of secondary air into a free jet



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

If the confined jet has momentum in excess of that required for complete entrainment of the secondary stream, jet recirculation occurs. The secondary air stream is initially pulled into the jet as described above, but a point is reached when all the available air has been entrained. At this stage, the jet pulls back exhaust gases from further up the kiln and draws them into the flame to overcome this excess momentum. This phenomenon, known as external recirculation, is illustrated in Figure 4.2.

Figure 4.2 Idealised recirculation



The role of primary air

Primary air has two major roles in burners:

1. controlling the rate of fuel/air mixing.
2. assisting with flame stability.

The effect of primary air on fuel/air mixing

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

The presence or absence of recirculation has a great effect on flame characteristics. 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. In this case, significant levels of carbon monoxide are normally produced.

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 make contact with refractory bricks, tending to wash them away and causing subsequent failure. The recirculating gases from a flame with high momentum ratio, however, provide a 'cushion' of cooler neutral gas which prevents impingement.

A high-momentum recirculatory burner jet will also produce a more responsive and stable flame that is more controllable, making plant operation easier. The characteristics of kiln flames with and without external recirculation are summarised in Table 4.3.

Table 4.3 Characteristics of flames with and without external recirculation

	Flame with recirculation	Flame without recirculation
Fuel/air mixing	Good	Poor
Reducing/oxidising conditions	Oxidising conditions exist throughout the flame. Oxidising conditions exist elsewhere	Strongly reducing conditions occur in fuel-rich parts of the flame
Flame impingement	None. Recirculating gases protect bricks and clinker from flame impingement	Flame impingement ocurs 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 per cent	CO produced at levels of oxygen as high as 2-4 per cent
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

Secondary air aerodynamics

Since secondary air has to be entrained in the fuel/primary air jet, its aerodynamics can have a huge effect on fuel/air mixing, and its flow patterns are considerably determined by cooler uptake and hood system design – or, for planetary coolers, by the satellite throats. To obtain the optimum potential performance from any kiln, it is absolutely essential that its aerodynamic characteristics are taken fully into account when designing the burner. For complete combustion and uniform heat transfer, an even distribution of fuel throughout the cross-section of a kiln is essential. Modelling techniques are an important tool for process optimisation. Extensive tests of kiln aerodynamics have been conducted by using water/bead model tests, computational fluid dynamics (CFD) and full size investigations. One example of the aerodynamics for a grate cooler kiln is illustrated in Figures 4.3 and 4.4, where asymmetry in the airflow pattern is clearly evident.

Figure 4.3 Typical aerodynamics from a grate cooler

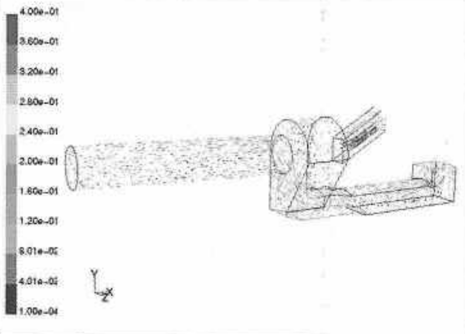
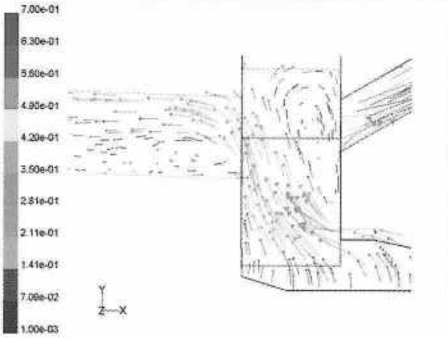


Figure 4.4 Close-up of aerodynamics in the burning zone



Burner design cannot overcome certain airflow patterns and modification to the equipment geometry is necessary. For example, in Figures 4.3. and 4.4 the tertiary air off-take on the rear of the hood has been relocated to eliminate poor aerodynamics.

However, in some cases, the solution may be as simple as changing the location of the burner tip as shown in Figures 4.5 and 4.6.

Figure 4.5 Physical model of secondary air pushing the flame to the top of the kiln

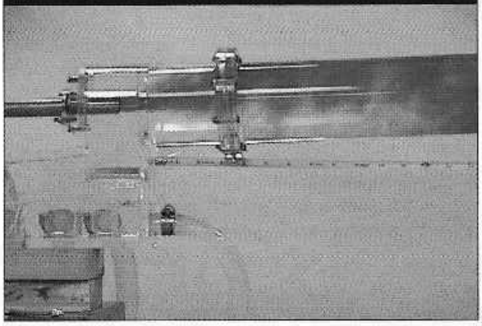
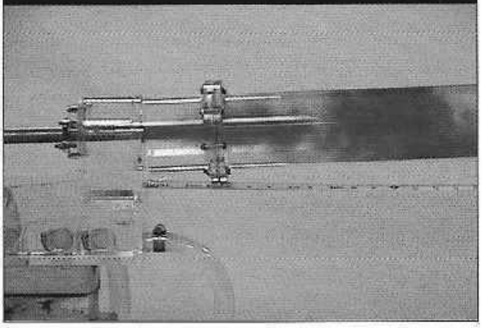


Figure 4.6 Same system as Figure 4.5 after inserting burner further into the kiln



Similar effects are observed with riser ducts where a sharp angle of entry from the kiln can give a highly asymmetric airflow, producing poor fuel/air mixing and an intense recirculation zone on one side, while in flash calciners the velocity profile of gases exiting the kiln can dominate the distribution of particles injected to the riser.

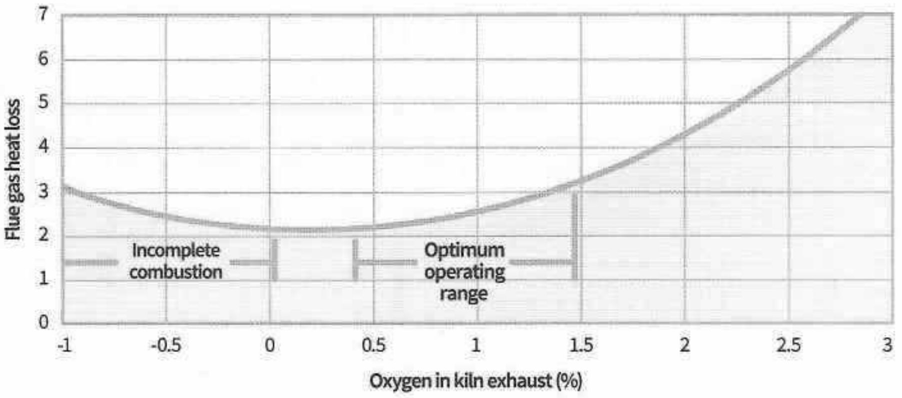
Effect of excess air on fuel consumption

Although the effect of excess air level on overall thermal efficiency has been understood for many years, it is often surprising how little attention it receives. As the oxygen level increases, more excess oxygen and nitrogen passes through the system and requires heating firstly to flame temperature and ultimately to exhaust gas temperature.

In cement plants increased airflow through the cooler reduces secondary air temperature, and therefore, the flame temperature, thus requiring even more fuel to heat the charge to the required process temperature and the increase in fuel consumption is much greater than that needed just to heat the excess air to back-end temperature. Improved cooler designs and control tend to avoid such effects.

If the excess air level in a flame is reduced below a certain level, incomplete carbon combustion produces carbon monoxide and fuel consumption increases (see Figure 4.7). The better the fuel/air mixing, the lower the excess air level at which these emissions occur.

Figure 4.7 Effect of kiln oxygen on flue gas heat loss

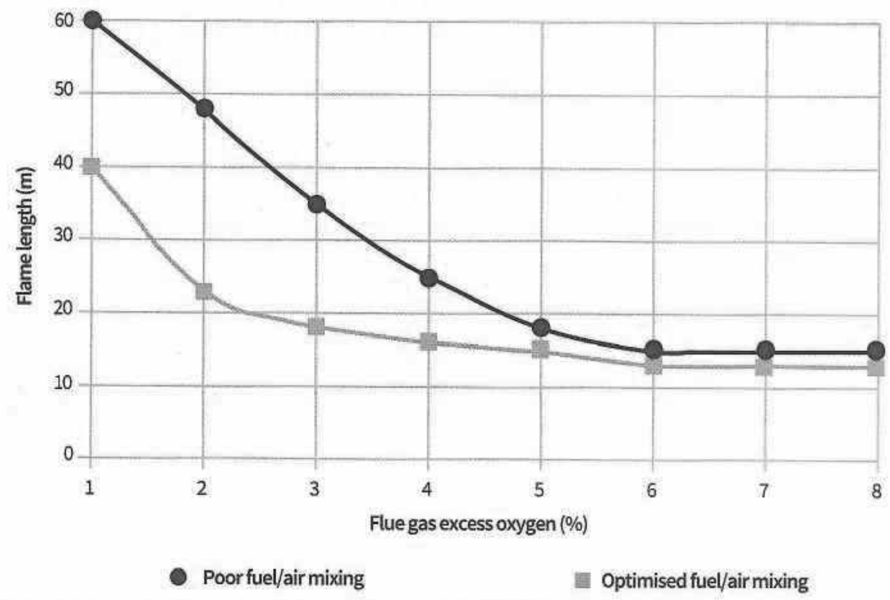


Excess air also has a dramatic effect on flame length and on the heat profile in the kiln. Many operators tend to believe that flame length increases as the draught from the ID fan increases. The opposite is true. Figure 4.8 shows a typical relationship between flame length and excess air for an optimised kiln with good fuel/air mixing and for one with poor fuel/air mixing. A shorter flame, other things being equal, produces a more reactive clinker.

Two important characteristics of Figure 4.8 are:

1. flame length dependence upon excess air levels
2. responsiveness of the flame to excess air levels.

Figure 4.8 Flame length versus excess air



The optimised flame can have a length of 30m at 1.5 per cent excess oxygen, whereas the poor flame has the same length at 3.5 per cent excess oxygen. In addition, the responsiveness of the optimised flame allows the operator to fine-tune flame length with minor adjustments to the excess air level, whereas the poor flame requires a much broader range of adjustments.

4.4 Burner design

The flame in a rotary cement kiln, riser or calciner is for the most part produced by a turbulent diffusion jet. Scientists have paid far less attention to diffusion flames than to premixed flames, despite the fact that most 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, and even the mixture strength has no clear meaning.

When any jet mixes into its surroundings, steep concentration gradients are set up near the orifice from which it emerges. 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 is there any approximation to homogeneity.

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

4.5 Cement 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 under the control of the burner designer. Most of the air comes from the clinker cooler and the aerodynamics of that flow depends upon other factors.

Apart from precessing jet burners (p49), all kiln and calciner burners are jet entrainment burners. Off-the-shelf rotary kiln burners are often simply nozzles with cross-sectional area determined on the basis of either the kinetic energy of the primary air flow or its momentum flux. These approaches do not account for the fact that fuel/air mixing is strongly affected by entrainment of secondary air and by the flame confinement within the kiln. These design methods usually result in a very high primary air velocity (>300m/s), employing 5-10 per cent of the stoichiometric air requirement. It is preferable to base designs upon the phenomenon of jet entrainment (see Figure 4.1). This approach determines the degree of external recirculation as the burner fuel jet mixes with the secondary air, as shown in Figure 4.9. The cross-sectional area of the burner nozzle or nozzles is derived from relatively complex calculations, generally related to:

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

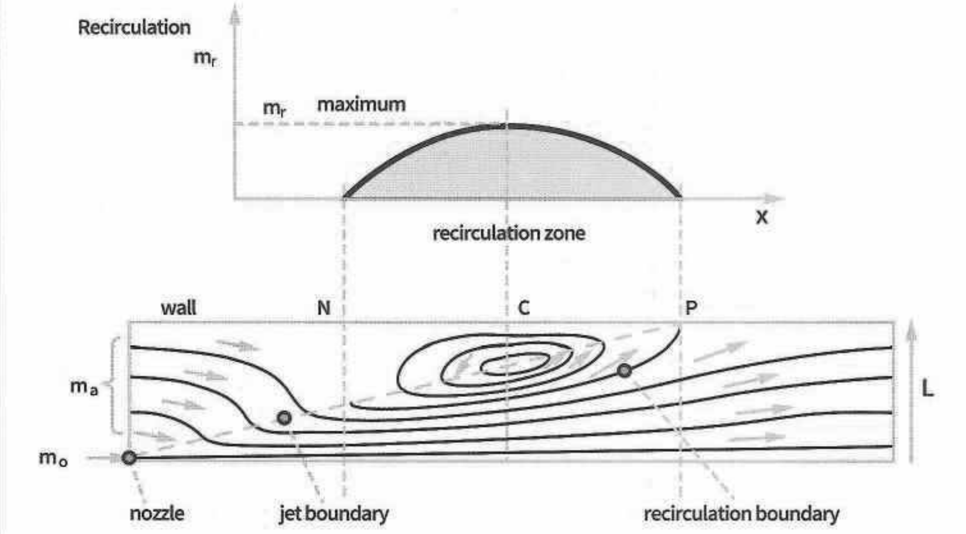
where: m_e = mass flow of entrained secondary air

m_o = mass flow-rate of fuel and primary air through the burner

m_a = mass flow of secondary air

In comparison with the simpler and unsound methods, jet entrainment designs usually require more mass flow of primary air at a lower velocity to provide enough momentum for external recirculation.

Figure 4.9 Mixing and recirculation downstream of a confined jet



The mass flow and velocity of primary air are 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 CO emission is held constant. In practice, if the burner momentum is insufficient to effectively mix fuel with secondary air, the heat consumption increases by two per cent for every one per cent increase in excess oxygen. This is one of the main reasons why NO_x emissions are reduced with low primary air burners. Hence, when designing a kiln burner there are competing forces between minimising the amount of primary air and excess air that must be taken into consideration.

Flame stability

A stable flame has a constant point of ignition very close to the burner nozzle, whereas 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 the flame extinguishing and, in any case, the substantial quantities of unburned fuel between the 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. Stabilisation of oil flames requires proper atomisation plus some means of local recirculation. Stabilisation of pulverised fuel flames (eg, coal/coke) is affected by fineness, ash properties, volatile content and conveying velocity. Flames produced by coal nozzle velocities in excess of 80m/s are susceptible to severe instabilities. Despite all these potential hazards, few kiln burners have adequate means of ensuring good flame stability. For gas the most effective technique is to form an internal recirculation zone just in front of the nozzle; burning gas is carried back from further down the flame and constantly ignites the incoming fuel, anchoring it to the nozzle.

The internal recirculation zone can be achieved by a number of methods: either a bluff body or swirl on the fuel, the primary air, or both.

Bluff bodies suffer overheating from the flame and tend to be unreliable over long periods. Swirl on the injected fuel 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. This can have adverse side effects on the overall flame characteristics such as impingement on the refractory.

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

Except on some relatively primitive burners, still available in some kilns, reradiation from the hot walls should rarely be used as the primary means of flame stabilisation: more positive methods are preferred.

Calciner burners

Combustion conditions in the calciner are very different to those in the kiln, involving lower temperatures and oxygen concentrations, which inhibit combustion. A retention time of at least 3s is required, rising to over 5s for fuels that are difficult to combust completely. Designs must also achieve eradication of toxic residues and keep NO_x emission to low levels (Schürmann and Streit, 2018).

Many original calciner burners were simply open-ended pipes, burning fuel in intimate contact with the feed. Flame stability is not normally an issue, since the incoming combustion air is usually preheated to above the gas ignition temperature. Hence sophisticated swirl and bluff body devices are rarely necessary. 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 1000ppm and sometimes up to several per cent.

More sophisticated calciner burners are scaled-down kiln burners, and like kiln burners, should be matched to calciner aerodynamics to optimise performance. Some calciner aerodynamic flow patterns give serious combustion problems because of poor airflow and mixing (as discussed below for AFs). In these cases the burner alone cannot ensure optimum fuel/air mixing: airflow patterns must also be improved.

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 across a wide range of flows requires some means of stabilisation, as discussed above.

One interesting design uses the patented Precessing Jet (PJ) nozzle developed at the University of Adelaide, Australia. In combination with other jet flows, this provides a high-radiation, low-NO_x flame tailored for a given application and its initial installations were mainly in gas-fired rotary kilns. 'Precessing' refers to a gyroscopic motion which creates a flow field with a stirring motion, so that combustion occurs in large fuel-rich structures within the flame envelope. This results in cracking of hydrocarbons to produce intermediate soot particles, yielding a highly radiant low temperature flame that promotes high heat transfer with low NO_x formation. It is marketed by FCT-ACTech as the Gyro-Therm burner (Wilson et al, 2007).

Oil burners

Oil is an excellent kiln fuel as its high-emissivity flames provide high rates of heat transfer to the charge. However, to burn oil efficiently it must be atomised and sprayed into the kiln in a controlled manner, as previously discussed.

Rotary kiln oil burners are similar to gas burners, with an oil sprayer replacing the gas gun. Some primary air is always used. Common atomisers 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 reradiation from hot refractory to achieve stability, but generally, aerodynamic swirl flame stabilisation assures excellent flame stability.

Because the cement kiln requires a precisely-controlled heat-up, operators should use a high-performance twin-fluid atomiser with a wide fuel turndown (8:1).

Calciner oil burners

Oil burners for flash calciners vary from open-ended pipes spewing oil into the vessel to sophisticated burners employing twin-fluid atomisers.

The open-ended pipes tend to produce large drop sizes (over 1000µm) that cannot burn out during their residence time in the vessel. Such pipes may have been satisfactory for the lighter fuels of the 1960s, but they are totally unsuitable for modern high-asphaltene oils. High performance, internal mixing, twin-fluid atomisers are essential to burn all the fuel within the vessel. As with gas burners, oil burners should be designed and optimised for their location using modelling.

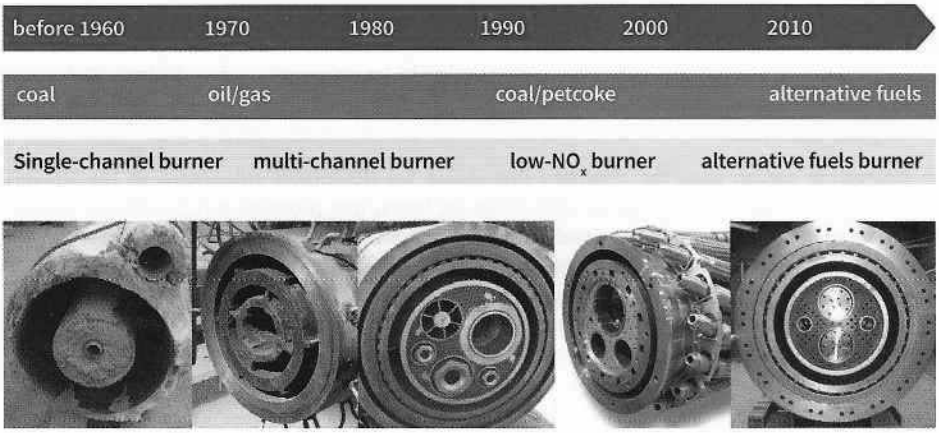
Coal burners

Where ash deposition is acceptable, coal is the best rotary kiln fuel owing to its very high-emissivity flames, which provide high rates of heat transfer to the charge. Generally, the lower cost of coal gives it a significant advantage, though it is rather more difficult to handle than liquids or gases, being a solid of varying composition and calorific value. Regardless of burner design, coal must be dried and ground before injection to the kiln. As for oil and gas, a burner is a critical kiln component and the variable nature of pulverised coal requires a flexibility of design to allow the use of differing grades of fuel.

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

With the conversion of many kilns from oil- to coal firing in the 1970s, new burners were developed from oil burner designs. These generally used less primary air than open-ended pipes and were suited to use with indirect firing systems. However, in many cases, performance was very poor due to inadequate fuel/air mixing caused by low jet momentum. Many of these burners exhibit sophistication for its own sake, gaining no benefits from the extra cost. However, benefits can result from using multi-channel burners, especially with more difficult low-volatile fuels such as petcoke, provided the burner is matched to the kiln.

Figure 4.10 Burner technology and its development



Source: ECRA, 2010, Unitherm CemCom and Fives PIllard

Dual and multi-fuel burners

Dual and multi-fuel burners combine the essential features of single-fuel burners previously described and provide flexibility in fuel choice, a major advantage when unstable world fuel prices make any medium-term prediction of costs very difficult. A true multi-fuel installation allows the plant to choose the most economical fuel currently available. In many cases by-product fuels may be used in place of, or in addition to, the primary fuel. Vering and Hermann of Venti-Oelde (2006) outline the repercussions upon firing fan design now that pressures of 25-40kPa are required.

4.6 Heat transfer

Fuels are burned to heat a product and the mechanism of heat transfer from the flame is vitally important. This is a very complex subject worthy of several books and is covered here only in sufficient detail to allow a reasonable understanding. Radiation falling on a surface may be reflected, transmitted, or absorbed. The fraction of energy that is absorbed is manifest as heat. If there is a temperature difference (ie, a driving force) between two parts of a system, heat will be transferred by one or more of three mechanisms: conduction, convection and radiation.

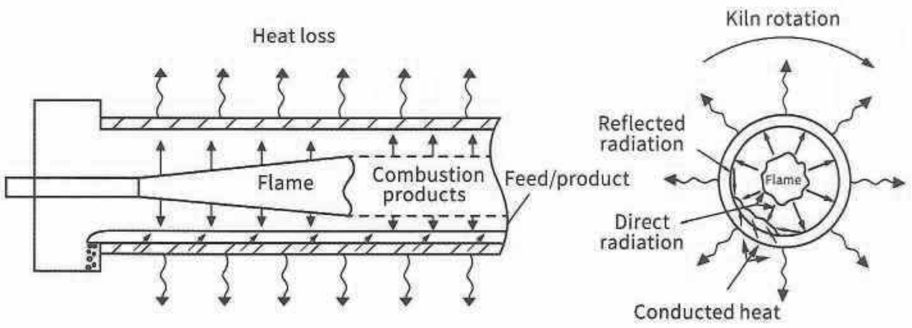
Radiation dominates in cement kilns, transferring over 95 per cent of the heat in the burning zone. It can be seen from Figure 4.11 that the process is more complex than the basic equation would suggest. The factors which affect the rate of radiative heat exchange are the temperatures, emissivity and relative geometry of the flame and surroundings.

- Basic equation for radiative heat transfer:

$Q = \sigma\varepsilon A\,(TF^4 - Tr^4)$

where: Q = the heat transferred (J/s)
σ = Stefan-Boltzmann constant - 5.6697×10^{-8} W/m^{2}K^4
ε = emissivity
A = surface area (m^2)
TF = flame temperature (K)
Tr = product temperature (K)

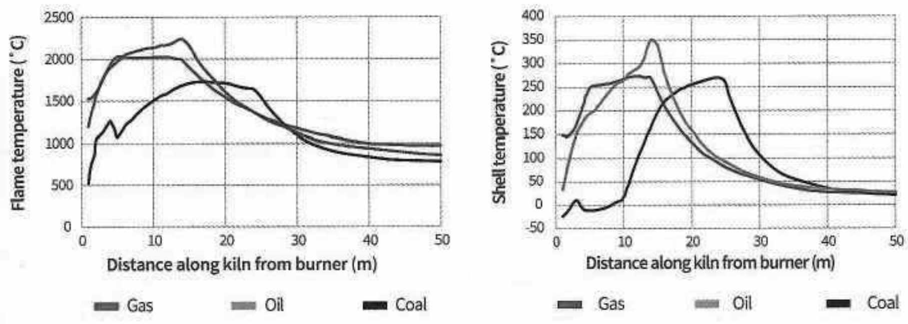
Figure 4.11 Heat transfer paths in the rotary kiln



Effect of fuel type on heat transfer

The most observable difference between gas, oil and coal flames is the brightness or emissivity, which is ~0.3 for a gas flame, ~0.5 for oil and ~0.85 for coal. Emissivity varies along the length of the flame as shown in Figure 4.12.

Figure 4.12 Heat transfer rates from different fuel types

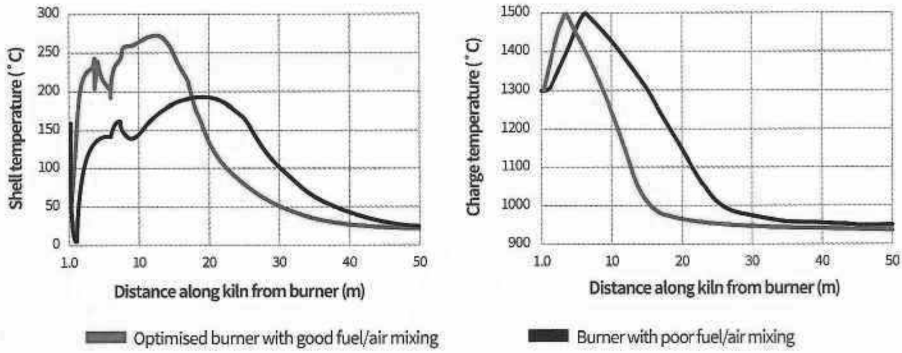


The higher emissivity of the oil and coal flames result in higher heat transfer rates in the near flame region and a 'peakier' heat flux profile.

Effect of burner design on heat transfer

Burner aerodynamics also significantly modify flame shape and the heat transfer profile as shown in Figure 4.13. A long, flat heat flux profile equates to a long burning zone. This is usually detrimental to clinker quality, giving rise to larger alite and belite crystals. An optimised heat flux can be achieved by adjusting the burner settings.

Figure 4.13 Heat transfer rates from different burner designs



The optimised heat flux was produced by a recirculatory flame, produced by optimising the kiln aerodynamics and tailoring the burner momentum relative to the secondary air momentum through physical and mathematical modelling.

4.7 Pollutant formation

The origins of some gaseous pollutants formed during combustion are briefly discussed here, while Chapter 9 provides the main discussion.

NO_x formation

The two main formation mechanisms are 'thermal NO_x' and 'fuel NO_x'. Thermal NO_x involves the combination of atmospheric nitrogen and oxygen. The high activation energy of this reaction demands very high temperatures and thermal NO_x formation is hence temperature sensitive. Owing to the very high flame temperatures, ie above 2000 °C (3600 °F), thermal NO_x generally dominates in kilns. (No fuel NO_x is produced in gas flames, though emissions usually remain important because flame temperatures are higher.) The reaction takes place between oxygen radicals, nitrogen radicals and molecular nitrogen in the 'Zeldovich reaction couple'. Apart from temperature, in-flame oxygen concentration and residence time in high temperature zones influence the final level of thermal NO_x generation.

Most fuels contain some nitrogen in organic compounds. During combustion, this is converted into a range of cyanide and amine species that are subsequently oxidised to 'fuel NO_x', depending on local oxygen availability. This is a complex mechanism, important in calciners, and is less dependent on temperature.

(A third mechanism – known as 'prompt NO_x' – has been identified by some workers, involving rapid fixation of atmospheric nitrogen by hydrocarbon fragments in fuel-rich areas in low-temperature regions of the flame, but it makes a minimal contribution to emissions from coal-fired cement kilns.)

Carbon monoxide formation

Carbon monoxide (CO) is highly toxic and results from 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, before being oxidised.

However, in good combustion systems CO should normally be limited to a range of 20-50ppm or less. Slightly higher levels may be tolerated in specific circumstances, but levels of 1000ppm or more usually result from poor fuel/air mixing or air starvation (high fuel/air ratio) and are symptoms of serious combustion problems. In addition to its toxicity, there is also a heat penalty when CO is emitted, as outlined earlier (see Section 4.1 and Figure 4.7).

Unfortunately, CO tends to increase when NO_x reduction techniques are applied (see Chapter 9), since these generally involve delays in fuel/air mixing.

Spasmodic injection of fuel is intuitively offensive but is sometimes driven by cost considerations as when burning whole used tyres or other lumpy fuels. Inevitably the effect is either derating of the kiln to ensure adequate oxygen, or pulses of CO as peaks of combustion exceed available oxygen: neither is desirable.

Sulphur compounds

Sulphur (S) compounds are generated when sulphur-bearing fuel (oil, coal, etc) is burned or where sulphur-bearing materials are processed. Sulphur dioxide (SO_2) is the most common compound, and is acidic and corrosive. It causes major problems with corrosion in cooler plant areas and in 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 and leads to sulphuric acid formation with water from combustion at the so-called acid dewpoint, leading to severe duct corrosion.

Hydrogen sulphide (H_2S), rarely encountered in kiln exhausts, is extremely toxic and has a nauseating smell. It is formed when sulphur-bearing fuels burn under reducing conditions, or is driven off from sulphur-bearing feeds, such as some clays.

Mullinger and Jenkins (2008) provide further details on these and other related matters.

4.8 Modelling

Mullinger and Jenkins provide Table 4.4 which compares modelling methods. A detailed discussion is beyond the scope of this book, but all methods have found success in cement plants, when applied and interpreted sensibly.

Table 4.4 Comparative merits of modelling methods

Method	Scaling	Turbulence	Heat transfer	Mass transfer	Momentum
Physical	Distortion parameters used	Inherently correct	Isothermal – post corrected	Correct but see heat transfer, no sub-chemistry	Correct but see heat transfer
Mathematical	Coarse grids give 'stepped' answers	'Crude' mixing length models used	Correct	Correct at macro-chemistry sale. Sub-chemistry limited by turbulent model	Relies on other model predictions
CFD	Correct if fine grids are used	Selection of suitable model is critical	Correct	As good as available sub-chemistry data	Correct but see turbulent model

Akritopoulos and Abbas (2013) discuss the methods of one company, CINAR. They suggest that most modern high-momentum kiln burners can avoid the traditional problems of CO generation due to either detached flames (also leading to a long burning zone and high NO_x levels) or to inadequate secondary air diffusion and subsequent delayed fuel burn-out and/or coarse fuel drop-out. There remain problems arising from secondary air flow disruption due to kiln hood asymmetry and tertiary air duct off-take. Calciner aerodynamics are more complex. There are no reverse flow flame regions, and the co-flowing streams of fuel and secondary air are replaced by streams of fuel and meal injected at an angle to the walls and immediately entrained by air flow from the riser and/or tertiary air duct. These locations are difficult to probe or model, so a CFD approach (with validation) is more appropriate. The key design criteria are to achieve sufficient residence time and avoid production of hotspots, via positioning of burners and meal injection points. The latest CFD models compute chemical reactions (eg, LoI development and NO_x generation) as well as flow patterns and temperatures.

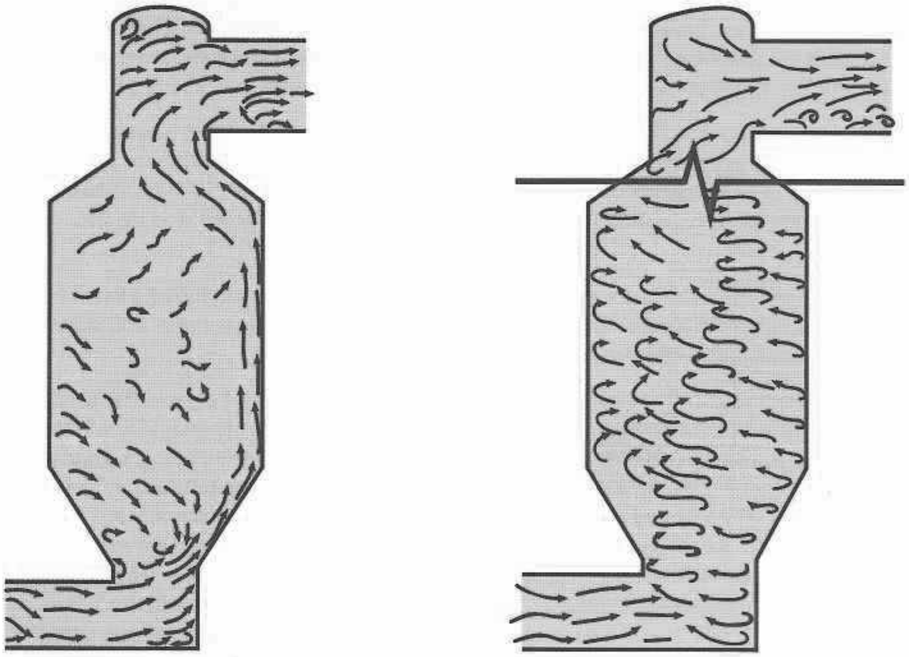
More detailed discussion on modelling is provided in the book by Mullinger and Jenkins, and methods have continued to develop: Lowes et al (2018) present a range of case studies, with particular reference to eliminating stratification and enhancing fuel burn-out.

Possible optimisations with the aid of CFD simulations include:

- calculation of the optimum fineness of fuels with regard to rate of descent and flight properties
- determination of the optimum feed location for various fuels with regard to retention time and burn-out
- optimisation of calciner design with regard to gas stratification (streaming) and improved mixing.

Figure 4.14 illustrates how calciner airflow patterns can become quite complex with changes in cross-section producing effects such as strong recirculation zones and streaming aerodynamics with the tertiary air flow dominating due to its mass and comparatively-high velocity at the inlet (Manias, 2011).

Figure 4.14 Typical aerodynamic problems encountered in calciner operation that impair calciner combustion



Source: Innovations in Portland Cement Manufacturing, 2nd Edition

4.9 Fuel storage and firing systems in practice

Whatever fuel is used, appropriate and robust measures need to be in place for placing sound supply contracts to ensure sustained delivery of good quality material and appropriate equipment for its storage, preparation, controlled metering and dosing.

Gas

As mentioned earlier, gas flames are the least efficient, requiring more combustion air per unit of heat. Kiln production typically increases by 2-3 per cent when coal replaces gas. 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 costs.

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

Oil

Oil is injected with a nozzle pressure of about 20kg/cm^2, except for pressure atomised systems, which employ pressure to 100kg/cm^2.

Coal

Coal varies in composition much more than oil or gas. Major suppliers working to an established supply specification should minimise short-term fluctuations in deliveries, while long-term variation can be compensated by analysis and normal kiln control procedures. However, if the supply is from small-scale or multiple suppliers – often a false economy – adequate blending is essential prior to use. Procurement staff require a sound understanding of the repercussions of quality variations (in composition, CV, moisture content, grindability, ash content or ash analysis) upon both the production process and product quality. Lieberwirth (2012) discusses a handling system for coal delivered by trucks from up to eight different mines and requiring a high degree of homogenisation.

Coal stockpiling requires vigilance as spontaneous combustion is common, particularly with wet, low-rank or pyrite-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.

Bituminous and anthracitic coals are usually dried and ground so that either the 90µm or 75µm residue is less than half the percentage volatile content and that at 200µm is below two per cent. Conveying air at a pressure of 120-150g/cm^2 assures injection to the kiln with a burner tip velocity of 60-80m/s to achieve adequate flame momentum, which some suggest should be targeted at >11N/MW.

Petcoke

Physically, petcoke can be milled with coal, but care is needed to avoid producing a twin particle size distribution, as can also happen for two markedly different coals. This produces a disadvantageous double combustion pattern due to differential burn-out.

As a general rule of thumb, oxygen concentration ex-kiln for a 100 per cent petcoke flame needs to be 1-1.5 per cent higher than for a high-volatile bituminous coal when both are ground to a 90µm residue equal to half the percentage volatile content. However, for many 20th century burners, grinding to 5-10 per cent >90µm does not significantly improve combustion rate. The '90µm residue rule' usually works for common grades of coal because at lower volatile contents, finer grinding compensates for the lower specific surface generated on devolatilisation. With petcoke the solid carbon reaction with oxygen in the boundary layer of the flame becomes more important than those involving volatile hydrocarbons and the hydroxyl radicals produced during their stoichiometric combustion. The maximum 200µm residue for coke may need to be one per cent to reduce the possibility of coarse char particles depositing on the kiln charge and creating local reducing conditions for calcium sulphate or ferric iron. Greco (2007) suggests that the 90µm residue for petcoke should be half-the- volatile-content-minus-one as a starting point. A fuel mix of 100 per cent petcoke can be successfully fired by adopting a pragmatic approach to fineness, seeking first to ensure that sulphate cycles can be controlled by adjusting the oxygen level, ex-kiln, as discussed in Chapter 5.

Solid fuel preparation and use

Coal firing may be direct, ie feeding coal directly to the burner with all of the drying/carrying air as primary air (typically 15-30 per cent of total combustion air) or indirect (see Figure 4.15 on next page). But the modern norm is indirect firing, with intermediate storage of ground coal, and separate cleaning and venting of the drying/ carrying air (see Figure 4.15). Several variations are possible.

It is commonly assumed that indirect firing yields higher thermal efficiency by reducing primary air and excluding water vapour from coal drying. This may be invalid due to the poor fuel/air mixing of low primary air burners, while water vapour in the flame has a catalytic effect on combustion. The ability of a single mill to supply two or more burners is of greater importance. Note that venting an indirect milling system may lose significant fuel volatiles, and hence heat content (up to 280kcal/kg).

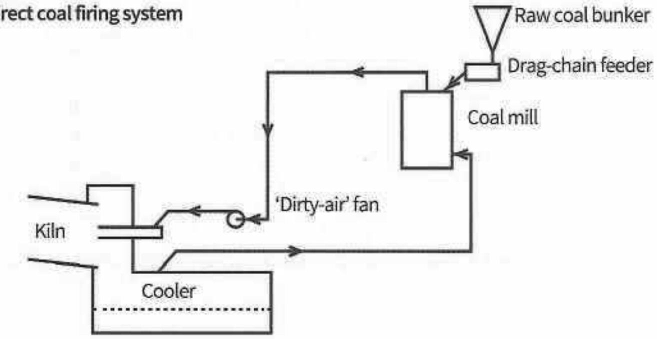
In 2018 FCT Combustion announced the availability of a new, relatively low-cost, upgrade design for direct firing systems, the "Direct-Flex™". Excess air is split from the primary and conveying air supplied to the burner, and is injected at low velocity to either the cooler or to the kiln hood, providing improved ability to fire AFs and control NO_x generation, along with reductions in ring formation, stable coating in the burning zone and more stable kiln operation.

Vertical roller and pendulum mills probably predominate for coal grinding, though ball mills may be preferred to cope with abrasive ash. Curiously, the Claudius Peters EM mill (or "Babcock E-mill"), widely used in the power industry and comprising large steel balls compressed between fixed and rotating grooved rings, is not common in cement (Floter and Thiel, 1992).

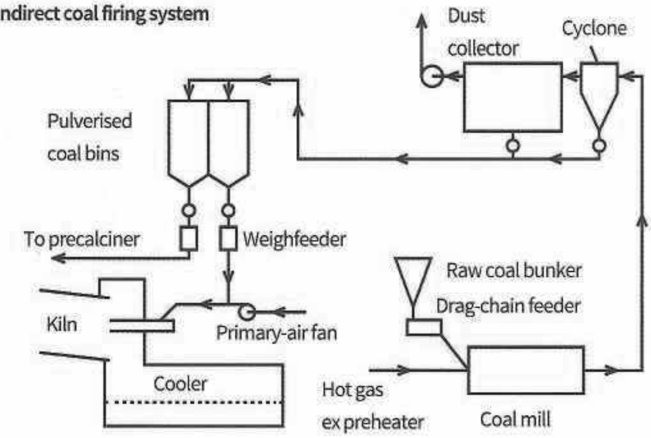
Generally roller mills have integral static classifiers, though dynamic classifiers may be employed. Roll separation from the grinding table should be maintained at 5-10mm and coal feed size should be 100 per cent <25mm with approximately 30 per cent >10mm. Rock and metal rejects fall into the hot air plenum and are swept away by a rotating scraper through an air-locked discharge chute. Abnormal spillage (ie, more than two per cent of feed) may be due either to roll clearance exceeding 15mm or to excessive clearance between table and louvre ring: the required 25m/s air velocity through the louvre vanes cannot be maintained for clearance above about 10mm.

Figure 4.15 Coal firing systems

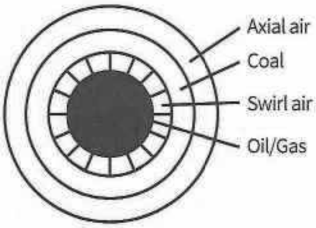
Direct coal firing system



Indirect coal firing system



Three-channel burner



Roller mills can dry coal up to 10 per cent moisture, beyond which the mill is de-rated according to manufacturers' data. Similarly, mills are normally designed for 55 Hardgrove Index and harder coals (lower HGI) result in de-rating. Finally, when sizing coal mills, a 10 per cent fall in capacity between maintenance stops must be expected. Mills with common table and fan drives may be given separate drives, and capacity can often then be increased by raising table speed. Bastue (2007) indicates the demands of different coal types and describes how FLSmidth's ATOX mill is arranged and operated for solid fuel grinding.

Young et al (2007) provide detailed guidance on the design and operation of systems for solid fuel handling, preparation and firing, enabling safe operating values to be used for parameters such as mill inlet and discharge temperatures, fuel moisture content and transport velocity.

Vendors specify a minimum airflow, typically 1-1.5kg (0.8-1.2Nm^3)/ kg coal, which must be maintained even for negligible drying requirements, to ensure that coal does not exceed its ignition temperature for long enough for auto-ignition. Fires usually result from 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), and extinction is effected either by water injection or, better, CO_2 or perhaps N_2.

Hot air for drying coal can come from cooler exhaust (normal air) or preheater exhaust (with the benefit of low oxygen content). Mill inlet temperature is controlled to maintain the outlet temperature as previously-described and the product is initially collected by a cyclone. Tempering to about 370°C is effected by bleeding in cold air between cooler and cyclone. If preheater exhaust gas is used, it will typically be at 300-350°C with five per cent O_2 and six per cent moisture. Temperature and moisture levels must be considered in system design. Although most mill fires occur at start-up or just after shutdown, the low-oxygen atmosphere reduces fire risk during normal operation.

In the United States, 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, equating 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). In the EU mill design and operation are covered by the 'ATEX' directives, 95 and 137 (Cowie, 2005).

Direct coal firing involves injection at a tip velocity of approximately 80m/s. Tip velocity must always be substantially greater than the flame propagation velocity which may be up to 25m/s. Flames produced at nozzle velocities above 80m/s are susceptible to severe instabilities. The pipe is usually narrowed near the tip to minimise parasitic pipe losses and convert the flow into the desired static pressure. For indirect firing, multi-channel burners of various designs are employed. One annulus is used for conveying pulverised fuel from the mill and one or more separate streams are used to supply primary air for controlling the flame (see Figure 4.14.). Typical specifications used by vendors for burners with indirect firing systems are shown in Table 4.5.

Table 4.5 Typical specifications for burners with indirect firing systems

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

Coal firing almost inevitably involves a normal operating condition where air contacts pulverised coal before it reaches the burner, presenting an inherent risk of fire and explosion. Therefore, system design is critical and should consider numerous factors including:

- minimum ignition temperature of air/fuel mix - coal ignition temperatures are in the 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: 12 per cent
- The entire pulverised coal system must be designed to contain $3.5kg/cm^3$ (NFPA 85F) or with explosion relief as appropriate (NFPA 68). VDI 3673 is also relevant.

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

It is liable to spontaneous ignition, with risk increasing with thickness of dust layer and the presence of easily oxidised contaminants such as pyrites (above two per cent). A smouldering fire can become explosive if disturbed.

Coal obviously can be handled safely, but pulverised coal should always be considered as a potential explosive. Carini and Hules (1986) review safety considerations.

Explosion venting is needed as part of explosion protection for PF silos and is often inadequately separately treated from fire prevention measures. One supplier taking a special interest in these matters is Thorwesten Vent GmbH (www.thorwesten.com). The correct functioning of all fire- and explosion-prevention equipment and measures should be monitored and regularly tested.

4.10 Insufflation

Insufflation is the injection of fine material to the kiln flame, usually dust, either metered into the primary air (using a venturi), or separately conveyed pneumatically and injected adjacent to the main burner. This:

- returns dust to the kiln with maximum chance of incorporation into clinker rather than re-entrainment with exhaust gas
- increases the luminosity, and hence the heat transfer, of oil and gas flames
- reduces flame temperature and, hence, thermal NO_x production
- increases volatilisation of alkalis in the dust to facilitate production of low-alkali clinker
- allows 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 is temporarily substituted for coal as kiln fuel, the effect of coal ash upon clinker chemistry can be maintained by injection of an appropriate correcting mix
- facilitates the addition of small quantities of hazardous waste which must be fed directly to the high-temperature zone of the kiln.

Material entering 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, emissivity may be reduced and fuel-air mixing deteriorate. Insufflation at a rate of more than five per cent relative to clinker will excessively reduce flame temperature. Insufflation of either water or $CaCO_3$ can reduce NO_x formation by reducing the maximum flame temperature (Schorcht et al, 2013).

4.11 Alternative and waste fuels

In recent decades, CO_2 reduction targets and high fuel costs – usually the largest single cost for a cement plant – have stimulated a search for alternatives: sometimes lower coal prices are gained by openly investigating a possible switch. Gäbel & Nachtwey (2001) review fossil fuel reserves and the future of alternative fuels (AFs). Other reviews include Stoppel (2004) for tyres, Angelo (2006) for animal meal and Brachthäuser (2003) for other wastes.

Table 4.6 Global thermal energy by fuel

Year	Fuel type share in total fuel mix (%)			
	Fossil and mixed wastes	Biomass	Fossil fuel	Total
1990	1.69	0.27	98.0	100
2000	4.43	0.79	94.8	100
2011	9.27	4.06	86.7	100
2016	11.07	5.60	83.3	100

Source: Cement Sustainability Initiative, 2018

Petcoke was the most commonly-used conventional fossil fuel in 2016, closely followed by coal (including anthracite and coal waste), whilst for biomass, the most common category was the combination of agricultural, organic, nappy (diaper) waste and charcoal, with wood, animal meal and dried sewage sludge making appreciable contributions. For fossil and mixed wastes, plastics were in the lead, well ahead of tyres and mixed industrial waste. The use of a variety of other wastes has been reported, including:

- **Liquid waste fuels**
  Asphalt slurry, chemical wastes, distillation residues, inks, oil sludge, oily water, paint waste (including thinners), petrochemical waste, tar, used oils, varnishes, waste solvents, wax suspensions.
- **Solid waste fuels**
  Animal meal, battery cases, colliery waste, domestic refuse, nut shells, oil-bearing soils and shale, packaging waste, paper waste, plastics, plastics residues, pulp sludge, refuse derived fuel, rice chaff, rubber residues, sawdust, sewage sludge, spent pot liners, used tyres, vehicle shredder residue, waste textiles, wood waste.
- **Gaseous waste fuels**
  Landfill gas, pyrolysis gas.

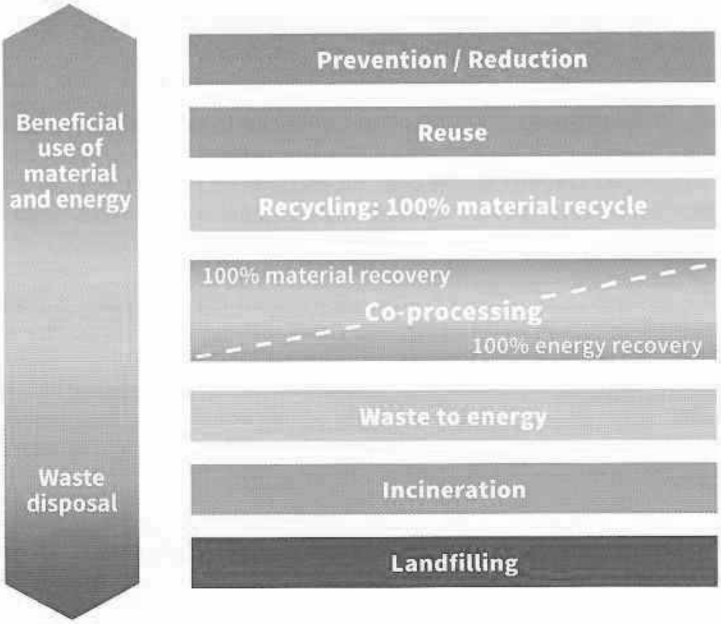
The original drivers for burning AFs in cement kilns were purely economic as the cost of fossil fuels to fire kilns rose in the 1970s following the two Middle East 'oil shocks'. Switching cement kilns to lower-cost fuels became an economic necessity. Prior to those oil shocks most cement kilns were fired with heavy fuel oil. The rise in the oil price following the oil shocks led to the widespread installation of equipment for grinding and drying coal in plants as cement companies sought to reduce the impact of rising thermal energy costs.

Such diversification of the fuels used for firing cement kilns has continued. As time passed the costs of coal rose and cement manufacturers kept searching for lower-cost fuels. In many parts of Europe, the USA, Latin America and India this resulted in kilns being switched from coal- to petcoke firing. However, this fuel brings its own challenges due to the low volatile content and high sulphur content of most petcoke types. Cement plants in parts of the Arabian Peninsula, Africa and Latin America are still in the process of switching from fuel oil to coal firing. Elsewhere, such as central Asia and parts of the Middle East the switch is being made to natural gas firing. Cement factories in Egypt went through the process of switching from natural gas to coal firing, along with introducing the burning of AFs, following the revolution in 2012 and the ensuing shortage of natural gas to fire cement kilns.

Besides this switching between fossil fuels such as fuel oil, coal and natural gas the imperative of reducing fuel costs has resulted in many different AFs being fired in kilns. The majority of these AFs are waste products from society or other industries. As experience of burning AFs grew, cement companies came to recognise the contribution they were making by providing a disposal service for the wastes generated by other industries and society. A major driver encouraging the burning of AFs was the introduction of environmental taxes on the landfilling of wastes in Europe. Such landfilling taxes create an incentive to find alternative methods for waste disposal and mean that cement companies can command a 'gate-fee' for accepting such wastes. These fees then allow the cement companies to invest in the equipment for receiving, storing, processing, dosing, transport and burning of increasing quantities of AFs and more-difficult-to-burn AFs in their kilns.

Therefore, using AFs is clearly a 'win-win-win' strategy. The cement plant benefits, with avoidance of landfill or incineration of material with no other obvious beneficial use (along with resultant CO_2 emissions) and, additionally, the use of fresh 'noble' resources is reduced. Using cement kilns for recovery and use of alternative raw materials and fuels is termed 'co-processing', a practice situated in the waste hierarchy between recycling and energy recovery. This recognises that when production of waste cannot be managed technically or economically by prevention and reduction, reuse and recycling, cement manufacture (with full energy and material recovery) provides a more ecologically-sustainable solution compared to landfill or dumping. Comprehensive guidance for co-processing in the cement industry has been produced, and includes Figure 4.16, illustrating the waste hierarchy (Cement Sustainability Initiative, 2014).

Figure 4.16 Co-processing in the waste hierarchy



Source: Cement Sustainability Initiative, 2014

The recovery of AFs in the European cement industry increased from around 2.5 per cent in 1990 to over 40 per cent of the overall fuel energy demand in 2015 (Cembureau 2017). Some countries now do better. Orujov (2018) reported that the German cement industry reached an overall calorific replacement rate of 64.8 per cent in 2016. This represents current best practice for whole-country AF calorific replacement, followed by the Czech Republic (62 per cent), Belgium (53 per cent), Poland (52 per cent), Sweden (48 per cent) and the UK (44 per cent) (De Beer et al, 2017). These high calorific replacement rates are being incentivised by the highly-regulated EU waste market. A self-perpetuating trend is established where increased regulation of the waste generation industry creates demand for disposal of waste streams in cement kilns. The fees cement companies earn for satisfying that demand justify the capital investments to allow more wastes to be burnt in cement kilns. A wide range of AFs are burnt in European cement kilns with widely varying calorific values. Depending on the quality and source of AFs, typical heating values for solid materials are between 17-23MJ/kg (see also Table 4.7).

Table 4.7 Examples of calorific values for different types of wastes used as fuels in the 27 nations of the EU

Examples of types of waste fuels (hazardous and non-hazardous)	Examples of calorific values (MJ/kg)
Wood	10-16
Paper, cardboard	3-16
Textiles	up to 40
Plastics	17-40
Processed fractions of domestic refuse (RDF)	14-25
Rubber/used tyres	26
Industrial sludge	8-14
Municipal sewage sludge	12-16
Animal meal, fats	14-18, 27-32
Animal meal (carcass meal)	14-21.5
Coal/carbon waste	20-30
Agricultural waste	12-16
Solid waste (impregnated sawdust)	14-28
Solvents and related waste	20-36
Oil and oily waste	25-36
Oil-shale based fuel mix (85 –90% shale)	9.5
Sewage sludge (> 10% moisture)	3-8
Sewage sludge (< 10 to 0% moisture)	8-13

Sources: Schorcht et al, 2013, and VDZ, 2016 & 2017

A further driver for the burning of AFs on cement kilns are any carbon credits that can be gained by burning AFs containing biomass. In countries with a CO_2 emissions trading scheme, such as Europe and Japan, any biomass fuels (eg, rice husks, animal meal, oil palm husks, coffee husks or other agricultural by-products) burnt in cement kilns are granted a zero-carbon emission rating. Other AFs are partially biomass and are granted a zero-carbon rating for their biomass content. So tyre-derived fuel (TDF) has 25-30 per cent biomass content, while refuse-derived fuel (RDF) can have up to 50 per cent biomass content. Table 4.8 shows typical biomass content of AFs quoted by the IFC (2017).

Table 4.8 Biomass content of waste fuels

Type of waste fuel	Typical biomass content (%)
Spent solvents	0
Waste and industrial oil	0
Wastewater	0
Used tyres and rubber waste	25-30
Industrial sludge	variable
Non-hazardous industrial waste	25-50
Municipal solid waste	35-45
Sewage sludge	100
Construction and demolition waste	Wood fraction = 100% Average wastes = 10-50%
Biomass and green wastes	100%
Animal meal	100%

Source: IFC-World Bank, 2017

These regulatory drivers for AF burning apply equally to other energy-intensive industries such as thermal electricity generation and therefore create competition for such AFs. This competition increasingly limits the amount of biomass fuel that is available to the cement industry.

Various buzzwords and acronyms have appeared. Solid biofuels (usually referred to as biomass) include plant tissues, such as wood, charcoal and yarns; and by-products from agriculture, forestry, fisheries and aquaculture, such as coffee husks, straw, sugar cane and its leaves, rapeseed stems, palm nut shells, rice husks and straw. There are other non-agricultural biomass elements such as algae, animal fat, sewage, waste meats and bones, food scraps and domestic or industrial biodegradable wastes. In all cases, these are primarily composed of carbon based organic matter, which releases energy through combustion (Kerton, 2013).

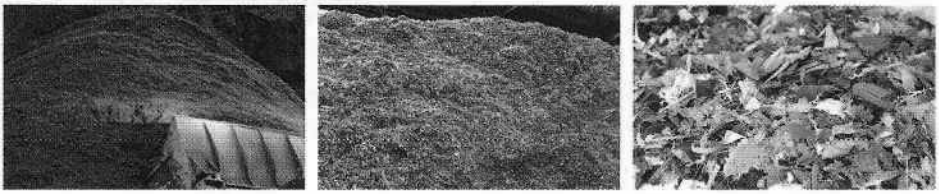
The responsible use of biomass fuel requires the use of procedures that have positive results for people and the planet. As well as complying with all relevant regulations, the entire supply chain associated with biomass production, transportation and use should be positive (produce less greenhouse gas – GHG) than the use of conventional fuels. Biomass production should not adversely affect biodiversity, including protected and rare species, and be protective of soil and air quality, and the quality and quantity of water (both ground and surface). Biomass fuel production should never compete with food supply and community subsistence. Cement plants must ensure that they do not endanger local food supplies by dramatically increasing costs or affecting the life of communities for whom biomass usage is essential for subsistence (for example: heating, medicines, animal fodder, building materials, etc).

Biomass includes bioliquids and biofuels. Bioliquids are fuels produced from biomass and used for energy purposes outside the transport field. Biofuels – usually used for transport – are based on organic materials (plant or animal) and include many wastes, along with some ethanol and biodiesel. Jepsen (2015) forecasts the prospective economics of using CO_2 from exhaust gases to grow algae in solution, for harvesting and use as a prospectively carbon-neutral biofuel.

Untreated domestic refuse has a CV of 8-10MJ/kg and requires processing to remove 50-70 per cent of undesired fractions to yield a fuel that is easier to handle with a CV of 12-16MJ/kg. Sassenrath (2006) outlines Vecoplan's bunker arrangement for receiving, blending, shredding and storing refuse from several sources. Secondary raw materials (SRM) are also a by-product of municipal recycling and consist of materials such as paper, glass, metals and some plastics that have been manufactured, used, then discarded but are to be re-used. Solid recovered fuel (SRF) – also known as "refuse derived fuel (RDF) – is produced by sorting, shredding and drying mixtures of domestic refuse ('municipal solid waste', or MSW) and other low-grade materials. Strict specifications are laid down to maintain a consistent fuel quality. SRF is predominately a mixture of paper, plastic and textiles shredded to a particular size. This sorting and shredding is also undertaken on specialist facilities. Processed sewage pellets (PSP) are made by heat treating the sludge remaining after sewage processing. Meat and bone meal (MBM) - also known as "animal meal" - is the chopped and shredded carcasses of animals and some wastes from meat processing. No cement plant would take on this rendering of the carcasses, which is undertaken at specialist rendering plants associated with slaughter houses.

The 'biomass fraction' of a mixed fuel or material is the ratio of carbon stemming from biomass to the total carbon content. The environmental benefits of using biomass fuel are under question, partly with regard to the sustainability of various products and their delivery and consideration of potentially undesirable side effects upon other sectors such as food production, but in particular the opinion about whether biomass is 'carbon neutral'. This implies that, given a specific set of circumstances, the net transfer of carbon to the atmosphere associated with its use is zero: the debate was current when our previous edition was published and was continuing as we went to press in 2019, centring around the definition of these circumstances and how various legislators have taken them into consideration. The outcome may favour or disfavour the use and development of forest products and perhaps affect both their traditional and emerging uses.

Figure 4.17 Examples of waste fuels



AF preparation

The wide range of calorific values of AFs burnt in European cement kilns as given in Table 4.6 means that such AFs need to be prepared before they can be used in kilns. Andusia (2018) reported that to produce a fuel suitable for the cement industry, general waste needs to be treated by shredding, screening, removal of metals and contaminants, drying, baling and finally, wrapping for transport. The preferred option for cement plants is not to carry out any preparation of AFs within the cement plant. AF preparation complicates the operation and maintenance of the cement works. For that reason AFs that do not require any preparation, such as used car tyres or rice husks, are ideal AFs. Rice husks do not require preparation before burning in a cement kiln and can be delivered directly to the main burner or precalciner at a controlled rate.

Used car tyres can be added as a whole to the inlet of the rotary kiln. However, this inevitably leads to local reducing burning conditions as the car tyre burns in the feed material in the inlet of the kiln and as it passes down the kiln with the rotation of the kiln. These reducing burning conditions will lead to increased volatilisation and circulation of sulphur within the kiln and can lead to problems of build-up, ring or ball formation in the kiln. Therefore, to prevent these issues, car tyres and larger truck tyres are better chipped or shredded before delivering to the precalciner. However, chipping or shredding of the tyres increases the cost of such tyre-derived fuel (TDF). This would not normally be undertaken by a cement factory but by a specialist TDF provider at their own facilities before delivery to the cement works.

When fly ash with (non-volatile) residual carbon is used as a raw material for clinker, it can be conveniently added with kiln feed and yield useful heat without a polluted exhaust (Borgholm, 1992), as can also small amounts of low-volatile petcoke. Note, however, that fly ash sometimes contains high and variable carbon amounts (1-30 per cent) and, unless pre-blended, can seriously destabilise kiln operation.

Di Matteo (2012) reported that the powdery, granulated, pelletised, fibrous and flocculent AFs derived from quality-controlled industrial wastes are generally known as fluff. These AFs have difficult physical bulk material properties such as a very low bulk density, resulting in very high volumes and volumetric flows.

Waste fuel preparation must include a quality assurance system to provide a consistent fuel in terms of composition and calorific value. The operators of kilns rely on the fuels being of consistent calorific value such that changes in the fuel delivery rate will result in consistent changes in the thermal energy supply to the kiln. In addition to the AF supplier's quality assurance, the cement factory must regularly undertake quality control checks on AF batches received in terms of calorific value and composition.

Nevertheless, some cement plants do find that they need to become involved in the preparation of AFs. SRF bales need to be debaled before they can be burnt in the kiln. That debaling can be undertaken manually for a small number of bales. Manual debaling consists of cutting the plastic wrap and plastic ties with a knife, and then breaking and fluffing the bale to loosen the material. Debaling large SRF volumes requires an automated system. The process can be achieved through the use of a waste shredder or a specific system. An automatic debaler is the cheapest and most effective method of debaling SRF, as it will separate the wrap and ties from the SRF and break up the material, making it more suitable for use (Andusia, 2018).

Further processing of the AF in the cement factory might be necessary where there are no suitable specialist processors of AFs in the particular geography of the plant. More common is for cement plants to become involved in the preparation of AFs to mix, dry or reduce the particle size of AFs to allow greater quantities to be burnt on their kiln(s).

Some of the earliest AFs to be utilised on cement kilns were spent solvents from other industries and waste oils. Spent solvents require special facilities due to their low flashpoints and must be continuously mixed to prevent the solvents separating into different fractions with different calorific values, moisture contents and combustion properties. This requires fully-enclosed, explosion-proof mixing tanks. An analogous requirement for mixing would be required with industrial sludges. This might involve mixing with a high-calorific value miscible liquid such as spent solvents or might involve mixing with a powdery carrier such as sawdust. An interesting variant is the use of waste oil lagoons associated with oil refineries in the GCC countries, where the waste oil is mixed with sand in the cement plant and the mixture delivered to the kiln by concrete pump.

AFs such as sewage sludge or RDF have low calorific values, in part due to their inherently-high moisture content. This moisture content needs to be reduced to increase the amount that can be burnt in the kiln. That drying might be completed before delivery to the cement plant. However, cement plants do have waste heat from cooler and preheater exhausts that can be used for drying AFs in drum or belt dryers. Leitão (2018) described Cimpor's waste dryer project to increase the substitution of RDF on the main burner of Kiln No 3 at the company's Souselas plant in Portugal. The belt dryer uses the excess hot air from the clinker cooler, sending the cooled air to the existing cooler stack. The dryer consists of a slightly porous polyester conveyor, 12m long and 6.2m wide, which is crossed from top to bottom by a stream of hot cooler exhaust air. This project enabled the RDF firing rate to increase from 2.5tph at 30-40 per cent moisture to 7tph at 15 per cent moisture. Investment cost of the dryer was €1.86m with a further €1.98m for upgrading the pneumatic transport of the RDF. This investment was projected to be recovered in less than 4.5 years.

MVW Lechtenberg & Partner (2017) described the 'V-Mill' for fine grinding and drying of wet MSW, where drying improves the ignition and combustion of the RDF producing a short, intense main burner flame. The V-mill grinds up to 8tph of 50-120mm RDF with 30 per cent moisture to less than 10mm particle size and less than 10 per cent moisture. The equipment's first installation was at the Dyckerhoff cement plant in Geseke, Germany.

As long ago as 1996 ENCI's cement plant in Maastricht, The Netherlands, began trials grinding sewage sludge. Initially a sewage sludge and coke mixture was ground in a ball mill. These trials led to the construction of a vertical roller 'BioMill' for grinding sewage sludge. The first Biomill was commissioned in February 2000 allowing sewage sludge consumption to steadily increase from less than 1tph in 1997 to 3.5tph using the first BioMill, and to over 6tph in 2004 with the commissioning of a second BioMill. By 2007 the sewage sludge consumption rate had reached 11tph.

AF mixtures such as SRF will have a low density. Therefore, pelletised SRF is an excellent solution to reduce transport costs due to the increased density of the pellets when compared to the loose SRF. However, the pellets should be destroyed again before injecting in the kiln/calciner for better combustion, meaning that they need to be ground in some form of mill before delivery to the kilns (Maia, 2017).

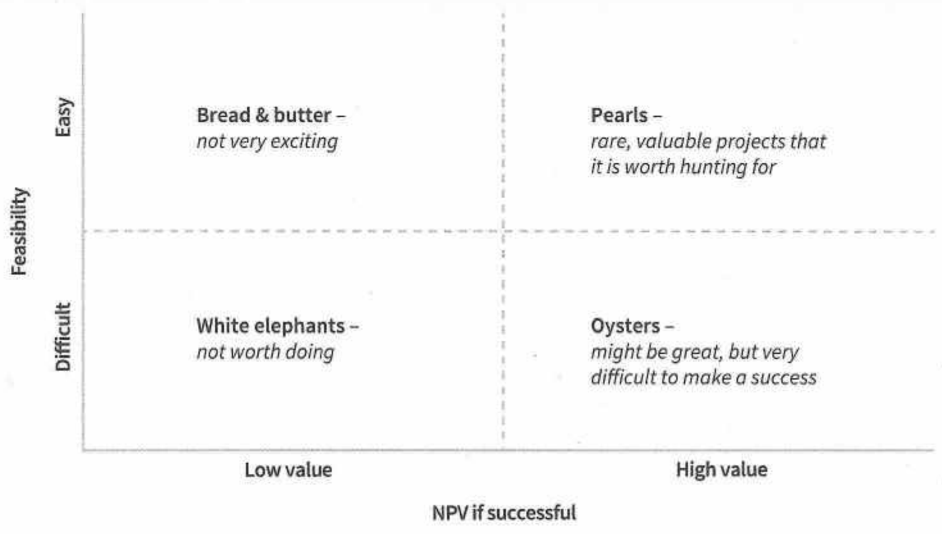
Introducing AFs to a cement plant

Brian McGrath (2014) describes a risk-based approach to implementation of AF projects in a cement factory. This segmented the risks involved into commercial, regulatory, community and technical risks.

Commercial risk

For assessing commercial risk a Boston-type project portfolio matrix is advocated with feasibility of the project on the vertical axis and net present value (NPV) on the horizontal axis (see Figure 4.18).

Figure 4.18 Assessing commercial risk in introducing AFs



Source: McGrath, 2014

Feasibility, or how easy the project would be to implement, ranges from a difficult project with a low chance of success to an easy project with a high chance of success. For NPV, or commercial potential, the range is from a project with low value to one with high value.

Such a matrix approach allows potential AF projects to be segmented into:

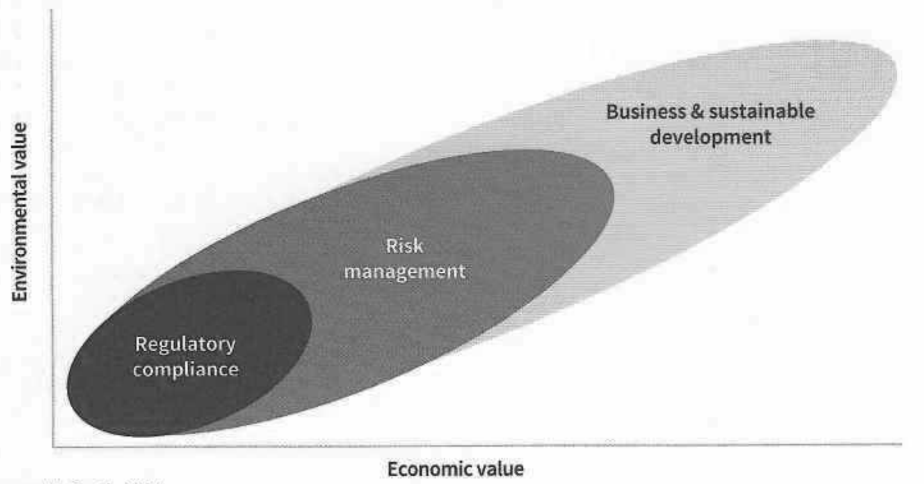
- those with low chance of success and low value as 'white elephant' projects that are not worth implementing
- those with a high chance of success but low value, being 'bread and butter' projects, which are worth carrying out but not very exciting
- projects with a low chance of success but high value if successful being 'oyster' projects, ie there might be a pearl in there, but it will be difficult to make it work
- those 'pearl' projects with a high chance of success and a high value. This approach is recommended because it justifies evaluating those apparently-difficult to do projects (the oysters), because if they are not evaluated (opened) then you will never know whether there is a pearl in there or not.

This matrix approach is an excellent method of deciding whether an AF project is worth implementing commercially. Key considerations will be the price differential compared with the traditional fuel the alternative would replace, the costs of the receipt, storage, handling and dosing systems for the fuel, the operating costs, the amounts of the fuel that can be used and any disadvantages introduced by the AF. All these considerations will be based on the technical risks and assessments associated with the particular AF. It might be low cost or a cement company might be paid to take it, but if it is foul smelling or obnoxious in some other way, then additional equipment will be required to contain and handle it. It might have some trace component that will limit the amount of the fuel that can be used.

Regulatory risk

Concerning regulatory risk, compliance must remain as a central requirement for burning any AF. The AF project will never be a success if regulatory compliance cannot be assured. However, as the economic value or environmental value of an AF project increases so a cement company's efforts need to expand beyond mere compliance to identify any potential risks and the actions necessary to mitigate against those risks (see Figure 4.19). Where there is major environmental value in burning the AF in a cement kiln, cement companies can work proactively with the regulatory bodies to develop the project. Prime examples would be sewage sludge, RDF or meat and bonemeal where the cement kiln provides an alternative disposal route, avoiding the need for landfilling or incineration. Some projects are very valuable to the community as well as the cement company and it is worth expending major efforts to get them approved and accepted.

Figure 4.19 Regulatory risk and going beyond compliance



Source: McGrath, 2014

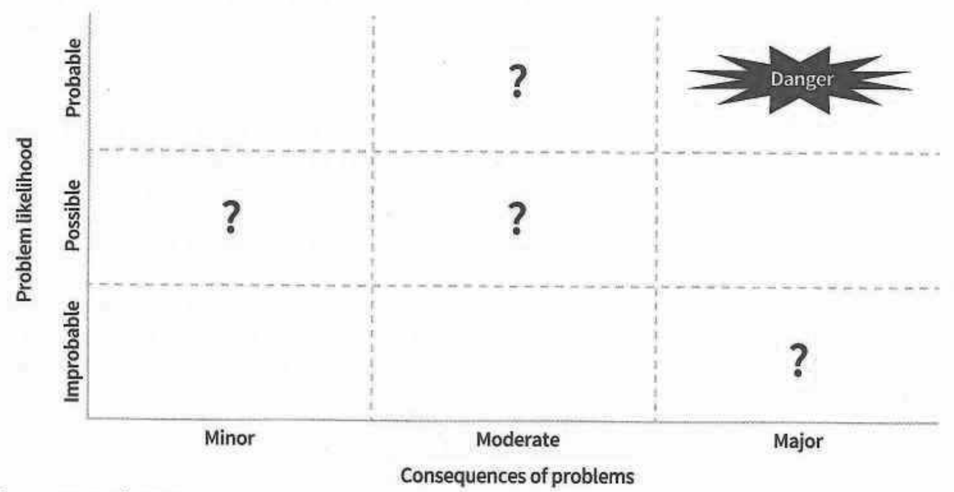
Community risk

Working proactively with regulatory bodies and the society within which the cement company operates is also the key to minimising community risk. The need for and benefits of maintaining good relations and communication with local communities cannot be overstated if burning AFs is to be successful for a cement company.

Technical risk

Finally we come to technical risk. McGrath again advocates a matrix approach to technical risk assessment considering whether the likelihood of a technical problem is either improbable, possible or probable on the vertical axis, and whether the consequences of that technical problem arising would be minor, moderate or major on the horizontal axis (see Figure 4.20). Clearly the AF projects to be most wary of are those where the likelihood of a technical problem is probable and the consequences of that technical problem would be major. An example might be the presence of some trace component that would be emitted in the exhaust from a cement kiln and would be toxic in the environment. Such a combination would be sufficient to kill the potential AF project.

Figure 4.20 Technical risk matrix



Source: McGrath, 2014

Thankfully such combinations are unusual. It is more likely that technical judgements will be required to identify the potential technical risk, the probability of it arising and the severity of the consequences if it did arise. Also to identify the mitigation strategy to manage the technical risk. Potential technical risks and technical judgements/decisions to be made include the choice of AF firing location, understanding the limits of the particular cement kiln and precalciner, the impact on chloride and alkali cycles, impact on clinker productivity, the limits on the amount of the AF that can be burnt and matching the amount of available fuel to the chosen firing location.

The following approach to assessing such technical risks and arriving at the required technical judgements/decisions is suggested:

- Know the characteristics of the AF, ie variability of composition, moisture content and calorific value, density, size grading and materials handling properties, combustion behaviour, environmental, operational, health and safety issues in handling the fuel.
- Understand the raw materials, raw mix, current fuels, kiln feed and clinker produced on the particular kiln or cement factory. Also an understanding would be useful of the limitations and bottlenecks of the particular kilns or calciners as well as the chloride and alkali cycles in the kiln and the external cycles between kiln and raw mill or dust collection equipment.

Armed with the this knowledge and understanding, certain judgements can be made on the potential impact on clinker output and composition/quality/mineralogy as well as on the potential impact on emissions of NO_x, SO_2, trace metals, dioxins, furans and plumes from the kiln(s). These judgements will lead to setting the limits on the amount of the fuel that can be burnt and the appropriate firing locations.

With the most appropriate fuel firing location having been chosen, the fuel quantity can be matched with the chosen firing location. The equipment for receiving, storing, dosing and delivering the fuel into the kiln at the chosen location can be selected. Key considerations will be the reliability of that equipment and it's ability to contain the material and deliver it consistently into the kiln. Finally, having been selected the equipment can be integrated into the existing cement factory layout.

These technical considerations will inform the commercial, regulatory and community risk assessments. Following this methodology should guide a cement company on which AF projects to pursue and give the best chance of making a success of the projects that are pursued.

Cement plant equipment modifications for AFs

Equipment will need to be installed for receiving, storing, dosing and delivering the AF into the kiln for the introduction of any AF to a cement plant. Initial trials might take place with temporary feeding, storage and feeding systems, but if the trials are successful then a permanent installation will be required.

Storage

From a production engineering standpoint AFs are no different to any other input to the cement manufacturing process. Sufficient buffer storage capacity must be provided to ensure that regular consumption of the AF can be sustained while the deliveries of AF to the cement factory are intermittent. Therefore, determining the appropriate storage capacity requires knowledge of the anticipated AF consumption rate and the AF delivery rate and frequency into the cement plant. If 2tph of AF will be consumed and there will be no deliveries from Friday evening to Monday morning then a minimum storage of 2 x (8 + 24 + 24 + 8) = 128t must be provided and that storage must be full at the beginning of Friday evening. A larger buffer storage of at least 200t would be appropriate to guard against the storage not being absolutely full on Friday evening, or some delay in the arrival of fresh deliveries on Monday morning.

The ideal would be a fully 'just-in-time' supply of AF so that no buffer storage would be required. However, that would require complete control over the delivery rate of the AF to the cement factory, which is seldom achievable. It is often necessary to continue to be able to take deliveries of AF when the kiln is not operating. The waste processor supplying the AF may demand a continuous offtake such that their waste processing is not adversely impacted by kiln shutdowns. As repair of the refractory lining of the kiln may sometimes require 20 days or more it might be that AF storage capacity of 20 days or more must be provided to satisfy such arrangements with waste processors.

The type of storage can vary from open storage in the cement factory yard or a building, to fully-enclosed circular or rectangular silos. The appropriate choice will require a technical and economic feasibility assessment.

Handling and transportation

AFs require complete characterisation to enable correct design of storage and feeding equipment: relevant characteristics are appearance, bulk density, compressibility, particle size distribution, contents of moisture, chlorine, sulphur and heavy metals, ignition temperature, calorific value, retention time (to define injection velocity ex-burner), maximum rate of explosion pressure rise in dust clouds, maximum explosion pressure rise of dust clouds and dust explosion class (to define ATEX zones). (Aufderheide and Di Matteo, 2018[1]).

Most wastes are more difficult to handle than conventional fuels, and appropriate storage and handling systems are key factors for sustained success. Sometimes abatement of odours from fuel stocks and dryers may be necessary, perhaps with venting to the clinker cooler. Aufderheide and Di Matteo (2017) discuss AF characteristics and the design and selection of suitable types of feeding and dosing equipment. Inadequate storage and handling equipment will have adverse effects on plant performance, not least decreasing system availability.

Open storage and reclaim with a wheel loader will be the lowest capital cost option. However, the properties of the AF such as smell and dustiness need to be taken into consideration. Operating costs will also be higher as a wheel loader and driver will need to continuously deliver the AF into the process. An automated process crane can be used to reclaim AFs from open storage eliminating the operating cost of the wheel loader but at the expense of initial capital cost. Where bales of RDF or SRF are stored in open storage the AF can be debaled manually for small input rates or a debaling machine can be provided. Again the economics of capital cost for equipment and operating cost for labour need to be considered. Where AF inputs rates are high or the AF has noxious properties, investment in process cranes and debaling machines becomes advisable.

For noxious AFs fully-enclosed storage in circular or rectangular silos is the preferred option. The cement industry is familiar with circular silos for the storage of kiln feed and finished cement, however, these are not the ideal storage for AFs. Di Matteo (2018[2]) explains that AFs undergo a consolidation effect once they are not continuously moved or activated. This 'time consolidation' is a consequence of the increase of interparticle adhesive forces with time. RDF can exhibit a 10-fold increase in the bulk density. Biomass can exhibit a seven-fold increase in the bulk density. These affect the flow properties of the AF. Circular silos are usually significantly higher than their diameter meaning that a high column of material is stored. For AFs this promotes the consolidation of the fuel with attendant change in flow properties. Agitators should be provided to ensure the continuous flow of potentially sticky AFs from such circular silos. Rectangular silos are preferable as the material depth can be limited to 5-10m maximum and the length and width of the store adjusted to provide the required storage capacity.

Di Matteo further recommends that the delivery and extraction of AF from the silos should be on a 'first-in, first-out' (FIFO) principle. One option for filling and extraction from such a silo is a loading and unloading conveyor, but that inevitably extracts the material from the top of the pile leading to 'first-in, last-out' (FILO) operation. A better option for extraction is a moving or push floor that extracts from the bottom of the pile and does work in a FIFO manner.

Receipt into the storage is commonly via reception boxes capable of receiving moving floor or tipper trucks. The reception boxes are often provided with moving or push floors for the extraction of the received AF to storage.

The physical properties of the AF determine the equipment required for transportation of the AF to storage and from storage to the injection point into the kiln. Roller conveyors are commonly deployed for whole motor tyres. However, with the majority of dusty, low-density, fluffy AFs, containment of the AF within the transporting equipment is a primary consideration. Chain-belt conveyors, side-wall conveyors or pipe conveyors all provide the necessary containment and transport of the AFs. Vecoplan's air-assisted pipe conveyors minimise the requirement for the mechanical maintenance of the rollers of such long-distance AF conveyors (Hamer, 2018).

Feeding and dosing

Delivery to the kiln should be via a gravimetric dosing device to control any variation in the density of the AF. The gravimetric dosing device must have an accuracy of ±1 per cent, with a wide turn-down range, short response time, good reproducibility and reliability. Secondary fuels generally have a high dust content, which represents an extraordinarily high explosion hazard. Therefore the gravimetric dosing system must be enclosed and in some cases explosion shock resistant (Marotz et al, 2012). Similarly, fire protection must be considered with all receiving, storing and transporting equipment.

For actual delivery into the precalciner or kiln inlet, minimising the inleak of false air with the AF is a primary consideration. Double flap airlocks or rotary valves are commonly deployed. Delivery of AFs to the main burner is most commonly carried out pneumatically through a dedicated channel in the main burner of the kiln.

AFs differ markedly from conventional fuels in regularity of properties and combustion characteristics. Plant experience suggests that maximum flame temperature may be lower, leading to higher temperatures at the upper end of the kiln. For high substitution rates, the demands upon burner and kiln operation increase. Fuels are often retrieved from materials that were originally produced or used for other purposes. In contrast to the homogeneous properties and a high specific surface of pulverised coal, solid AFs exhibit higher heterogeneity and moisture content, a lower specific surface and a considerable variation of physical and chemical properties affecting combustion.

The equipment installed for receiving, storing and dosing AFs into the process should be suitable for the handling of multiple AFs. Supplies of waste suitable for use as fuel tend to be variable with waste material markets rapidly developing. Equipment installed for burning animal meal might have to be switched to burning RDF or other SRF. Once committed to burning AFs a cement company and factory have to continually search for new AFs to burn and respond to waste market opportunities. The AF equipment installed should ideally be suitable for multiple AFs.

AF burners and kiln systems

It is important to note that special burner designs have been developed to cater for the simultaneous firing of fuels with different properties, with flexibility to cater for changes in their nature as economics dictate. Among the established turnkey suppliers, FLSmidth launched the Jetflex® design in 2017, thyssenkrupp offers its established Polflame®-VN system and KHD its well-known Pyrojet® design.

As regards the specialist burner suppliers, alongside their established Rotaflam®, Fives Pillard introduced the Novaflam® design in 2009, involving a novel tip designed with CFD methods (de Piver, 2013). A TEC offer the Flexiflame® high-momentum burner, placing a new generation on the market in 2013 (Helmreich, 2012 and 2013), Dynamis proposes its D-Flame design (Favalli et al, 2015), Unitherm-Cemcon produces its MAS® burner, whilst FCT's waste fuel offer is the Turbu-Jet® (Hassold, 2018). With these new designs, 100 per cent AFs can be successfully achieved. D'Hubert (2017) reviews recent trends.

Multi-fuel/multi-channel burners provide the necessary flexibility to fire ground or coarse solid fuels, as well as liquid, paste-like and gaseous fuels. The intensity of heat supplied and the temperature profile in the kiln are influenced both by the combustion performance of the fuels and the burner settings. It is possible to gradually shape the flame and enhance combustion by precise control of air injection. Flame geometry, temperature distribution and fuel combustion can be altered by the geometry of the outflow system and by the volume flows and momentum of primary and swirl air streams at the burner tip, opening the way to greater rates of fossil fuel substitution.

With wastes, the kiln system may well have to handle larger flue gas volumes and supply a greater volume of combustion air, or operate at a lower output rate. Also, the flame heat release profile is likely to change. To avoid disappointment, these matters should be addressed at the design stage.

The cement plant modifications described above are required for the introduction of AFs into the process. Turesay of KHD (2018) introduced the concept of 'processing depth'. The greater the proportion of the thermal energy requirements to be provided by AFs the greater are the investments for kiln modification likely to be. The 'V-Mill' described by MVW Lechtenberg & Partner (2017) or the ENCI Maastrict Biomills were examples of the need to install additional equipment to raise the proportion of AFs that could be burnt on the kilns of the Geseke and Maastrict cement plants. A common limitation on the ability to burn AFs arises from the elevated chloride content of AFs such as RDF, SRF, animal meal or waste plastics. Leitão (2018) reported that the limitation on the amount of RDF that could be burnt at Souselas after the installation of the belt dryer was the chloride content of the AF rather than the dryer capacity. Installation of a chloride bypass at the kiln inlet is a common requirement to reach high AF substitution rates. Chloride is only one example of unwelcome trace components that might be introduced with AFs as discussed in the following section.

Turesay's 'process depth' was provided by KHD's PYROCLON CC (Combustion Chamber) and PYROROTOR for burning lumpy AFs with the PYROROTOR providing 10min residence time for the combustion of such lumpy AFs. FLSmidth's HOTDISK or the thyssenkrupp PREPOL step combustor also provide extended residence time for the combustion of lumpy AFs.

Lump fuels

Used tyres are attractive, though the cost of shredding or pyrolysis eliminates much of the potential financial benefit, while adding fuels such as whole tyres or baled waste discontinuously at the precalciner or upper end of the kiln de-rates the system, since sufficient oxygen must be maintained to support peaks of combustion.

Often, the first option that springs to mind for a new fuel is fine grinding and firing in suspension, like coal or petcoke. However, some materials come in the form of sheets that are shredded and old ideas about 'particle diameter' need to be revisited. In other cases, a heterogeneous or difficult-to-shred character renders fine grinding impracticable. Shredding to below 30mm maximum size is generally costly, so that it is preferable to establish burner settings that assure complete combustion for such "flakes". Sturn and Willitsch (2006) of A TEC describe storage and processing options for the more popular European waste fuels. Among those offering more complete solutions is Vecoplan, and Baier and Menzel (2011 and 2012) described Vecoplan's approach when deciding how to manage different streams of waste. For the kiln, they recommended a flexible multichannel burner, with primary air nozzles that are selectively adjustable, both radially and tangentially. In particular, several methods of using 'lump' fuels (above some 20-30mm in effective diameter) are being used or developed, including pyrolysis or gasification alongside the kiln.

Maximum fuel and gas residence times in conventional calciners are typically 4-10s (twice as long for raw meal) and will be as low as 2-3s for unmodified older designs intended for use with conventional fuels (at 5-10t of clinker/day/m^3 of calciner volume), whereas residence times of up to 15min or more could be needed for 'difficult' fuels. KHD has added a precombustion chamber to its Pyroclon calciner to safely and completely burn fuels with a low CV with only a low degree of preparation, using pure tertiary air, with no vitiated combustion gases. Hot calcined raw meal is introduced around a central swirling flame which attains a high temperature. (Schümann, 2012).

PSP offers an analogous technique in its KS-AS-W two-stage system, again ensuring that combustion occurs in oxygen-rich air in the centre of the first (off-line) chamber, with the walls protected by a curtain of meal. Fives-FCB also proposes a precombustion chamber with a hotspot and flame surrounded by fresh air fed from the tertiary duct with hot meal mixed in (Devroe, 2012). This design may benefit from precombustion chamber trials by Technip-CLE (now part of Fives-FCB). This burned several fuels (including coal, petcoke and plastics) at high temperatures (1100-1600°C) and, in addition, can accept fuels with poor ignition behaviour (eg petcoke). Both Fives-FCB and KHD additionally claim low exit levels of NO_x.

Precombustion devices usually burn waste tyres, sticky materials, tyre chips or low-volatile fuels. In the FLSmidth 'Hot Disc', waste materials (whole tyres or fluffy materials) are dropped onto the turntable. When exposed to hot tertiary air, fuels slowly gasify and burn out while the turntable rotates (10-45min/revolution). The disc is connected to the kiln via the precalciner riser duct. In the TKRT precombustion chamber, coarse materials (usually whole tyres) are burnt and converted into a fuel gas, petcoke and residues which can be fed separately to the calciner or the kiln inlet.

The residence time in conventional precalciners is commonly insufficient for burning low calorific value AFs. Mersmann (2018) described how the AF substitution rate in the twin separate line calciners of a semi-dry kiln was raised to 100 per cent by narrowing the precalciner to increase gas velocity and also extending the precalciner to increase residence time. Guidance for the use of fuel blends for the main kiln burner is that the net calorific value of the fuel mix should be above approximately 20-22GJ/t, whilst for use in the calciner the net calorific value should exceed approximately 13GJ/t.

Temperatures at the main burner of the kiln are >500°C higher than the precalciner. Therefore, the combustion of the AF is more intense. This is the reason to that waste materials likely to contain relevant concentrations of persistent organic substances, eg PCB-laden spent oil, are fed via the main firing system to ensure their reliable destruction (Schorcht et al, 2013). However, burning multiple AFs with different calorific values and combustion properties at the main burner of the kiln can lead to lengthening of the flame with two or more temperature peaks. This is detrimental for clinker quality and can limit the AF substitution rate at the main burner. Oxygen enrichment of the main burner combustion has been applied to overcome such problems and boost the amount of AF that can be burnt at the main burner when economically viable.

AF impact on the process, clinker quality and emissions

The impact of increased chloride inputs with AF on chloride cycles has already been discussed, as has the impact of differential combustion rates of AFs injected through the main burner of the kiln. Also noted is the increased sulphur circulation that can arise from whole tyres of lumpy AFs burning in the feed material causing locally reducing burning conditions.

Higher chloride input with AFs will increase the intensity of chloride cycles between the rotary kiln and the preheater with the chloride content in the hot meal at the base of the preheater rising towards 100x the inputs in the feed and fuel. If the chloride content in the hot meal exceeds two per cent then the material will become increasingly sticky due to the low melting temperatures of alkali chloride compounds. This will cause build-up to be formed in the lower stages of the preheater and can lead to restriction of the gas flow through the preheater, and in the worst case, blockages to the flow of material through the preheater. Clearing such blockages is dangerous work and to avoid them, the installation of a chloride bypass to allow higher AF substitution rates can be required.

Similarly, high sulphur circulation can cause build-ups in the preheater and kiln inlet as well as ring formations in the upper sections of the rotary kiln. Problems of dusty and high free lime clinker can also be experienced. If lumpy AFs are introduced to kilns with a high sulphur burden then the sulphur recirculation caused by the local reducing conditions can soon lead to severe choking of the preheater. From a sulphur circulation perspective any local reducing burning conditions should be avoided and therefore, the size of AFs should be reduced to ensure that no AF burns in the material charge in the kiln. However, those local reducing conditions caused by burning of lumpy fuels in the kiln inlet do result in degradation of thermal NO_x formed in the main burner flame, and therefore, can reduce NO_x emissions from the kiln.

Ensuring sufficient combustion air is drawn into the kiln is a prerequisite with the burning of AFs. Fives-Pillard (Morata, 2014) advised that AF burning requires higher excess oxygen of four per cent at the kiln inlet. The combination of insufficient oxygen and lumpy AFs burning in the material at the kiln inlet can lead to CO and volatile organic carbon (VOC) emissions from the kiln.

Lengthening of the main burner flame and the formation of multiple temperature peaks in the burning zone reduces clinker quality as rapid heating to the final combination temperature followed by quench cooling results in the most reactive C_3S crystals in the clinker. Drying and size reduction of AFs burnt at the main burner are the ways to avoid these problems in addition to potentially enriching the primary air with oxygen to increase the intensity of the combustion.

Burning AFs on a cement kiln invites greater scrutiny from regulatory authorities. Increased emissions testing is commonly required and in many cases continuous emissions monitoring and reporting. The majority of any heavy metals entering a cement kiln with AFs will be bound with clinker and pass out of the kiln. However, mercury and thallium are too volatile to pass through the kiln and will build to high concentrations in the external cycle between the preheater and raw grinding mill, eventually to be emitted into the atmosphere. RDF and sewage sludge can contain mercury – therefore, the mercury content of such fuels should be regularly checked and controlled.

One of the key advantages of a cement kiln for disposing of wastes is that any ashes from the combustion of the waste will be bound in the clinker product, as is the ash from coal. As with coal the chemistry of the kiln feed must be adjusted to produce the target clinker composition after incorporation of the ashes. This presupposes that the amount and mixture of fuels with their ash contents is known. Kiln feed is produced to the required chemistry ready to combine with the known mixture of ashes. Problems can arise when the mixture of AFs changes and the prepared kiln feed is no longer matched to the AFs that will actually be burnt. For most AFs this mismatch is negligible. However, with rice husks with their ~25 per cent ash content, which is ~90 per cent silica, this can mean that the prepared kiln feed has an LSF more than two per cent higher than required for the AFs that will actually be burnt. That is likely to increase overall thermal energy consumption and may lead to the production of high free lime clinker.

Similar problems might be encountered with spent pot liners (SPL) from the aluminium industry used as an AF. This SPL is primarily carbon but will contain sodium and fluoride from the cryolite used as the electrolyte in the electrolytic pots. The fluoride in the SPL acts as a mineraliser promoting the combination of the clinker in the kiln. If the supply of SPL is interrupted then the kiln feed will become more difficult to fully combine in the kiln.

All AFs need to be checked for the presence of chemical species that are not normally encountered in cement raw materials, which may adversely affect clinker quality. Animal meal is an example where the phosphate from the bones of the animals is an oxide that is not commonly found in cement raw materials. C_3S can take up to 0.5 per cent P_2O_5 into solid solution with no loss of hydraulic properties. Beyond that level there is progressive decomposition of C_3S in favour of a solid solution of C_2S and tricalcium phosphate, C_3P. If the P_2O_5 content rises beyond ~5 per cent then C_3S is no longer a stable phase and completely breaks down to C_2S, C_3P and CaO. Therefore, the proportion of the total kiln fuel that can be provided by animal meal is constrained by limiting the P_2O_5 content of clinker to less than 0.5 per cent.

Kilns employing AFs use detailed supply specifications to prevent operational or environmental problems, and each shipment is usually sampled and checked before use to ensure compliance. Typically, CV should be above 4000kcal/kg, chloride is limited to one per cent (depending upon the proportion of the waste being blended in the kiln fuel) and most plants decline fuels with PCB content exceeding 50ppm. If waste fuel is introduced to the kiln flame, all organic compounds can be completely destroyed (>99.99 per cent) while all trace metals except Se, Cd, Tl and Hg can be contained in clinker or kiln dust (>99.8 per cent) (Von Seebach and Tompkins, 1991). Non-hazardous waste fuel may be burned in the riser or the precalciner, but some compounds may volatilise before combustion. Solid waste may also be pyrolysed before injection, although this is not yet a very common practice.

ECRA (2017), in looking to the future of the industry, has provided analysis of some of the key success factors for establishing a strategy for a successful AFs and materials business.

In summary, the use of AFs and materials has little impact upon cement quality, provided that sensible guidelines are followed. To avoid the two main potential problem areas for the cement plant and its customers, attention should be paid to the following:

1. variable burning (which can be taken care of via QA/QC and investment in appropriate equipment)
2. changes in clinker chemistry that impact upon key clinker properties (which can be compensated by specification of the fuels and materials, ensuring that these add no more than 0.2 per cent to the content of metals in clinker, plus adjustment of raw materials as needed, eg, limiting clinker P_2O_5 content to avoid belite stabilisation at the expense of alite), and adjusting raw feed LSF to compensate for changes in ash levels.

To avoid loss of output it may be necessary to invest in handling higher gas volumes and delayed fuel burn-out. Again, various modelling methods have been successfully employed to indicate the most appropriate and cost-effective modifications.

Burning high levels of AF in a calciner can be problematic. Features which influence burn-out include:

- gas retention time
- geometry (to increase particle retention time and turbulence to enhance gas mixing)
- hot core combustion
- kiln riser restrictor (to increase velocity and reduce any tendency of coarse fuel to fall back into the kiln and burn in the material bed)
- a chlorine bypass system (allowing acceptance of higher levels of Cl-rich fuels)
- other measures to enhance turbulence and hence fuel/air mixing. CFD modelling (see Section 4.8) has provided several successes in this field.

5. Burning and cooling

The basic cement kiln system comprises a preheater in which feed is heated by hot exhaust gas, a precalciner in which the majority of fuel is consumed and the feed is prepared for transition to a fired rotary kiln in which clinkering reactions occur, and a cooler in which hot clinker exchanges heat with ambient combustion air.

5.1 Chemical reactions

As shown in Table 5.1, kiln feed is subject to successive reactions as temperature increases (Hewlett, 2003 and Taylor, 1997).

Table 5.1 Reactions of kiln feed

Temperature (°C)	Reaction
100	Evaporation of free water
>500	Evolution of combined water
>600	Decomposition of any $CaCO_3$ which is intimately mixed with silicon compounds
>900	$CaCO_3 \rightarrow CaO + CO_2$ (This reaction is called calcination.)
	Reactions between CaO and Al_2O_3, Fe_2O_3 and SiO_2
>1200	Liquid formation
>1280	Formation of C_3S and complete reaction of Ca

The artificial clinker minerals are combined from the oxides of calcium, aluminium, silicon and iron, forming C_3S, C_2S, C_3A and a ferrite phase with composition close to C_4AF, along with a level of residual free (uncombined) lime low enough for the cement to yield sound mortar and concrete, unaffected by the expansive hydration reaction of lime to form $Ca(OH)_2$. Final combination takes place via a molten flux phase of around 25 per cent by weight of the clinker. Spahn (2004) reviews clinker formation and concludes:

- Dimensions of alite crystals (industrial C_3S) are largely determined by the particle size of limestone/marl in kiln feed.
- Size and distribution of SiO_2 particles in raw meal have a decisive influence on alite and belite (industrial C_2S) formation.

Minor elements in raw materials are incorporated to the Bogue compounds, both altering elemental compositions and/or stabilising high-temperature crystalline forms (polymorphs). These are usually more reactive towards water, sulphates and admixtures than lower-temperature polymorphs, so that industrial clinker should be of better quality than expected for pure Bogue potential compounds. Industrial clinker also contains minor phases such as free lime, free magnesia (periclase) and different sulphate compounds, all of which affect concrete performance.

Chemical equilibrium is not attained and the industrial minerals contain different materials that influence clinker composition and reactivity in various ways, not apparent from the plain Bogue calculations presented in Section B5.1. The effects of minor elements from wastes upon quality and process must be assessed, alongside economic benefits (Bhatty, 1995; Moorthy, 2003 and Klaska et al, 2003). In general, actual alite content is around 13 per cent higher than calculated, belite is usually some six per cent lower, whilst other phases can be either higher or lower than calculated (Manias, 2011).

Minor constituents entering the process may influence production costs and product quality. Two important properties are volatility and condensation characteristics. As feed is heated, the greater the volatility of a substance, the more is driven off and carried back in the kiln gases to condense in cooler regions. Condensation characteristics of the component (or compounds it may form) and the plant temperature profile influence whether it (a) condenses in the kiln or preheater, possibly causing blockages, (b) condenses on dust escaping the system, possibly causing a handling problem, or (c) escapes up the stack, possibly resulting in an environmental problem. Non-volatile components are more likely to influence combinability, and the quantity and nature of the liquid phase in the burning zone, and hence, clinker nodulisation and/or the propensity for clinker rings to form, as well as affecting product quality.

Table 5.2 Origins of some minor components

Material	Typical minor components
Limestone or chalk	Mg, S, K, Na, Sr, F, Mn
Shale, clay and coals	Mg, S, K, Na, Cl, P, Ti, Mn, Pb, F, Cd, Ga In, Tl, Ge, As, Te, Se, Cu, Ni, Cr, I, Zn
Iron oxide/ore	S, Tl, B, Ti
Bauxite	Ti, Cr, Ti, Mn
Blast furnace slags	Mg, S, Ti, Mn, K, Na, Cl, Sr, P
Petcoke	S, V, Sb, Se, Te, Ni
Used tyres	Zn, Fe, Pb, S, Cu
Waste solvents	Cl, Cd, Hg, Zn, Pb, B, Ni
Waste oils	Li, Sr, As, S, Cl, Cu, Zn, Co, Ni
Fly ash	Be, Ga, In, Tl, Sn, Se, Te, V, Mn, Co, Cu, Cd
RDF/WDF	Ba, Ga, In, Tl, Sn, Pb, P, Se, Te, Cl, Ti, Cr, Ni, Zn, Cd, Hg
Sewage sludge	P
Animal meal	P

The cyclic behaviour of the main volatile species (Cl, SO_3, K, Na) is discussed later, but in addition, both these and some other substances can enter into solid solution with silicates and aluminates, perhaps altering their relative proportions, substituting for the expected elements, stabilising one of them at lower or higher temperatures, or encapsulating lime before entering solution. Magnesium, potassium, sodium, sulphates, manganese, chlorides, fluorides and trace metals all influence the liquid phase formed during clinkering – either its temperature of formation, its quantity or its viscosity/surface tension. Without due care, further elements may end up in the clinker, presenting potential health hazards to customers, or be emitted to atmosphere with the potential to damage the ecosystem either locally, regionally or globally.

The tolerable amounts vary somewhat from plant to plant, depending upon what other elements are present, kiln system design and operation, and the ease of burning of the particular raw mix. It is important to carefully consider these implications when planning changes to raw materials or partial substitution of a fuel. Table 5.3 lists tolerable input loads from all origins, taking a deliberately-pessimistic view. Slight excursions above the listed values may be tolerable at a specific location, but preventing significant excesses will limit the use of a material.

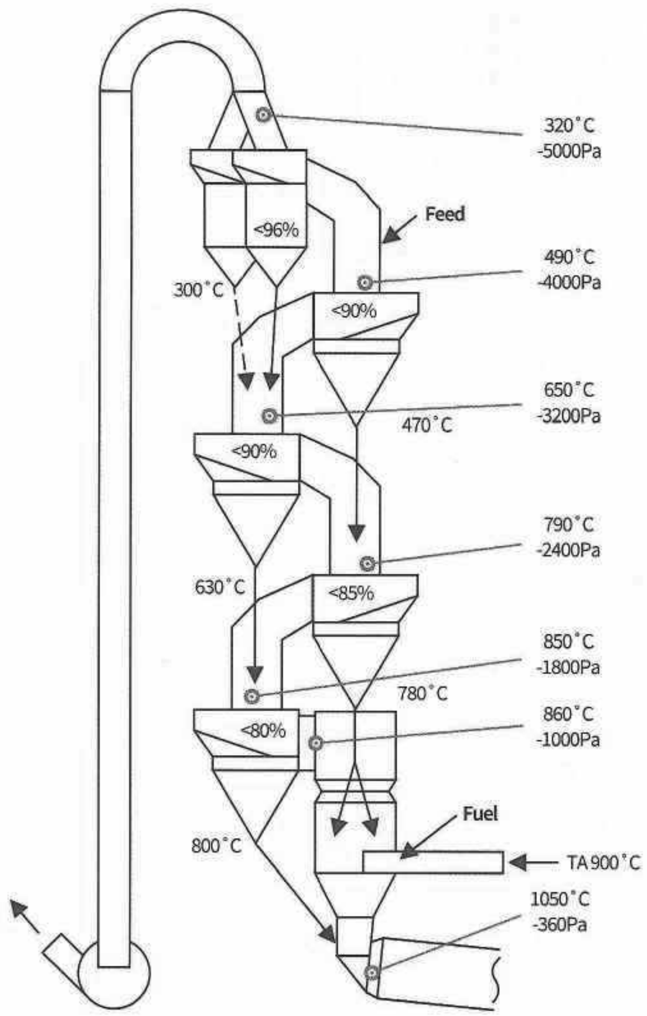
Table 5.3 Summary of minor and trace component limits

Element	Maximum recommended input		Limitation E = Environment H = Health at work P = Production process Q = Clinker quality
	(kg/t clinker)	(% by wt in clinker)	
Antimony (Sb)	3	0.3	E
Arsenic (As)	0.2	0.02	E/H
Beryllium (Be)	0.09	0.009	E
Cadmium (Cd)	0.01	0.001	E
Chromium (Cr)	0.1	0.01	H/Q
Fluorine (F)	2	0.2	Q
Cobalt (Co)	0.1	0.01	H
Mercury (Hg)	0.002	0.0002	E
Lead (Pb)	0.3	0.030	E
Nickel (Ni)	0.1	0.01	E
Selenium (Se)	0.05	0.005	E
Tellurium (Te)	0.4	0.04	E
Thallium (Tl)	0.005	0.0005	E
Vanadium (V)	0.6	0.06	E/P
Zinc (ZnO)	10	1.0	Q
Strontium (SrO)	5	0.5	Q
Barium (BaO)	5	0.5	Q
Phosphorus (P_2O_5)	10	1.0	Q
Titanium (TiO_2)	10	1.0	Q
Copper (Cu)	10	1	Q
Sulphur (SO_3)	15	1.5	P
Chloride (Cl)	0.3	0.03	P/Q
Alkalis (eq Na,O)	10	1.0	P/Q
B_2O_3	5	0.5	Q

5.2 Process variants

Cyclone preheater kilns have developed considerably since first introduced by Humboldt (now KHD) in the 1950s. Several early four-stage preheater units suffered blockages, slowing acceptance until design modifications were proven during that decade. Relative to the previous long wet and dry kilns (Section B5.17), air suspension in cyclones greatly increased efficiency of heat exchange between hot gas and feed material up to about 800 °C and also allowed about 10-15 per cent calcination to occur before hot meal entered the kiln. Kiln gas is cooled from, typically, 1100 °C to 350 °C. Feed is preheated by what appears to be counter-current flow but is in fact a series of parallel (co-current) flow processes in each successive duct and cyclone (see Figure 5.1). Heat transfer in each stage is completed in less than one second as the powder, redispersed by impact upon a plate, accelerates towards the gas velocity of 10-20m/s.

Figure 5.1 Typical profile of precalciner preheater (five-stage)



The remaining heat treatment occurs in a rotary kiln: a slowly-rotating tube with a length-to-diameter ratio of 10-16, inclined at an angle of around 3-5 per cent to the horizontal, heated by a flame at the lower end. Solid material passes down by a combination of slipping and rolling, counter-current to the combustion gas.

There are a number of potential process bottlenecks, notably the 'kiln hearth' where the solid feed changes direction at the bottom of the preheater and moves into the kiln in opposition to the exit gases. Material pick-up in the hearth can generate excessive pressure loss in the riser duct and limit kiln output. Inadvertent apparently minor changes to the open area when re-bricking can markedly restrict throughput. The hearth must be designed to keep gas velocities as low as possible - certainly below some 30m/s - to prevent undue entrainment of material but high enough to avoid material short-circuiting the final preheater stage and falling straight into the kiln. The specific throughput rate per unit duct area (tph/m^2) can be a useful parameter for comparing cyclone performance in different systems.

During the early 1970s, IHI of Japan developed the Flash Calciner process, employing an additional vessel between the first (lowest) and second cyclone stages. The calciner is fired with about 60 per cent of the total fuel input and combustion air is taken directly from the clinker cooler via a separate (tertiary) duct. Feed discharged from stage 2 is fed to the calciner and the reaction product is collected in the lowest (stage 1) cyclone. Material is discharged from stage 1 after less than a minute, apparently over 90 per cent calcined (including some recycled kiln dust, already 100 per cent calcined), and enters the kiln. The speed of rotation for a precalciner kiln is usually at or above 3.5rpm, faster than for the earlier preheater kilns and more antiquated long kilns (outlined in Section B5.17). With a greater degree of precalcination ahead of the kiln, higher rotational speeds bring freshly-calcined material rapidly into the burning zone, preventing it becoming dead-burned and unreactive (Clark, 2013). Section B5 provides some supporting data for this chapter. (Note that while the usual historic convention was to count cyclone stages from the top down, we count here from the bottom for clarity, when the number of stages is above four.)

Due to the reduced degree of calcination required in the rotary section, kiln size has been reduced by about 50 per cent, alleviating problems in maintaining linings in large-diameter kilns. Whilst the addition of a precalciner in itself does little to improve fuel consumption, it allows the scale-up of production by a factor of two or more, resulting in output of over 13,000tpd from a single kiln.

In addition to the above advantages, as discussed later, low-NO_x calciners can usefully reduce thermal NO_x generated in the kiln. The lower-temperature precalciner combustion regime has also proved very effective for firing waste fuels. The precalciner kiln has, for these reasons, been virtually the only type installed over the past 40 years.

Cyclone efficiencies rise through the preheater tower, being about 75-80 per cent at the lower stages and 95 per cent at the top stage, reducing the load on ancillary equipment for gas cleaning, dust handling and kiln feeding. Early four-stage cyclone preheaters commonly had pressure drops of around 600mm, even 700-800mm (or higher if induced-draught (ID) fans had been upgraded without modifying cyclones and ducts) and specific fuel consumptions of 850-900kcal/kg.

In the mid-1980s, FLSmidth developed low pressure drop cyclones, quickly introduced by other suppliers (Höse and Bauer, 1993). The innovation was primarily the enlargement of the inlet spiral and elimination of the horizontal inlet shelf, together with replacement of internal meal distribution splash plates by external splash boxes. The overall result was a reduction of pressure drop by 40-50 per cent. This allowed enlargement of preheaters to five or six stages with enhanced heat recovery: for example, a modern six-stage preheater can operate with a pressure drop of 450mm or less, a specific fuel consumption of 700kcal/kg and an exit gas temperature around 270 °C. LV Technology's offering is described by Jensen (2007). A popular innovative retrofit design dating from the mid-1990s is the Hurriclon from PMT-Zyklontechnik, now A TEC. This initially replaced top-stage cyclones, while there are now also designs for lower stages. The selection of the number of stages depends upon balancing construction cost against waste heat required for raw material drying, fuel cost and power cost. Whereas exhaust gases from a four-stage system can typically dry raw materials with above eight per cent moisture, the capacity for five- and six-stage systems drops to around five per cent and three per cent, respectively (Bech and Mishulovich, 2004).

In general terms, adding a fifth stage to a reasonably-efficient four-stage system should save about 25kcal/kg when considering fuel consumption alone. A sixth stage could save a further 15-20kcal/kg. However, as well as drying raw materials, even with a cyclone of advanced design, there is still extra fan power consumption to consider as well as extra capital cost.

Preheater gas ducts must have a cross-sectional area that ensures sufficient gas velocity to transport raw meal upwards whilst none drops down to the cyclone below without being fully preheated ('short-circuiting'). Many designs use a riser gas velocity of 13-15m/s, which normally gives effective lift, but this does require both good material dispersion into the duct and uniform gas velocity profile to ensure trouble-free operation.

Traditionally cyclone efficiency is increased by application of any of the following:

- increasing inlet velocity (refractory deflectors)
- gas flow smoothing at the inlet
- increasing separation velocity (increase dip tube diameter)
- increasing outlet velocity (cone the dip tube)
- increasing vortex finder (dip tube) length (providing it is not deeper than the existing inlet depth)
- installation of flap valves on the material down-pipes from the cyclones, if these do not exist (to close the free cross section not used by passing meal, so as to avoid gas bypass)
- installation of guide vanes
- installation of a sloping 'goat track' inlet
- increasing the inlet 'wrap around' from 180-270 °.

The new low pressure drop cyclones generally incorporate many such features, varying among suppliers, although Polysius (now thyssenkrupp) has produced a more novel concept, including a sloped gas inlet and roof, and very small vortex finders.

Typical exit gas temperatures, static pressures and specific fuel consumptions for modern precalciner kilns are shown in Table 5.4.

Table 5.4 Typical operating parameters for preheater stages

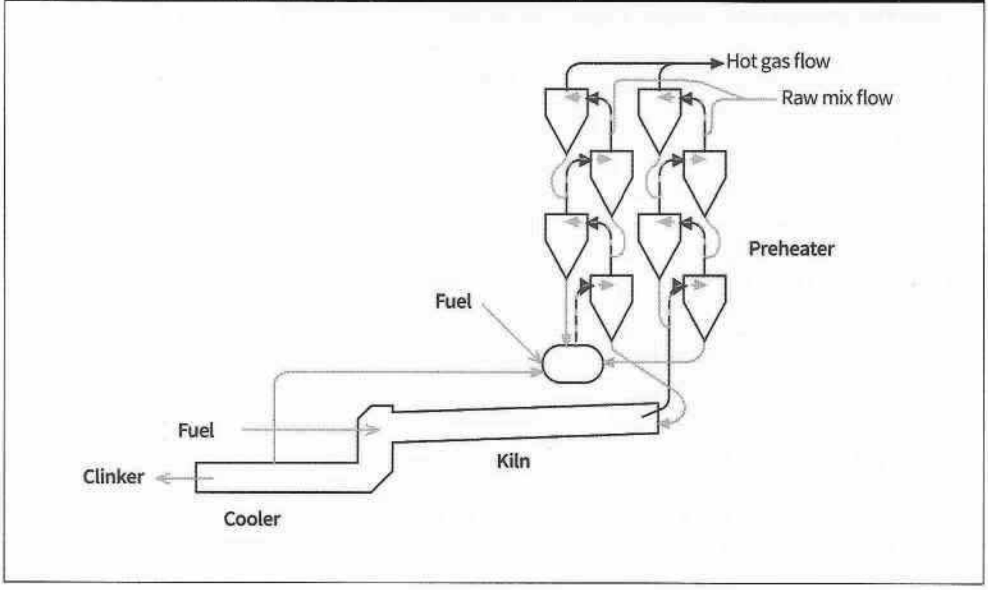
No of preheater stages	Exit gas temperature (°C)	Static pressure (mmH_2O)	Specific fuel consumption (kcal/kg)
6	260	550	750
5	320	500	775
4	350	350	800

Temperatures are 20-30 °C lower without precalciners, though usually 20-30 °C higher than the above for older systems. A typical temperature and pressure profile with cyclone efficiencies is shown in Figure 5.1.

Large modern kiln systems aim to achieve 680kcal/kg (~2.8GJ/t) and below.

Single-string (precalcining) preheaters capacities range up to about 7000tpd, limited by the largest feasible cyclone size with current refractory linings (FLSmidth, 2008). Larger kilns now have two- or even three-string preheater/precalciner towers, sometimes requiring care to balance loads but allowing unit capacities in excess of 12,000tpd and a possibility of 'turn-down' operation with one string out of service (see Figure 5.2).

Figure 5.2 Twin-string precalciner



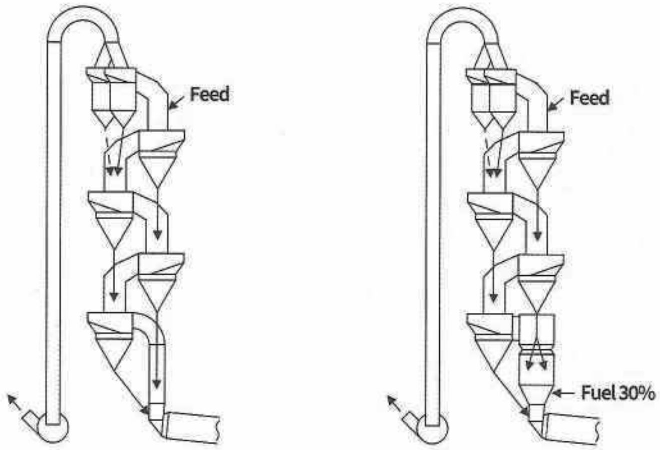
The vortex finders of lower-stage cyclones were for many years prone to collapse without replacement. However, lack of vortex finders in either of the bottom two stages is considered to reduce cyclone efficiency by 10-15 per cent and bring a fuel penalty of about 10kcal/kg clinker. During the 1990s a new segmented design in high-temperature alloy became standard (Anon, 1994 WC) and subsequently, a fibre-reinforced monolithic refractory construction was successfully tested (Gasser and Hasler, 2003). However, these were still subject to failure initially. Now, with improved designs combined with improved hanging techniques, they are likely to survive for around two years, with improvements still appearing. Any replacement project should consider the latest technology available and the possible impact of kiln volatile cycles on the preferred vortex finder composition.

In cyclone preheater kilns without precalciners, the feed is apparently 20-40 per cent calcined at the kiln inlet. Riser duct firing increases this level, and the addition of a precalciner allows over 90 per cent calcination before meal enters the kiln. Although calcination could be completed in 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 5.3. Terms frequently encountered include:

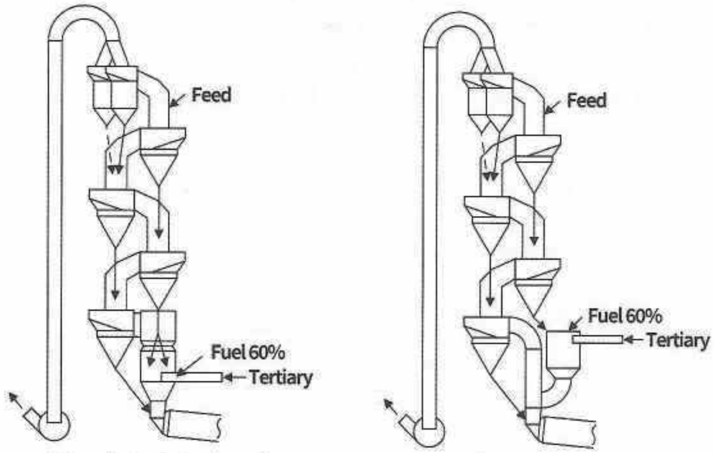
- NSP (New Suspension Preheater)
- AT (Air-Through) – using combustion air drawn through the kiln
- AS (Air-Separate) – precalciner using tertiary air
- ILC (In-Line Calciner) – AS precalciner in which kiln exhaust and tertiary air are premixed .
- SLC (Separate Line Calciner) – AS precalciner fed only with gas from the tertiary duct.
- SF (Suspension preheater with Flash Furnace) – IHI design, which is an AS/ILC system
- RSP (Reinforced Suspension Preheater) – Onoda design, which is an AS/SLC system
- MFC (Mitsubishi Fluidised-bed Calciner).

Figure 5.3 Major configurations of cyclone preheater kilns



SP – cyclone preheater with or without riser firing

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



ILC – in-line calciner fed with gas from both kiln exhaust and tertiary

SLC – separate-line calciner fed with combustion gas from tertiary only

5.3 Kiln burning

Kiln operators need to produce the maximum potential tonnage of good quality clinker at minimum energy consumption. In addition, by operating under stable conditions, mechanical stresses are reduced and brick life extended. This, in turn, contributes to achieving consistent clinker quality, maximum running hours and annual output.

Operators control internal kiln conditions using four variables: fuel rate, feed rate, kiln rpm and fan speed. Many plants link feed rate and kiln speed so that when one is altered, the other is adjusted automatically. Once the kiln speed and fan setting reach values close to their normal maxima with a reasonable feed rate, and the heat distribution within the kiln and cooler system is near optimum, kiln control mainly involves altering fuel rate. However, in practice the situation is rarely quite so simple and operators have to watch other parameters closely, especially historical trends, ie, whether the kiln is heating up or cooling down, and how rapidly.

To monitor preheater conditions, a series of continuous temperature and pressure readings must be available. A high pressure difference across a particular stage suggests that build-up has developed in ducts, or that meal is being re-entrained and carried round and round. Furthermore, loss of suction at a cyclone base, or rapidly-rising material or gas temperatures indicate that a blockage has developed in that cyclone. Gas sampling probes are normally installed at the preheater exit and kiln exit, with monitoring for O_2, CO, NO_x and perhaps SO_x. In addition, there are a number of useful indicators potentially available for estimating conditions inside the kiln, such as kiln drive power ('kiln amps'), burning zone temperature and exit gas NO_x analysis. Aspects of control are discussed below.

Kiln operation is evaluated by:

- production rate (tph clinker)
- operating hours – feed-on (h)
- involuntary downtime (h)
- total fuel rate (tph)
- 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 suction (mm H_2O)
- kiln feed-end O_2 (%)
- downcomer O_2 (%)
- kiln feed-end material
    - LoI (%)
    - SO_3 (%)
    - Cl (%)
    - K_2O (%)
    - Na_2O (%)
- kiln drive power (kW – often monitored as kiln amps).

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

These other factors include:

- primary air flow and burner tip velocity (m/s)
- specific kiln volume loading (%)
- specific heat loading of burning zone (kcal/h/m^2 of effective burning zone cross-section area)
- cooler air (Nm^3/h/m^2 grate area)
- cooler air (Nm^3/kg clinker)
- cooler (tpd clinker/m^2 grate area)
- temperature, pressure and oxygen profile of preheater (°C, Pa and per cent by volume, respectively).

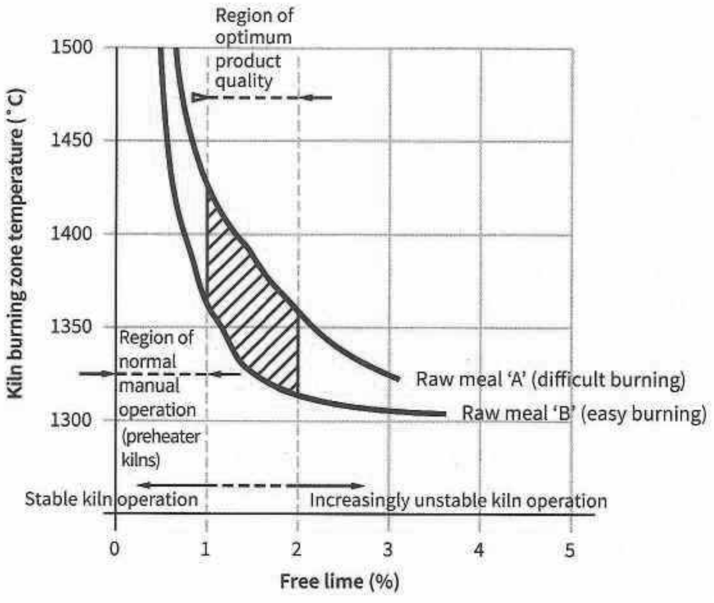
To deduce the reasons for poor performance, the most important aid is comparison data collected during periods when the plant runs well. Such periods are opportunities for staff to fully identify the conditions needed for optimum operation.

Modern kiln operation and maintenance should aim for above 90 per cent run factor (7884h/year), below three per cent lost time per month between planned outages (22h), and continuous operations exceeding 100 days (Buzzi, 2003). The best performers surpass 95 per cent run factor year upon year.

Excessive heat consumption should be investigated immediately and may indicate 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 secondary air temperature, or distortion or collapse of preheater splash-plates.

Variable or hard-burning mixes should be avoided, as higher temperatures and longer retention times involved in controlling free lime result in large alite and, worse, large belite crystals. Clinker free lime should be as high as needed to avoid the thermal inefficiency of hard burning but safely below the limit for onset of mortar expansion – typically 0.5-2 per cent. Having reached the target, free lime should, if possible, be maintained within a range of about 0.5 per cent. Variation of kiln feed rate or composition makes this control more difficult. Over-burning – a common solution to variable feed chemistry or operator circumspection – wastes fuel, stresses refractories, increases power required for cement milling and reduces cement strength. Sasaki and Ueda (1989) found a 14kcal/kg heat penalty for each per cent reduction in free lime, though other references vary. It is important to note that the relationship between free lime and burning zone temperature is far from linear (see Figure 5.4), so that it is easier to overburn clinker – for example, when raw meal composition varies – than to use a softer burning regime that risks a potentially large increase in free lime.

Figure 5.4 Relationship between clinker free lime and burning zone temperature



Obviously, if clinkering can be achieved at reduced temperature there will be energy savings. Fluxes, which reduce melting point, and mineralisers, which increase reactivity by incorporation in a solid phase, are reviewed by Hills (2002) and by Kerton (2003). The addition of up to 0.5 per cent CaF_2 in kiln feed has reduced specific fuel consumption by 25-60kcal/kg clinker (Clark, 2001), while higher levels can cause preheater build-ups and set retardation in cement.

A convenient supplement for free lime measurement is the more rapid determination of litre-weight by screening a sample of clinker ex-cooler to a selected size range, perhaps +5/-12mm, and weighing a standard 1l volume. Litre-weight is typically 1100-1300g/l (varying inversely with free lime), and the target range for each kiln should be determined with a minimum value equivalent to the established free lime upper limit. A surrogate and continuous measure can be obtained by passing a small stream of screened clinker in front of a gamma radiation source and measuring attenuation.

Secondary air temperature should be as high as possible, usually 800-1000°C, to recover the maximum heat from the cooler. Historically, secondary air temperature was difficult to measure directly without a hot-gas take-off from the hood for tertiary or coal mill air. An improved acoustic pyrometer has become available, which is a low-cost and low-maintenance instrument, integrating temperature across the hood and unaffected by entrained dust concentration (Sei Inc, 2002). This overcomes problems of background noise with a sharp, high-intensity pneumatic sound generator coupled with advanced signal detection algorithms (Kychakoff et al, 2005). Fluctuating secondary air temperature causes cycling of kiln operation. The availability of reliable signals offers potential to direct cooler grate speed control towards maintaining constant secondary air temperature, rather than just maximising clinker cooling.

Good clinker granulation is essential to optimise heat recuperation from grate coolers: this requires a sufficient liquid phase, typically 23-25 per cent, with high surface tension (Timaschev, 1980). A high alumina ratio and low alkali content increase the surface tension of the melt, while a low burning zone temperature will increase liquid viscosity and cause a small crystal size.

Many preheater kilns without precalciners benefit from feeding 10-25 per cent of total fuel to the kiln gas riser. Kiln operation is noticeably more stable and lining life is extended. This is also a useful means of consuming low-grade fuels or waste materials. The limit to riser duct fuel injection depends upon its size and consequent gas retention time, and upon fuel-air mixing characteristics. Overfuelling results in operational problems, an increase in exit gas temperature with CO detected. Auxiliary firing can also reduce costs by using cheaper fuels or by spending less on fuel preparation.

The ID fan limits the capacity of a good number of kilns, but simply increasing fan power may lead to excessive pressure drops or inadequate dust collection. Oxygen enrichment is sometimes cost effective, especially if a short-term production increase is needed to meet peak market demand. Traditionally, this was applied to the burning zone, but difficulty in maintaining lances and the danger of an overheating refractory often largely outweighed benefits. More recently, injection to the tertiary air duct of precalciner kilns has been proposed (Tseng and Lohr, 2001), involving a maintenance-free injection port and avoiding refractory stress.

A typical addition rate is two per cent of total combustion air or 10 per cent of available oxygen and some 3.5t incremental clinker are obtained per tonne of oxygen. Economics depend on the cost of cryogenic oxygen or, a more permanent system, such as the installation of an on-site vacuum swing adsorption unit. However, a more likely reason to use oxygen is to increase substitution levels of alternative fuels that are difficult to combust. Tsiliyannis (2016) presents an analysis of impacts upon both kiln output and emissions.

Precalciner kilns maintain the desired fuel split between kiln and calciner either with fixed or adjustable orifices in the riser or with tertiary air duct dampers. Frequently, dampers fail and fuel rates must be adapted to suit the resultant air flows, maintaining oxygen content at the upper-end of the kiln at the level needed to control CO levels or sulphate cycles (see below).

For kilns with grate coolers, the burner tip should usually be slightly inside the kiln – by a metre or so providing it does not suffer damage from falling clinker – avoiding overheating of the nose ring and providing a measure of surface cooling of clinker within the kiln. It may be moved back towards the (hot) plane of the kiln nose if refractories are overheated. Burners should normally be concentric with the kiln and aligned along its axis. Horizontal or off-axis orientation may be tolerable with some highly-volatile fuels but usually results in local reducing conditions and should be avoided. Clinker produced under extreme reducing conditions causes reduced cement strength and abnormal setting due to reduction of iron from the ferrite phase. Flame impingement or fuel drop-out usually exacerbates sulphate decomposition and markedly enhances associated volatile cycles. Remember, though, that kiln hood aerodynamics may require different positions and physical or CFD modelling may be needed to find a satisfactory solution.

Kiln rings are sections of heavy coating – usually in the burning zone, though sometimes near the upper end – which grow to restrict both gas and material flow and eventually force shutdown. Conversely, ring collapse causes a flush of under-burned material. Burning zone rings commonly form due to operational fluctuations, though a low coal ash-fusion temperature or high liquid phase content will increase the risk (Bhatty, 1981 and Clark, 2011). Shell scanners allow early detection and rapid reaction to counter ring growth is essential, varying kiln speed or making small movements (10cm) of the burner in and out. Rings at the back of the kiln are usually associated with volatile cycles, particularly excessive sulphur at the kiln inlet. Recurrence merits an investigation of cause(s) (Hamilton, 1997).

5.4 Kiln control

As mentioned earlier, kiln operation is a complex art, involving adjustments to kiln feed, fuel rate and ID fan speed, based on the best available signals or upon temporary surrogates when the preferred sensor is offline. Whether normal operation is manual or automated, most kilns are liable to upset periods due to ring building, coating loss, etc, and while every effort should always be made to minimise such instability, truly effective computer control must be able to cope with such situations.

Table 5.5 Control variables

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

Kiln drive amps provide an important signal for evaluating the burning zone state. The running set point will vary with the raw material composition and change with both ring formation and collapse and with clinker size. A target range should be established (depending upon raw mix and other factors) through which the amps can safely fluctuate. The absolute value is less important than the trend, which is a very good indication of burning zone length and temperature. A drastic increase followed by a fast decrease indicates loss of coating or a broken ring inside the kiln, while a drastic increase in amps could indicate a mechanical condition problem.

Other important signals are required to ensure that the system does not enter unstable or dangerous states. These are:

- secondary air temperature (if available)
- O_2 and CO at the upper end of the kiln
- pressure at kiln hood and at upper end of kiln
- temperature, pressure, O_2 and CO at preheater exit (or at stage immediately below).

For calciner systems, signals include:

- NO_x at upper end of kiln
- temperature of gas leaving stage 1 (lowest) cyclone
- tertiary air temperature.

Kiln speed should be such that volumetric loading is within the range 7-12 per cent (Section B5.10). To achieve this, feed rate is typically set at a fixed ratio to kiln speed (eg, 1.5t of kiln feed per revolution). Correlations set up at commissioning may not remain optimum and it is an important periodic process engineering task to validate the operating graphs (Clark, 1994).

Typically, cyclone preheater kilns rotate at 2-2.5rpm and have material retention times of 20-40mins. Precalciner kilns rotate at 3.5-4.5rpm with shorter retention times. Material retention in the preheater tower is 20-40s. Scheubel (1989) asserts that CaO, upon calcination, is highly reactive, and this reactivity decreases rapidly so that slow heating between 900-1300°C can result in increased heat of formation of clinker compounds. Keeping the same retention time with increasing degree of calcination at the kiln entry results in extending this transition and there is evidence that, since 1998, the introduction of short two-pier kilns with L:D ratios of 10-12 (vs 14-16 for three-pier kilns) has reduced material residence time before entering the burning zone from some 15 to six minutes with resulting improvement in clinker mineralogy and grindability. The maximum kiln diameter generally considered with current refractory lining technology is about 6.6m.

Kilns frequently operate to the limit of the ID fan. In this case, low oxygen levels in the system must be corrected by reducing both fuel and feed. Originally, preheaters were operated with kiln exit oxygen levels maintained at the point where CO was just detected (generally 1.0-1.5 per cent). However, this is rarely the optimum situation for two reasons. Firstly, before increases in CO become evident at the kiln exit, the proportion of CO in each zone of the kiln flame increases significantly, leading to longer and cooler flames, with lower peak temperatures, poor heat transfer and potentially insufficient heat to control system perturbations. Secondly, volatile circulation (discussed below) increases greatly when the flame is less oxidising, resulting in material deposits in riser ducts and a reduced area for gas flow. In fact, the optimum oxygen level for any particular kiln system depends upon the relative proportions of potentially volatile species and on the efficiency of operation of individual components, such as firing system, cooler, meal distribution, etc.

Process optimisation

Various optimising systems are available, previously called 'expert' systems. More than 75 per cent of the installations appear to be held by ABB's Linkman (Plus Expert Optimizer Version 8.0) and FLSmidth's Fuzzy Logic (ECS ProcessExpert © Version 8), where operating rules are supplemented by techniques such as neural networks, soft sensors and model predictive control (MPC) technologies (Powitec). MPC is well established for mill control and has gained credibility for kilns, often coupled with an expert system: a relative newcomer is KIMA GmbH, which earlier worked successfully on mill control. There has been a plethora of one-off, PC-based MPC systems from time to time, with market penetration unknown. The expert systems essentially seek to model the behaviour of the best kiln operators, while the MPC systems aim to model the kiln process, several of them now including 'self-learning' or 'auto-adjusting' modules. An excellent series of review papers by Haspel covers the early decades of developments in Expert Systems, the last appearing in 2009. Prerequisites for success are:

- adequate and reliable instrumentation
- a stable and uniform raw mix chemistry
- optimised combustion
- highly motivated personnel
- a schedule of checks upon the continued good performance of the system.

Typical benefits from implementing process optimisation (not all achieved simultaneously) include:

- improvements of up to five per cent in kiln output, fuel consumption and results of 28-day strength tests using the clinker product
- a saving of up to 10 per cent in power consumption for cement milling
- improvement of up to 30 per cent in refractory life, NO_x emissions ex-kiln, number of kiln stoppages and in the standard deviation of key variables.

Qualitative benefits include a greatly enhanced working environment for process operators, freeing them from routine tasks and enabling them to concentrate upon other important matters.

Operators gain useful habits from observing the optimisers and improve their manual performance, while systems require intermittent 'tuning' to cater for the effects of changed raw material and fuel properties and of repairs and adjustments to plant ancillaries - if allowed to drift too far, specialist recommissioning is needed and on-site expertise may not be available for this task. Jacob & Rytter (2014) indicate the improvements recently made by FLSmidth to achieve successful installation and future sustainability, and Teja et al (2016) provide a more detailed description. Godoy et al (2018) describe the current ABB design.

Essential rules rapidly recover the system from any drift towards extreme or dangerous conditions. Process alarms should be carefully designed and maintained. Certain alarms are critical: normal mechanical alarms, shell monitoring for refractory failure and especially explosion potential. Critical alarms should be set up so that cancellation is impossible until the problem is corrected. Interlocks are not uncommonly jumpered (either by hard-wiring or by programming) to allow maintenance to cope with a temporary abnormality or for operator convenience. Such jumpering must be strictly controlled and frequently reviewed.

Detection of CO levels above 0.5 per cent should cause alarm and shutdown of the calciner fuel supply. Above 1.2 per cent, the kiln firing system is shut down and above 2.5 per cent the ESP (if so equipped) is de-energised (VDI 2094, Germany, 2003). Flame detection is particularly vital during kiln warm-up and fuel should be shut off by interlock if the flame is lost. It should be impossible to start a kiln without activating this protection.

As discussed earlier, it is important to avoid variable or hard-burning mixes, which result in poor grindability and tend to cause dusty clinker (>2 per cent of <1mm size), which can significantly re-circulate between cooler and kiln, exacerbating 'snowman' formation and overloading the cooler dust collector. After calcination is complete, transition to melt formation should be as rapid as possible, with a short flame, minimising growth of belite and CaO crystals (Weihrauch, 1997).

It cannot be over-emphasised that kiln stability, fuel efficiency, finish grinding power consumption and cement quality all depend greatly upon the provision of kiln feed and fuel with minimal variation both of chemistry and feed rate. Process variations always have a cause and any that cannot be explained by observed feed deviation or known operational disturbance should alert staff to the possibility of faulty data, either from instruments or manual reports that fail to indicate variations. Particular areas for mistrust are:

- false instrument signals, which pressure sensors and gas sampling probes are particularly prone to
- short-term variations masked by electronically-damped signals
- feed-rate variations, especially when material is either sticky or fine and dry
- chemical variations hidden by faulty analytical methods, statistical mistreatment, or outright fraud.

Automated kiln control seems, unfortunately, to have reduced operators' habits of looking into the kiln and inspecting clinker. Modern camera systems, however, are excellent tools for observing flame shape and position of the load in the kiln (dark interface of unburned material), and clinker cooler problems that are discussed below (Prokopy, 1996). Clinker appearance can also be instructive – preferably black with surface glitter, dense but not dead-burned, dark grey cores and absence of excessive fines. Brown cores usually result from reducing conditions but can also arise from decreased permeability of clinker resulting from high belite and sulphate concentrations restricting oxidation of ferrous (Fe^{2+}) iron to ferric (Fe^{3+}) during cooling. This in turn is due to variation in kiln feed chemistry and to low volatilisation of sulphur in the kiln (Scrivener and Taylor, 1995). Other causes have been proposed (Jakobsen, 1993) and clinker colour is discussed by Clark (2006). Brown clinker is associated with increased heat consumption, reduced grindability, and with strength loss and rapid setting in mortar tests.

5.5 Volatiles in the kiln

K, Na, S and Cl can volatilise as they approach or arrive at the burning zone and are then carried back with the combustion gases. On reaching cooler regions they condense, either on feed or solid surfaces, resulting in the formation of an internal cycle or build-ups, or as fine fume, forming an external cycle that only reaches equilibrium when the total quantity entering the system is balanced by the amount leaving in clinker and non-returned dust.

Alkali sulphates entering the burning zone largely form a separate molten phase, immiscible with the main liquid phase, and start to evaporate. The level of volatiles in recirculation is much greater than their rate of introduction to the system – reaching over 200 per cent for K, Na and S, and around 5000 per cent for Cl. Under reducing conditions $CaSO_4$ reduces to $CaSO_3$ which in turn decomposes, leaving CaO.

In oxidising conditions $CaSO_4$ decomposition occurs only at much higher temperatures and $CaO:CaSO_4$ melts can form. A double sulphate can be encountered at high potassium levels, $K_2SO_4.2CaSO_4$, stable up to 1500°C and often called 'calcium langbeinite'.

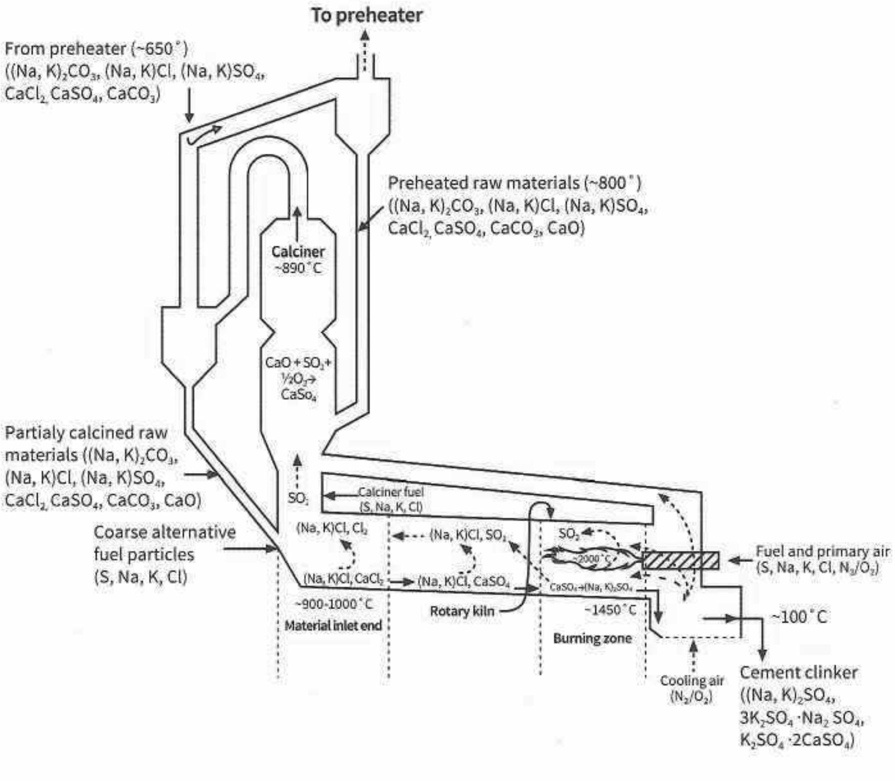
To ensure a minimum of disturbance from volatiles, most preheater kilns now operate with kiln exit oxygen levels between 2.0-2.5 per cent (typically one per cent or more higher when firing 100 per cent petcoke) and preheater exit oxygen levels some 1.0-1.5 per cent higher. These values change significantly when riser duct firing is applied, with the kiln exit level rising by about 0.5 per cent and the preheater exit level falling.

It is often instructive to calculate the potential compounds that can be formed by the cycling volatile species (KCl, NaCl, K_2SO_4, Na_2SO_4, $CaSO_4$), rather than separately listing the species Cl, SO_3, CaO, K_2O and Na_2O. It is calcium sulphate that is most prone to decompose (or fail to recombine) under slightly-reducing conditions, at temperatures as low as some 800°C and is also the most volatile, whereas in oxidising conditions, it is stable up to approximately 1100°C. Cyclic phenomena are further discussed by Kerton (1993) and by Nielsen (2012), who provides Figure 5.5.

Useful information on kiln operation can be obtained from frequent (two-hourly) analysis of clinker for SO_3 and periodic (eight-hourly) sampling of hot raw meal from the bottom cyclone stage(s) for LoI, SO_3, Cl, and alkali determination. Normal SO_3 levels (typically below one per cent in clinker and 2-3 per cent entering the kiln) should be determined and controlled by maintaining an adequate oxygen concentration in kiln exit gases. In precalciner kilns, kiln retention time and heat loading are particularly low and alkalis (K, Na) tend to pass through to clinker while the sulphate cycle at the back of the kiln is exacerbated by the alkali deficiency.

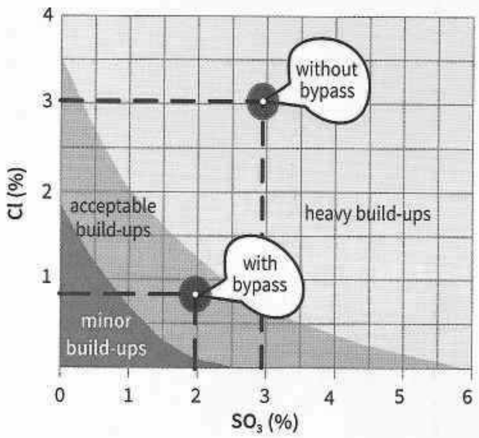
If the kiln runs too hot or the flame impinges on the load or produces local pockets of atmosphere that dissociate calcium sulphate, the sulphate cycle increases excessively until build-up or cyclone plugging occurs. This is matched by abnormally-low clinker SO_3 and free lime contents, which should be taken as a warning. Eventually, if the kiln is allowed to cool, this sulphur is released via a transient high-SO_3 clinker flush. Such variation in clinker SO_3 also causes varying grindability.

Figure 5.5 The identity of main S, Na, K and Cl-containing species in the kiln system



To minimise volatile cycles, hard-burning mixes should be avoided. To prevent problems Cl should be limited to below 1.5 per cent in hot meal entering the kiln but may be tolerated at up to two per cent in the absence of sulphur. SO_3 preferably needs to be below 2.5 per cent at the kiln entry (but may be tolerated at up to five per cent in the absence of chlorides), whilst total alkalis (expressed as equivalent Na_2O) need to be below 2.5-4.0 per cent (depending upon other factors). Binninger (2009) provides the following illustration (Figure 5.6) in his description of bypass design. Above all, the molar sulphur/alkali ratio should be maintained between 0.8-1.2 to assure formation of soft and easily-removed build-ups. Below this range build-ups based on carbonates will eventually be formed, whilst hard sulphate-based build-ups form at higher ratios.

Figure 5.6 Potential for build-ups, observed from analysis of hot meal at kiln entry, indicating the change brought about by installing a bypass at one site



There is less literature on condensation than on evaporation. On its own, KCl condenses between 800-900°C and NaCl at slightly lower temperatures. Most sulphates condense in the 900-1100°C range, probably initially mainly as alkali sulphates. Low-temperature ternary sulphate melts can form below 800°C (Ritzmann, 1971), even below 700°C when KCl is present. These melts have a poor dissolving power for the principal clinker oxides but have low viscosities and surface tension against silicates, so covering and encapsulating particles with ease. The location where calcium sulphate condenses may vary depending upon the atmosphere produced by riser duct firing or precalciner combustion.

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 cyclone plugging due to the 'stickiness' of condensing volatiles. 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-leaks. 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.

Build-up material may be manually removed during operation. Appropriate operator protection and training are mandatory, and ports should only be opened for cleaning after advising the kiln operator and ensuring that no people are below the working area (BCA, 2005). Complete cleaning is a major operation requiring shutdown and is normally effected by long air lances inserted through angled ports. High-pressure (700kg/cm^2/10,000psi) water blasters can be very effective but pose potential danger to both staff and equipment 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 that can flow like water: ports and doors below the cyclone should be closed, no one should stand in front of the hood or be allowed in the cooler (Renmer et al, 1996). In some cases, jack-hammers may be used.

Air blasters or 'cannons' provide automatic cleaning when installed permanently, with a discharge sequence optimised for build-up removal (Shelton, 2011 and Schmelzer, 2008). Some success is also claimed for sonic devices inhibiting build-up development. With 50-100 blasters in operation, the maximum tolerable sulphate concentration in kiln entry material rises to around four per cent, and can be as high as six per cent with optimised splash boxes and a 'raw meal curtain' – produced by feeding some meal from the second-lowest cyclone to the kiln feed end to quench the exit gases and volatiles by absorbing heat and calcining the material.

Refractories containing both ZrO_2 and SiC successfully reduce build-up tendency, but due to their physical properties they need careful installation (Anagnostopoulos, 2001 and Mosci, 2011). Smooth finishing of feed-chute and riser refractory is helpful. Refractory insulation should be maintained to avoid unnecessary cooling of hot surfaces.

Higher sulphate inputs

There has been success in operating with higher sulphate inputs in recent years, as required when moving towards 100 per cent petcoke utilisation. This is achieved by studying sulphate cycles for the individual kiln concerned. One key tool is reliable sampling and analysis of kiln exhaust gas. The probe at the feed end of the kiln should project inside to avoid the effect of false air in-leakage at the seal. This is a difficult location for sampling and an adequate probe is essential (Falligren, 2007), with regular reverse air-flow purging and external cooling. Air/fuel mixing is the slowest combustion step and the process gas is stratified and non-uniform in composition, as well as containing large and varying amounts of volatiles.

With a good kiln burner, process conditions can be adjusted over time to achieve a ratio of three or less – preferably two – between the sulphate concentration in kiln entry material and in clinker. As described earlier, adjustments include: correct burner alignment, adequate flame momentum (>11N/MW), correct fuel fineness, avoidance of hard burning, below 0.1 per cent carbon and one per cent chloride in meal entering the kiln, and acceptably high alkali inputs. Establishing some or all of these should enable the raw meal sulphate level to be controlled by varying kiln exit oxygen, and with this control available, the maximum-feasible fuel sulphate and 90µm residue can be established.

Changes to the kiln fuel or firing system alter the volatilisation pattern, which can settle to a new stable situation in a few hours. However, there will still be a significant quantity of condensed material in the preheater and the final equilibrium there may take tens of hours to emerge. Progress to equilibrium can be monitored via volatile concentrations in the hot meal entering the kiln.

5.6 Kiln bypass

Where volatile components are unavoidable, or alternative materials are costly, a bypass is installed to extract a proportion of kiln exhaust gas for separate quenching with 2-3 times its volume of ambient air, conditioning with water to 150-200 °C, dust collection and removal.

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

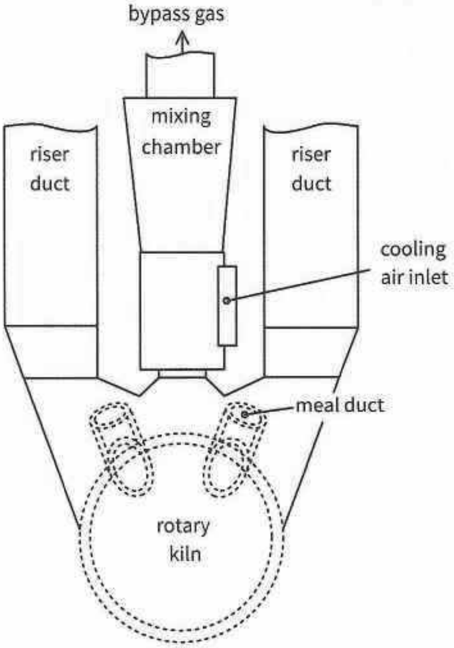
■ cyclone preheater kilns	ca 5kcal/kg x per cent bypass
■ precalciner kilns	ca 2kcal/kg x per cent bypass.

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

The criteria for bypass design and operation are complex, depending upon input concentrations of volatiles, stoichiometric balance between alkalis and sulphur, intrinsic volatility, system temperature profile, kiln retention time (shorter in precalciner kilns) and upon cement specification (Farag and Kamel, 1994; Kumar and Satija, 2009). Automated control of the bypass has been proposed based upon sampling and analysis of the hot meal (Triebel et al, 1994).

Figure 5.7 shows the kiln inlet area of a twin-string preheater with gas bypass system. The most efficient point for removing chloride is the kiln inlet chamber or the lower part of the riser duct, where flue gas chloride concentration is high. To minimise loss of feed, the bypass take-off should be at a point where the hot gas carries a minimum of dust (ECRA, 2011). In some circumstances $CaCl_2$ may be added to the feed to enhance volatilisation of alkalis and their subsequent removal via a bypass. Much cement must now meet a low-alkali specification ($Na_2O+0.658K_2O < 0.6$ per cent) while excessive SO_3 in the clinker inhibits C_3S formation. Alkali sulphates largely adhere to feed particles and do not leave via a gas bypass, requiring a hot meal bypass for removal.

Figure 5.7 Two-string cyclone preheater with a gas bypass discharge



5.7 Kiln start-up and shutdown

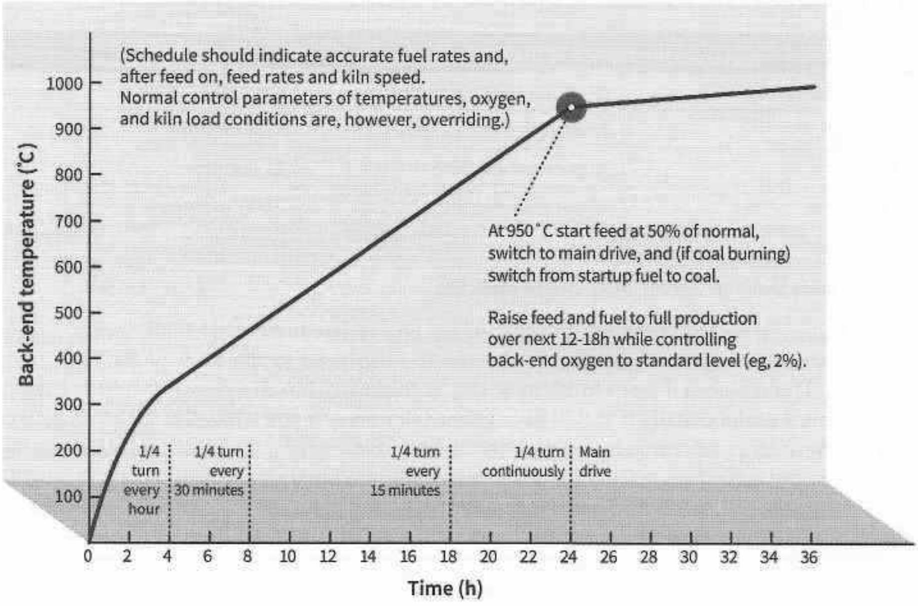
The light-up of kilns is potentially dangerous as there is insufficient temperature in the system to ensure continuous ignition. Unburned natural gas or volatile hydrocarbons from solid fuels, accumulate rapidly in the kiln and, if then re-ignited, will probably explode. It is important that ignition be achieved as soon as the fuel is injected and, if the flame fails during warm-up, the entire kiln system should be purged with an amount of air of at least five times its volume (including the dust collector), probably for some 3-5min, before re-ignition is attempted. Davies (1996) describes a simple and reliable ignition system.

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

Before and during warm-up, equipment checks should be performed to ensure that each unit is ready to operate when required. Warm-up follows agreement by production and maintenance management that all work is completed, all tools and materials have been removed, and 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 24h from ignition to feed-on but may be increased if extensive refractory work requires curing. A typical graph is shown (Figure 5.8) indicating the desired rate of increase in back-end temperature (this may also be set out in terms of fuel rate), the kiln-turning programme, the introduction of feed (usually 50 per cent of full rate), and the increase of fuel, speed and feed to normal operation should take another eight hours from feed-on. For PC systems, fuel is supplied to the calciner at the same time as, or soon after, feed-on. The fan should be operated to approximately 10 per cent 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 solid fuel at the time of feed-on. If the fuel mill uses hot gas from the cooler, there may be a delay before heat is available from the clinker.

Figure 5.8 Typical kiln warm-up schedule



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

At 950 °C start feed at 50% of normal, switch to main drive, and (if coal burning) switch from startup fuel to coal.

Raise feed and fuel to full production over next 12-18h while controlling back-end oxygen to standard level (eg, 2%).

Warm-ups from shorter stops where the kiln is still hot, say less than 24h, are conventionally cut to half the shutdown time.

Shutdown may be either:

- *emergency* – all equipment upstream of the failure must be stopped immediately (and a stand-by motor used to turn the kiln if power fails),

or

- *controlled* – the feed bin and fuel 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 flow is continued through the burner, and the kiln is rotated on a standard schedule for 12h or more with the ID fan running at reducing speed.

Suggested inching is outlined in Table 5.6.

Table 5.6 Inching schedule

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

For shutdowns of less than 24h and not involving personnel entering the kiln or preheater, heat should be retained either by stopping the ID fan immediately and shutting the preheater dampers after two hours, or (if there are no dampers) shutting down the fan after two hours, whilst continuing to turn the kiln intermittently to avoid distortion by non-uniform overheating from the feed bed. Unduly rapid cooling may damage refractories.

5.8 Kiln refractories

Some years ago, Refratechnik reported 'average best service lives' and a typical arrangement of brick types in Japanese cyclone preheater kilns (without precalciners) as shown in Table 5.7.

Table 5.7 Best service lives for Japanese preheater kiln refractories

Discharge - 1D	70-85 per cent alumina	8 months
1D - 8D	Basic, dolomite, or spinel	6-10 months
8D - 10D	70 per cent alumina	21 months
10D - feed-end	40 per cent alumina	21-37 months

(D = kiln diameter)

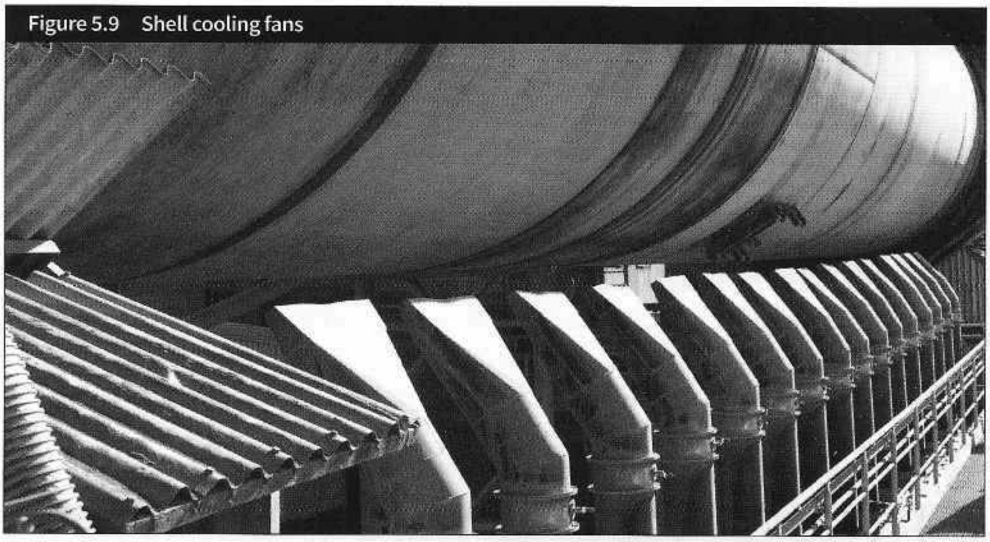
Precalciner kilns average significantly longer brick life.

A detailed historical record of refractory replacement and review thereof are important to minimise cost and service interruption. Typically, brick from the kiln nose to the back of the high-alumina section should be replaced if worn to 10cm or less in thickness, but this rule-of-thumb is subject to much variation. A useful practice is to drill through the brick every metre whenever the kiln is down and coating has been stripped, carefully locating the shell and identifying irregular circumferential wear (wider spacing and lesser frequency is adequate in low-alumina brick areas). 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 Höganäs Linometer).

A stable coating in and near the kiln burning zone is vital in protecting the kiln shell, and its extent should be observed whenever the kiln is entered. Basic brick should extend back to around the top of the coated zone. The coating location and refractory condition are usually monitored during operation with a shell scanner (Slot and Yazdi, 2004). Kiln shells should also be inspected visually, particularly under tyres where small hotspots may be concealed from the scanner. Unstable coating results from fluctuations in raw meal chemistry and in the fuel mix, poor flame conditions and from poor control of cooler operation.

Hot shell areas are often cooled by use of a fixed-fan array or movable fans directed at the area (see Figure 5.9). 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. An unpublished 1996 survey of 41 preheater and precalciner kilns operated by western European companies revealed that 80 per cent were fitted with shell cooling fans, assuring formation of a more stable coating, sometimes a little uphill of the expected region. Where 'before and after' comparison was possible, it suggested brick life increases 30-50 per cent for the cooled regions, or an average gain of 20 per cent in overall refractory consumption. Schmidt (2017) has recently described an improved system for using water sprays to cool the kiln shell.

Figure 5.9 Shell cooling fans



'Red spots', when the shell reaches incandescence, should always cause alarm and not be allowed to persist. If the spot is dull red and is in the burning zone it may be possible to re-coat the area and continue operation. Specifically, a small sharp hotspot, relating to the loss of one or two bricks can be 'repaired' by stopping the kiln for 2-5min with an air lance cooling the spot. However, response must be rapid and the long-term problems caused by shell warping should always be considered. A large red spot near a tyre should result in immediate kiln shutdown. Refractory choice is dictated by resistance to service temperature, chemical attack/spalling, thermal shock and abrasion.

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

- benchmarking refractory performance – Shepherd, 2000
- lining for critical areas – Kassau, 2001
- mechanical & thermochemical stress analysis – Klischat and Tabbert, 1998
- refractory materials –VDZ, 2009[1]
- resisting refractory wear – Schwarz, 2012
- refractory management – WC, 2013.

For plant cost tracking, both net and gross brick consumption should be recorded and used to produce internal benchmarks for best performance (REFRA.reSources, 2012). 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 markets, it may be preferable to remove and replace brick with several months of anticipated life and avoid shutting down during peak demand periods. In more uniform markets, it may be more cost effective to take relatively short outages every three or four months, allowing thinner brick to be left in the kiln, a practice observed at some plants with particularly low operating costs.

- Magnesia-spinel bricks have replaced chromium-containing basic bricks, which are no longer used due to the health hazard presented by Cr^{6+}.
- Dolomite is another important basic brick, with good affinity for burning zone coating. These present a risk of hydration, requiring special care in packaging, handling and storage.
- High-alumina bricks (70 per cent alumina) are preferred in the calcining zone, with still higher-alumina grades in the kiln discharge zone.
- Other brick types may sometimes be used to inhibit build-up formation.

Changes in fuels, feed or burning conditions affect burning zone location and at the stage where liquid phase begins to appear, uphill of the burning zone, the lining becomes more vulnerable. The more the variations, the greater care required in lining selection and installation. Factors include: variations in oxygen potential ('redox cycles'), arising with multiple fuels, which may render some brick qualities weak and friable; operation with unstable coating, exposing bricks to infiltration by clinker liquid phase; direct action of alkali chlorides and sulphates; build-up of abnormal rings, due to unbalanced sulphate modulus; and kiln shell corrosion (see later).

Clark (2018) has provided an excellent account of Refratechnik's analysis of the demands upon refractory performance that result from the properties of kiln feed, fuels and firing systems.

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 metres and the last two give brick thickness in cm. With considerable variation, installed brick thickness is related to internal kiln diameter (see Table 5.8).

Table 5.8 Examples of brick thickness for various internal kiln diameters

Internal kiln diameter (m)	Brick thickness (mm)
<4.2	180
4.2-4.7	200
4.7-5.2	225
>5.2	250

Approximate brick specific gravities are given in Table 5.9.

Table 5.9 Approximate specific gravities of various bricks

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

Then brick weight in t/m of kiln length can be calculated as follows:

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

  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, 1995/ www.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. Typically, installation after clean-out is at the rate of 0.5m/h.

In addition, monolithics, which comprise castable and plastic refractories, have various uses from the rapid gunning of large areas or dealing with complex shapes in preheater towers, clinker coolers and gas ducts, to the moulding of burner pipes and distorted kiln nose rings.

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 per cent CaO)
- low cement (1.0-2.5 per cent CaO)
- ultra-low cement (<1.0 per cent CaO).

They are mixed with water, moulded or gunned in place, and allowed to cure for 24h before being heated. The low cement types are very sensitive to water content and its addition must be strictly minimised, causing poor workability which requires intense vibration for installation. Excessive vibration, however, can cause aggregate segregation and loss of quality.

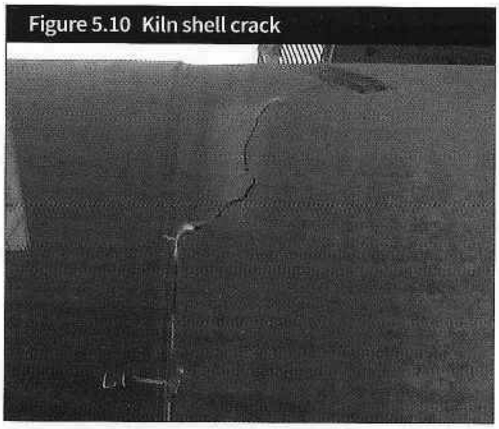
Plastics have the consistency of modelling clay and are usually rammed into place though they can be gunned if so formulated. They have to be heated at specified rates to cure, which can allow immediate kiln warm-up to begin. They are classed:

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

Provision is normally made for expansion and joints are essential when laying large sections of castable refractory depending upon anchors to accommodate expansion and contraction. While brick walls in coolers, feed hoods and firing hoods are designed to accommodate brick expansion, they can 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.

Figure 5.10 Kiln shell crack



The kiln shell should have a steel-retaining ring 0.6-1m uphill of the nose casting to resist the thrust resulting from rotation and inclination of the kiln. Experience with large numbers of kilns indicates that no additional retaining rings are necessary if the kiln is reasonably straight and does not have excessive ovality. If further rings are installed, they must be located more than one kiln diameter away from tyres because of the tendency to initiate shell cracks (see Figure 5.10). Conventional rings are rolled plates 40-50mm high and with a width slightly less than that 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, unless the relevant bricks are suitably shaped. The low ring profile results in massive loading of the contact surface with the uphill brick and consequent risk of failure. Gortan et al (1994) describe a wedge-shaped retaining ring comprising a honeycomb of special alloy steel filled with castable refractory. Coupled with specially-shaped bricks laid upon the rings, the system is claimed to produce a dramatic improvement in refractory life in difficult cases.

Precalcining plants have very large refractory-lined areas subject to abrasion due to the tertiary air line connection. To prevent premature wear in preheater cyclones, highly abrasion-resistant bricks with a minimum of five per cent Al_2O_3 must be used in the lower area and approx 60-65 per cent or 80-85 per cent in the upper area, backed by a lightweight insulating skin (Mosci, 2011). Magnesia spinel bricks are also adequate. Anchor points require particular attention, using a sufficient number of high refractory steel anchors (at least 6-9/m^2) (VDZ, 2009^2). The lower stages experience high concentrations of chlorine, potassium and sulphur, so that the refractory must be less permeable and resist alkali infiltration and attack, whilst experiencing temperatures below about 1000°C. A further requirement in this area is an ability to repel build-ups of alkali sulphates and chlorides and the thermal and mechanical shocks imparted when these are dislodged by probes and water jets. As a consequence, castables are used that contain zirconia, or silicon carbide if it is certain that water jets will not be used. As an alternative to castables, Yiakoumi et al (2015) describe bricks with good resistance to abrasion and alkali attack.

5.9 Clinker cooling

The cooler serves to cool clinker leaving the kiln at 1200°C to less than 100°C by exchange of heat with ambient air which is thereby preheated for use as combustion air. Efficient heat exchange is required to both ensure proper clinker cooling and maximum air preheat.

The preponderant class is the reciprocating grate cooler first introduced by Fuller Company in the 1930s using a sloping grate. This developed to a horizontal grate design during the next 20 years, less prone to clinker fluidisation. Soon, a 'combi' design followed, with a sloping inlet grate to avoid overheating in that zone. There is a series of up to eight under-grate compartments with two or three separate drives and fans which allow individual control of pressure and volume of injected cooling air. Under-grate pressures are around 600mm in the first compartment, reducing progressively to about 200mm in the last. The installation of tertiary air take-offs from grate coolers to supply hot combustion air to precalciners was developed in the 1970s by MHI/IHI and Babcock-Hitachi.

Frequently, grate coolers are 'upgraded' by increasing under-grate pressures and air volumes. It should be confirmed that such ad-hoc modifications do not cause significant deterioration such as clinker fluidisation on the first grate, reduced secondary air temperature and increased heat loss with cooler exhaust. Specialist suppliers, such as IKN, Fons and CPAG, now predominate in the cooler upgrade and replacement market.

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 the cooling effect. Realignment minimally involves shimming of each support girder in turn, starting from the discharge end.

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. For reciprocating grate coolers, 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 pressure varies with cooler bed depth and clinker size; keeping it constant assures a constant secondary air temperature. Maximising secondary air temperature involves optimising clinker bed depth and cooling air distribution to the recuperating zone. A common misconception is that increasing air flow to the hot end of the cooler will cool clinker rapidly and recover more useful heat. In fact, contact time between cooling air and hot clinker is reduced with consequent reduction of secondary air temperature. Also, only a constant air flow can allow the relationship between under grate pressure, bed depth and secondary air temperature to be valid. The second drive is commonly controlled at 10 per cent faster than the first, if of equal width, but the ratio may be varied as required. Airflow control maintains constant supply of air, independently from the undergrate pressure.

On traditional grate coolers, some 2-3Nm^3 of cooling air/kg clinker are required to achieve clinker discharge below 100 °C. Only approximately 0.9Nm^3/kg air is required for combustion, so the balance is exhausted and de-dusted, 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. Various air beam designs, first introduced in the 1980s, are now available providing for direct ducting of air to hollow grate support beams whence it is directed into the clinker bed more efficiently than was previously possible. This type of cooler consumes 1.5-2Nm^3/kg with a rating of 45-55t/m^2/d and bed depths of 800mm or more. The 'controlled flow grates' (CFG), fitted with air beams, 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 air flow less dependent upon bed resistance and so more controllable. The development of reliable flexible air connections between stationary and moving rows has been important. Similar grate designs are available for the non-recuperating zone but with lower resistance as required by plenum aeration; these are referred to as 'reduced fall-through' (RFT) plates (Roy and Brugan, 1995).

Broken grates can allow excessive hot clinker leakage 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 coolers are monitored by:

- secondary air temperature (°C)
- clinker discharge temperature (°C)
- discharge air temperature (°C)
- tertiary air temperature (°C)
- discharge air volume or fan power (Nm^3 or kW)
- airflow rates of each input air fan (Nm^3/h)
- pressure at each chamber (kPa)
- cooler plate temperature (°C)
- middle air temperature (where available) (°C).

Process analysis requires detailed records of individual cooler compartment under-grate air pressures, grate drive speed and power. The tuning of under-grate pressure control requires skill and experience.

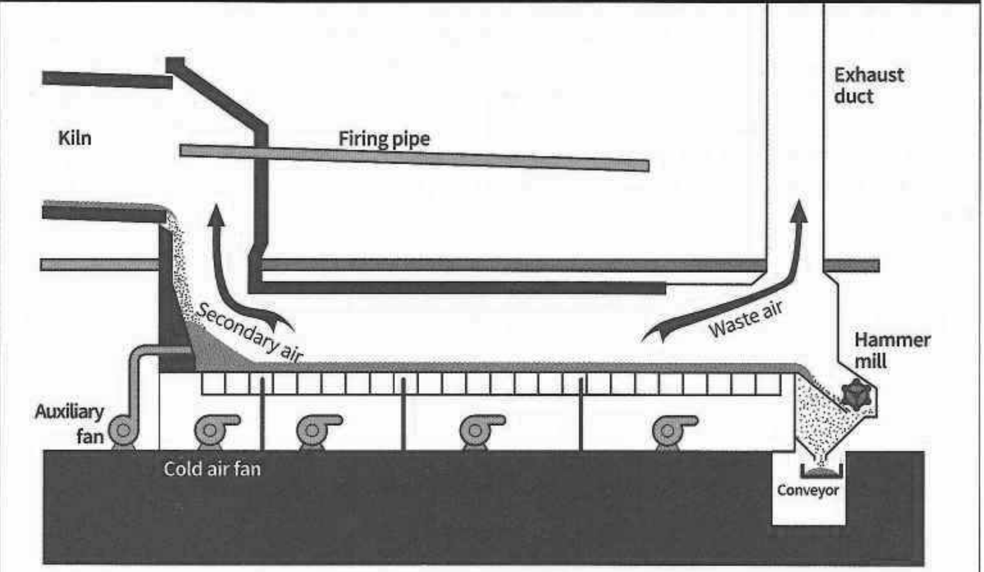
Useful grate cooler reviews include those by Wallis (2004), Lorimer (2002) and Harder (2011). Refinements include several originated by IKN and CPAG in the 1980s, as well as improved plate designs, these include pendulum frames for the moving grates, static inclined arrangements of the first 6-9 air beams to distribute clinker more evenly and a hydraulically-operated heat shield to reduce high-grade heat loss to exhaust (Von Wedel, 1995 and 2011). Coarse clinker predominates below the kiln-down, or unloaded, side of the kiln and fines tend to segregate below the kiln-up, or loaded, side. A static inlet grate section has been found particularly effective and has been incorporated to most subsequent cooler innovations.

FLS-Fuller introduced the Cross Bar Cooler® (Keefe and Bentsen, 1999) with separate conveying and aeration systems, and clinker transport effected by reciprocating pusher bars, plus an ingenious flow regulator on each grate which maintains a constant air flow through the clinker bed regardless of porosity. Cooler efficiency is excellent and much less subject to deterioration over an operating cycle than with conventional grate coolers. The supplier also altered aspects of the cooler inlet, using intermittent air blasts from below the clinker to keep material moving and reduce chances of snowman formation (Jessen et al, 2000).

Subsequently, Polysius introduced the modular Polytrack®, with a static cooler floor and above-floor transport (Morgenroth et al, 2004), Claudius Peters the ETA-Cooler® (GCL, 2004) and KHD the Pyrofloor® (Klöckner, 2006), enhanced to the PFC2 design in 2016. Progressively, too, designs have become more conservative with specific air flows of 1.8-2.2m^3/kg clinker and 40-50tpd/m^2 (Gasser and Hasler, 2003^2), with some installations running at higher loadings. Claudius Peters has described an improved design of the ETA ('η')cooler® which achieves greater bed depth via individual lane control, each with a different stroke length, removing red rivers which resulted from uneven air distribution caused by the arrival of fine, sandy clinker (Vos, 2013). Fons Technology offers its Delta Cooler, involving a shuttle floor and stepped air flow function (ICR, 2011).

As large clinker nodules and blocks of coating cannot effectively be cooled, coolers incorporate clinker breakers. Size reduction is more effective with a roll crusher at a midpoint in the cooler, but due to maintenance problems at high temperature, hammer breakers and roll crushers are more often located at the discharge.

Figure 5.11 Classic horizontal grate cooler design



Source: Dylan Moore

Visual cooler inspection is important: in particular, persistent 'red rivers' indicate problems with fine clinker segregation, grate geometry or air distribution; 'blow through' indicates excessive air flow to a particular compartment. Cooler cameras (Sagar, 2007) and thermal monitors (Idoux, 2012) are available. Trends in kiln and cooler design and operation have combined to yield a supply of generally hotter and stickier material to the (now stationary) cooler inlet and hotter secondary air flowing to the kiln, both associated with a greater tendency for material to agglomerate and/or produce build-ups. Accordingly, many cooler inlets are fitted with air blasters.

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 can be loaded to a truck to allow weight checks.

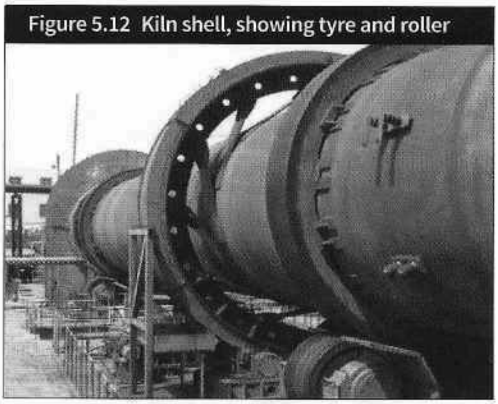
Other types of cooler are occasionally encountered, as listed in Section B5.18.

5.10 Kiln mechanical

Kiln shells provide a gas-tight support to refractory linings and impart rotary motion to convey feed and discharge clinker. Since refractory replacement is the major cause of kiln downtime in most places, it is critical to manage factors that affect refractory life. Further, if the kiln shell is designed, maintained and operated so as to maximise refractory life, then the shell itself will be preserved.

A kiln appears to be a straight, cylindrical, steel tube but does, in fact, sag between support piers and deform in cross-section, which is generally considered elliptical, with the orientation of the longer axis varying from one area to another. Near tyres ('riding rings') and in cantilevered overhanging sections, the longer axis tends to be horizontal, while between tyres it is vertical. These deflections impose significant mechanical stresses on the lining constantly varying due to rotation, adding to thermal stress. The lining absorbs stress through minute deflections within individual bricks and through relative motion between bricks. The material strength of individual bricks together with the strength of joints allows some deformation without failure. However, excessive stresses lead to failure. It is easy to conclude that conditions which exacerbate shell stresses increase risk to the kiln lining and should be avoided.

Figure 5.12 Kiln shell, showing tyre and roller



To cater for the generally large diameters of calciner kilns, "splined tyres" have been developed since the 1990s. These allow the tyre to interlock with the shell (while maintaining an air gap) in such a way that the kiln is suspended from the "3 o'clock" and "9 o'clock" positions rather than leaving its weight concentrated at the traditional "6 o'clock" position. This reduces the amount of ovality distortion by 75 per cent or more, with its added expense easily offset by savings in refractory. This has become the standard design for new installations.

Kiln vendors and providers of specialist engineering support services can detect and correct faults in various mechanical elements, but these services are not free and plant engineers need to monitor and detect non-ideal conditions before calling upon external support.

Proper maintenance of rollers and tyre pad clearances are the two primary precautions for minimising stress. Kiln support rollers bear the kiln's weight and some of the downhill thrust, turning more rapidly than kilns, and showing signs of wear earlier (Christen and Hankes, 2007). Axes of rollers on each pier should be aligned parallel to the theoretical kiln axis and slightly cut to impart an upward thrust to the tyres so that they touch lightly against their thrust bearings. The sum of all support roller thrusting should keep the kiln from contacting its thrust bearings continuously with normal and uniformly distributed internal coating. All thrust bearings are designed to support the entire kiln's downward load, but only certain types do this continuously, where hydraulic thrusters are fitted to sustain movement over a range of up to some 25mm, using a typical interval of 4-6h uphill motion and 4-6h down. Shell expansion when hot is approximately 20cm and tyres should be centred on the rollers when hot. Oil damage to grout on concrete support piers must be avoided.

Hard contact between a tyre and its retaining mechanism indicates excessive thrust, most frequently due to a slope difference between rollers and the kiln axis. Less-common causes are conical tyre or roller wear and excessive tyre pad clearance. A thrusting tyre requires one or more corrective actions before significant metal is removed from the tyre sides, making the resulting undercut difficult to remedy. Cutting support rollers is rarely an effective solution.

Excessive support roller thrusting affects bearing temperatures and wear rates. Temperature may be monitored directly via bearing metal thermocouples, or indirectly from the housing where the thrust bearing is mounted, or observing the roller shaft thrust shoulder with an infrared pyrometer directed through the housing hand-hole. Excessive thrusting is easily detected by carefully rubbing a finger across each roller surface (when cool), noting if it feels smooth or rough while wiping uphill or downhill, termed 'checking for fish-scales'.

Kiln seals

Kiln seals exclude false air, with differential pressure at the feed end of 4-5cm WG and 1cm or less at the discharge end. False air at the discharge seal displaces hot (900-1000 °C) secondary air and lowers fuel efficiency. False air at the feed end reduces production capacity and efficiency on fan-limited kilns and, if affecting volatile condensation, can cause serious build-ups which interfere with material and gas flow.

Reliable seal designs are usually simple and air cooled, and work reasonably well if properly maintained. The most common replacement utilises flexible sheet metal plates arrayed around a steel ring mounted to the shell (Geiger, 1995). At the discharge end, the most common cause of poor sealing is overheating of the sheet metal plates, which may need to be protected from radiant heat, with appropriate shields and cooling air flow.

The most common cause of failure is material loss resulting from unavoidable contact and wear between stationary and rotating components. Regular inspection and opportune maintenance are essential, and the rotating component run-out must be held within the seal's capability.

Kiln shell

Design historically considered kiln shells as beams of cylindrical cross-section. Effective designs locate tyres to balance the uphill and downhill loads on mid-kiln tyres and yield shell overhangs between one and two kiln diameters from feed and discharge end tyres. Shell thickness must maintain calculated material stress levels well within the steel's capabilities and manufacturers utilise historically-proven stress limits that accommodate variation from design assumptions.

Structural carbon steel plate ('mild steel') shells lose significant strength above 400-450 °C. At 500 °C most steels lose about half their ambient temperature strength and shell temperatures require continuous monitoring, with recordings and alarms. Temperatures above 350 °C should be monitored closely and corrective action should commence before 500 °C is reached. Continued operation at or above 350 °C generally causes permanent shell deformation or crack initiation. Infrared imaging systems are used increasingly to monitor this and provide information on refractory condition, coating thickness and tyre creep.

Creep – the relative movement of shell and tyre – can be measured manually by marking both with chalk and observing displacement after one or more revolutions, or more conveniently monitored automatically by most shell scanners. Creep varies with temperature and should never be zero, but if above 1-2cm/revolution, it should be corrected by shimming tyre pads.

Generally there are two shell thicknesses at each tyre. Thicker plate for the course directly under a tyre and thinner flanking plates uphill and downhill. Plate between tyre sections is even thinner and weld joints between different thicknesses require attention. 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 thinner. The most common location for shell cracking is at the transition between flanking plate and thinner shell plate, generally at the toe of the weld joint on the thin side. It is often acceptable simply to mark the extent of these cracks while continuing to operate until an opportune shutdown. Drilling a 'stopper' hole at the end of a crack is common but will generally not be effective unless at least 25mm in diameter. The crack should be removed by burning or gouging a groove from the outside with sides making about a 60° angle. Magnetic particle or ultrasonic inspection should determine that the entire crack has been removed. Welding should be performed with appropriate filler metals laid down in straight parallel beads, each overlapping the one below by about half of its width to provide heat to relieve weld shrinkage stresses in the underlying bead. There is no need to stress-relieve after welding, but 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-welded as described.

Apart from overheating, the most common cause of 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 clearance, which should be measured with the 'Obourg Pen Tester' device (Figure 10 in Chapman, 1985) to trace 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, to limit negative impacts upon refractory life and prevent excessive shell stresses, corrective action should be considered when percentage tyre ovality exceeds 0.15-0.2 per cent relative to kiln internal diameter or when shell ovality exceeds 0.3-0.5 per cent (the appropriate value varying with kiln diameter through the range 3.5-6.0m). 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 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.

Shell corrosion is not normally a serious problem with conventional fuels unless high levels of sulphur or chloride are present. Corrosion is reviewed by Mosci (2004). The main causes are alternate oxidation at high temperatures and acidic reaction at low temperatures during repair stops, largely due to the presence of oxides, chlorides and sulphides which can diffuse through the refractory at high temperature. Note that the use of some alternative fuels can pose challenges. The use of high-grade magnesia spinel bricks and SiC castables has increased in recent years because these can reduce or prevent salt infiltration.

Kiln drives

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 supplier's recommendations for viscosity at operating temperature. Lockett (2008) discusses appropriate modern tyre lubricants. At least three temperature measurements across the gear face should be recorded monthly using a pyrometer. A properly-aligned gear set will vary no more than 5°C across the gear mesh. Corrective action upon alignment should be considered when temperature variation exceeds 10°C.

The tip-to-root clearance between gear and pinion should be measured routinely, as well as after any refractory failure which may have caused 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 (2004).

Over 80 per cent of drive power elevates 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 rated current and torque to overcome inertia and static friction when starting. Large kilns typically use a pair of motors.

Precalciner technology and the desire to minimise material retention time between calcination and sintering have produced a trend to increased kiln rotation speed. As older kilns are upgraded it is common for drive speeds to rise via:

- gear reducer ratio changes
- weakening of DC motor field strength
- utilising higher than rated motor frequency on AC 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 speed which will seriously exacerbate control problems. Thus, a marginal drive motor should be replaced before attempting to increase kiln speed.

Kiln alignment - Most designers limit bearing pressure between tyre and support rollers to around 35kg/cm^2, though some designs allow for higher pressures with self-aligning bearings. Allowable 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 support roller journal rotation is usually below 0.125mm thick so that anything that impacts this value is of consequence. It is essential to inspect the shaft surface routinely during shutdowns and remachine before excessive circumferential scoring may cause penetration of the oil film, temperature elevation and bearing failure.

Kiln designs make generous allowances for weight of 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 lubricant. Variations in coating pattern resulting in non-uniform shell temperature can cause temporary changes in the theoretical axis of rotation. These temporary, process-induced bends, combined with other permanent deformations can cause excessive bearing pressures and subsequent failures.

The lowest-risk operating state is the one that allows the kiln to accommodate as much transient shell misalignment as possible. To achieve this, the kiln shell's theoretical axis must be measured during operation. This is done somewhat indirectly by measuring the position of the tyres, assuming them and the shell to be perfectly round, and calculating the location of the axis at each tyre. 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 shell axis. If the kiln is relatively straight, the location of each tyre's axis should lie within about 3mm left or right of the theoretical kiln axis when viewed from above. The vertical locations of the tyre axes relative to the theoretical straight line kiln axis may vary significantly from pier to pier. 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 to avoid excessive bearing pressures. The most effective means to make the final alignment adjustments is to use an ovality gauge to measure 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 support roller loading. Hot kiln alignment is described by Deventer (2004).

New installations now prefer twin kiln supports, requiring lower investment and providing a determined load distribution. Here, it is possible to avoid the girth drive by using the kiln rollers to transfer torque to the tyre. 'Gearless' or 'friction' drives have been introduced, initially by Polysius (POLRO) and FLSmidth-Fuller (ROTAX). Use of hydraulic drives for two rollers provides high torque and smooth and quiet operation, at some extra cost. Splined tyre fixings are an integral part of gearless drive systems and provide much better support of the shell, which is laterally suspended in adequately designed carrying bars. This reduces ovality, yielding noticeably better brick life.

5.11 Emergency power

Electricity supply is always prone to interruption, resulting in significant damage unless precautions are taken. The best system is a standby diesel generator (of ~1MW) that starts automatically upon failure of the main supply. This 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
- packhouse and load-out.

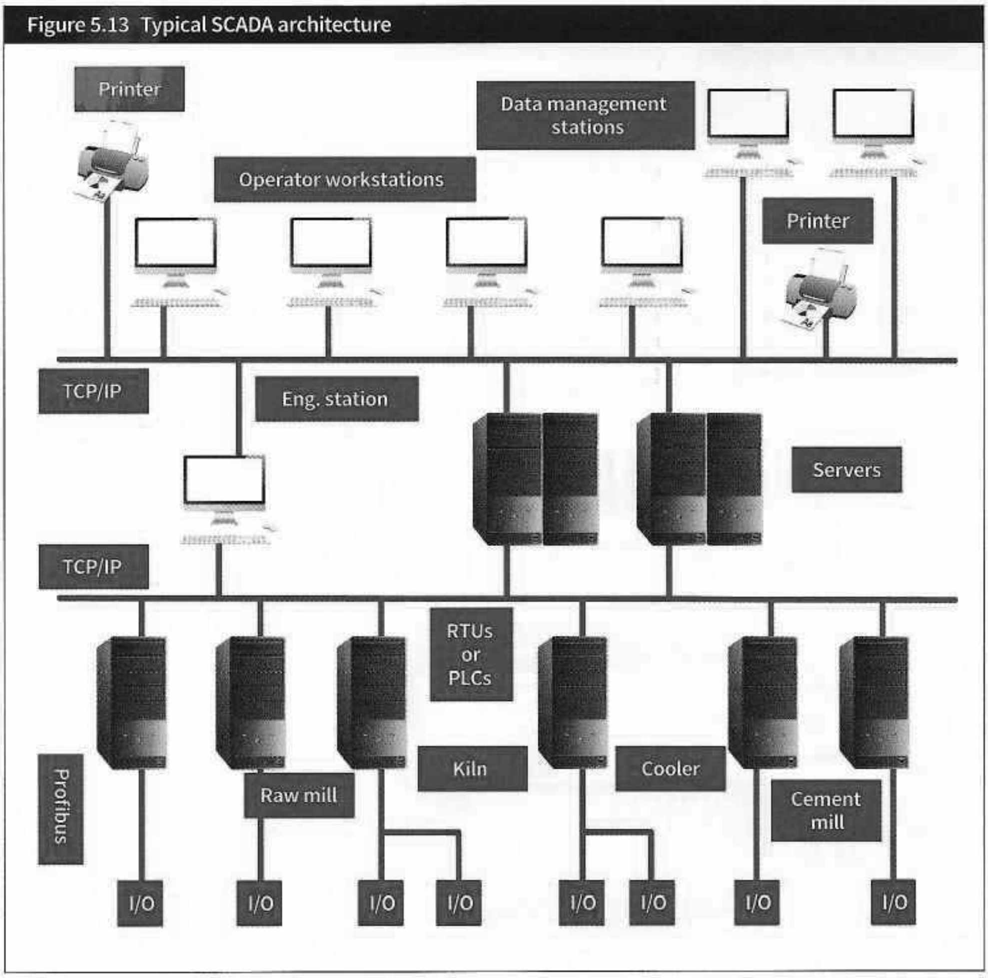
Alternatively, at the very least, there should be a petrol engine to drive the kiln at low speed, say 0.2rpm. Certain manual procedures are then essential, such as withdrawing the burner pipe from the kiln hood and opening doors at the top of the preheater if there are no automatic vents.

The more reliable the main power supply and the more infrequent the use of the emergency system, the more important the maintenance of the standby equipment becomes, allied with rehearsal of the procedures. Serious distortion of the kiln shell will result if it is not turned within 15-30 minutes of a crash stop. Kilns must be rotated on a regular schedule after a stop to ensure uniform cooling, as the feed bed and the refractory beneath it take more time to cool down than the portion of refractory wall exposed to the kiln gases.

5.12 Plant control systems

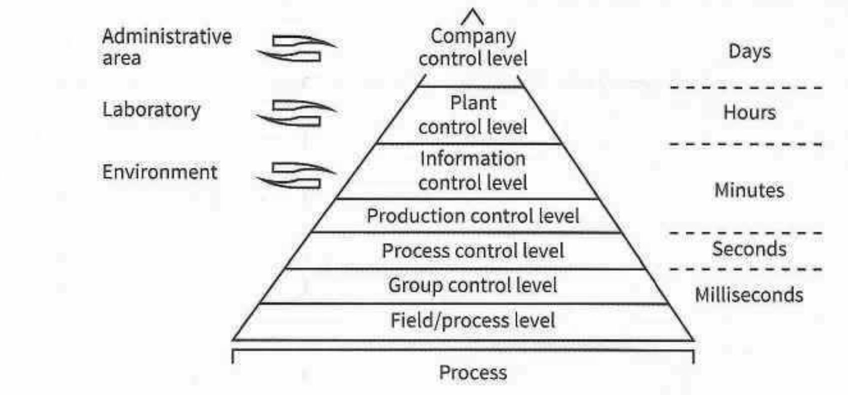
For many years the structure of cement plant control systems has been based on a plant-wide distributed control system (DCS), in which programmable logic controllers (PLCs) are distributed throughout the process, each controlling a certain section. The entire system may be networked for communication and monitoring. The fully-developed DCS comprises supervisory control and data acquisition (SCADA) software plus human machine interfaces (HMIs - operator and engineer workstations), which are usually PCs displaying process flowsheets, operating conditions and logic diagrams, along with data trending for process analysis and troubleshooting. Typical architecture of a SCADA system for a cement plant consists of input/output signal hardware, controllers, HMIs, networks, communication, database server and software (Figure 5.13). The Master Station refers to the servers and software responsible for communicating with the field equipment (remote terminal units - RTUs - and PLCs) and then to HMI software running on workstations in the central control room. Due to advances in computer capability, the software is developing rapidly. The applications cover many areas, such as production management, model-based control, real-time optimisation, plant asset management and real-time performance management tools.

Figure 5.13 Typical SCADA architecture



Communication between HMI, Master Station and controllers uses an open protocol such as Ethernet or TCP/IP. DCS systems invariably come from major system suppliers such as ABB, Honeywell, etc and are well supported internationally, while SCADA/PLC systems have typically involved hardware supplied from a manufacturer such as Rockwell Automation or Mitsubishi, with the system configuration developed through a software house or system builder.

Figure 5.14 Structure of the process control and information systems (CIM pyramid)



The process conditions for each production section are monitored and the resulting signals transmitted by wire or optical cable to each corresponding RTU or PLC via I/O (input/output) devices. The control signal generated from the RTU or PLC then returns through I/O to each execution device. The RTU or PLC performs PID control loop functions, data compiling and formatting, manages alarm logic, and creates alarm signal and interlocking control. The cost of the network is proportional to the number of I/O points.

As the reliability of the SCADA system is vital, back-up hardware and redundant signal transmission systems are usually designed in. Both the rapid advances of control and communication technologies and the globalisation of the cement industry serve to drive adoption open software platforms and communication protocols. These allow easy system expansion and networking, and linking to company information management systems (Garza et al, 20031 and 20032). On the other hand, complexity renders systems more vulnerable to cyberterrorism and a robust security shield becomes essential. Experience shows that the most vulnerable area of control still remains with the initial measuring devices that are placed in the aggressive conditions of the process stream.

Steininger (2013) reviews software innovations, pointing out criteria for evaluating a past or pending control system purchase, and noting that, with standardised interfaces now proven, it is now much more bearable to combine elements from different vendors.

6. Cement milling

Finish milling is the grinding together of clinker with some 3-5 per cent gypsum, natural or synthetic, for set control, and quite often other components: blastfurnace slag, pozzolan, silica fume, limestone or pulverised fuel ash ('pfa' – also known as 'fly ash'). The process and equipment circuits are similar to those mentioned in Chapter 3 for dry raw milling – ball mills, vertical roller mills, roll presses, Horomill – in combination with separators and other optional ancillaries. The finished product is collected by filters for transport to storage, prior to dispatch.

6.1 Storage of clinker and other components

Clinker in transit to the finish mills should be cooler than 100°C. It is, however, liable to be at a considerably higher temperature, at least intermittently, at the cooler discharge. The standard transport system is a steel deep-pan conveyor which can handle high-temperature clinker and elevate it at up to a 45° slope to the top of the clinker store without intermediate transfer.

In combination with cement storage, there should be adequate storage capacity for clinker and other cement components to maintain deliveries during maintenance shutdowns and, if the market is cyclical, to bridge low and high shipping periods. Total clinker capacity should be equivalent to at least 14 days of kiln production, with separate storage for different types of clinker (if produced) and for off-specification clinker. Storage serves also to homogenise the material.

Non-clinker mineral additions require separate installations for storage, pre-blending, crushing, drying and feeding. Materials such as granulated blastfurnace slag (GBS) with a high moisture content may require drying, preferably using waste heat.

High free-lime clinker must be blended into mill feed with circumspection to ensure that the resulting cement is not expansive. A maximum composite free lime for the cement should be established and used for control. Where an autoclave expansion test is employed (ASTM C151), the control target will need to be reduced for clinker with above four per cent MgO.

The need for large-capacity (up to 200,000t) stores results 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 (1999) reviews various designs. Such storage has cost as little as US$33/t (Conroy, 2006). Montoya (2004) reported a cost of US$50-60/t for silo construction, but costs are higher if foundations require the use of piling. Conical buildings with diameters of 80m or more are typical. Some installations suffer from an appreciable 'dead' volume. Flat-bottomed circular storage with gravity discharge may allow up to 80 per cent recovery without manual assistance, while mole reclaimers can recover 100 per cent. To avoid short-circuiting, stores should not be filled and discharged at the same time unless there are multiple extraction points.

Cylindrical silos may be preferred where space is at a premium, despite a need for costly foundations on poor soils. For example, 46,000t of clinker could be stored in a silo of ϕ25m x 85m height, fed by a sturdy inclined conveyor, whereas a corresponding conical store would be ϕ75m x 26m high. Clinker should be extracted evenly from four silo quadrants so that the load does not bear unevenly on one section of the wall.

There are obvious economies of scale for clinker storage and single silos of up to 150,000t capacity are available. Certain designs leave substantial quantities of dead material, and are better initially pre-filled with limestone and run down to refusal to avoid a perpetual inventory of high-value clinker. It is inadvisable to habitually run clinker silos down to the steel cone as abrasion will eventually cause failure. Loading of hot (>150°C) clinker to concrete silos should be avoided, as spalling and delamination will progressively weaken the walls.

Outside stockpiling of clinker or granulated slag is not usually worthwhile if subject to rainfall. It is also prohibited in some locations. Partial hydration of clinker before grinding seriously reduces concrete strength (unless ground to a significantly higher surface area) and increases setting time. Recovered clinker should not add more than one per cent to cement LoI and can seldom exceed 10 per cent of mill feed, while higher costs are incurred due to added handling and milling to an increased Blaine fineness (specific surface area – cm^2/g), whilst maintaining the 45μm residue at the normal level. If materials must be put outside, it is advisable first to screen out fines (below 5mm) to minimise fugitive dust and hydration.

6.2 Cement milling

In recent years there have been numerous plant capacity increases as well as the construction of new lines. Additional capacity has often been achieved by adding pre-grinding equipment, principally roll presses, to existing ball mills – though, in practice, the added complexity can increase stoppages. New lines have increasingly preferred vertical roller mills due to improved reliability, lower specific power consumption, and their ability to grind blastfurnace slag both separately and in composite cements. Cement performance in strength tests and its consistency have increased over the years. Progressively, the fineness achievable for Portland cement and slag has been increased to above $6000cm^2/g$ and it has been demonstrated that particle size distribution (PSD), and cement performance of ball mill and roller mill cements are compatible (Schafer, 2003). Ball mills, however, are still the most common in existing plants, and are included here (see Section 3.1 for some additional material). Europe's largest ball mill entered service at Gorazdze, Poland, in 2012, and is rated at 1.4Mta (236tph).

Finish grinding involves the largest unit consumption of power in cement manufacture and should be optimised. However, it must be recognised that clinker grindability is largely governed by its chemistry and burning conditions, so that kiln and finish mill must be considered together. Various correlations with microstructure have been reported and can provide clues about variations in grindability of clinker from a given kiln (Harrisson, 2013).

The principal cause of hard grinding is excessive belite (C_2S) in clinker. Conversion of belite to alite (C_3S) in the kiln is poor if either raw mix silica is present as coarse quartz, the LSF is high (97-99 per cent), the liquid phase is relatively high (>24 per cent), or the silica ratio is relatively low (<2.5). It has also been noted that higher K_2O (>0.5 per cent) correlates with easier grinding (though this may be due to the presence of clay in raw meal, easing combinability). It is also important to avoid large alite and belite crystals (Hills, 2007). There are suggestions that belite formation in clusters makes clinker harder to grind compared to well-distributed individual crystals and, further, that the presence of various trace elements can sometimes ease grinding. Higher free-lime gives easier grinding (increasing free lime by 0.2 per cent reduces mill power requirement by around one per cent).

Higher clinker SO_3 gives harder grinding because less addition of easily-ground gypsum is permissible. One per cent extra gypsum yields some $12m^2/kg$ extra surface area at constant kWh/t and limestone yields about half this amount (Smith and Sumner, 2012). Any reduction in power available from altering raw mix design and burning practice must be considered in the context of overall manufacturing cost and cement quality. Pragmatically, cement grindability (with gypsum and/or grinding aid addition) is more important to a cement plant operator than pure clinker grindability, given that gypsum also acts as a grinding aid, reducing particle agglomeration and mill coating. Practical evaluation of such test results also needs to consider power consumption when producing cement to an appropriate and marketable strength class via sufficiently fine grinding.

Selection and optimisation of mill circuits require consideration of each step in the process: drying, coarse grinding, fine grinding, product cooling and mill ventilation, classification, mill operation and control.

Setting up a stand-alone grinding plant using imported clinker can be an attractive option, involving only about 10 per cent of the capital cost of an integrated plant, especially if there are reliable local supplies of mineral additives to enhance cement output. Several manufacturers offer modular units with proven reliability.

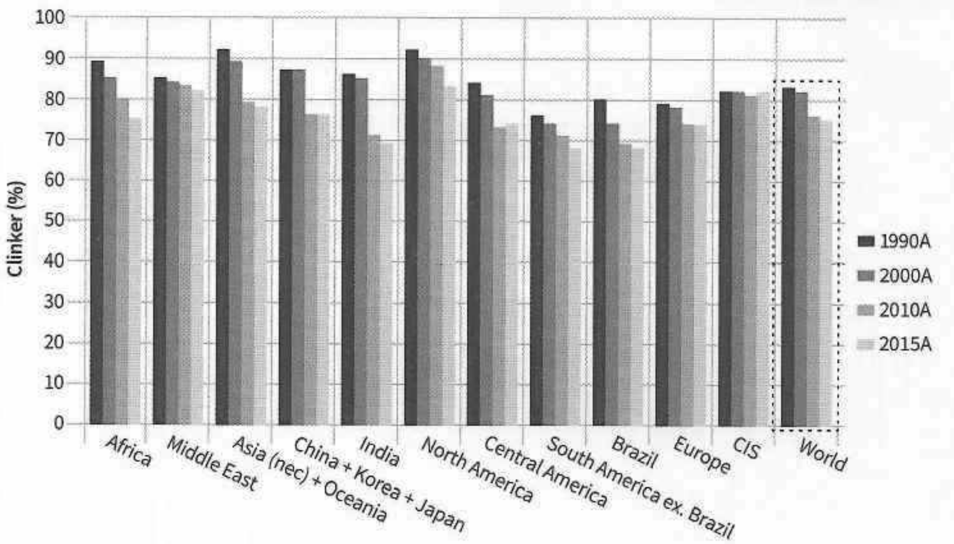
Mineral additives

Typical clinker substitutes are granulated blastfurnace slag (GBS), natural pozzolans, fly ash, silica fume and limestone. Properties vary a lot as far as reactivity (ie, clinker replacement potential), grindability, raw feed size, humidity and abrasiveness are concerned. More information can be found in Chapter 7.

Components with widely different grindabilities are best ground separately, and composite cements may also be conveniently and flexibly produced by use of a continuous flow-through mixing system, suitably located for use with a multi-cell cement storage silo with accurate dosing systems. Mixer capacities range up to some 250-450m^3/h and blending efficiency is claimed to be superior to that available when intergrinding in a vertical roller mill with a low residence time.

In 2015 the clinker-to-cement ratio varied around the world, with the lowest clinker factor in India, Latin America and Europe, and the highest in North America, the CIS region, and the Middle East (WBCSD CSI GNR data, 2015).

Figure 6.1 Regional cement-to-clinker ratio, 1996-2015



Source: WBCSD CSI GNR

Blastfurnace slag (granulated), when ground together with clinker, dominates the coarser fraction, especially in open-circuit mills. In closed-circuit systems, ground slag still has a steeper particle size distribution than clinker fines. Consequently, mill systems directing more energy to fine grinding are beneficial with respect to late strength and/or energy consumption. The benefits of combined grinding are good product homogenisation, reduced tendency to agglomeration and coating in the mill, and the ability to use the heat from clinker to dry the slag. Separate grinding uses less energy and the fineness of each component can be optimised. Moreover, with adequate storage and blending facilities, a range of products can be offered. In some countries separately-ground slag is sold directly to concrete producers. All types of mill are used for separate slag grinding. Woywadt (2013) describes Gebr Pfeiffer's Multidrive MVR vertical roller mill (VRM) for this application.

Natural pozzolans may, in some cases, require crushing before feeding to the cement milling process. Most are microporous and easy to grind, often quite abrasive, and may have a moisture content requiring drying during grinding. Combined grinding is expected to yield very fine pozzolan and comparatively coarse clinker, so exhibiting higher water demand than a blend of separately-ground materials. Separate grinding and subsequent blending may also be advantageous with regard to workability and strength development for the composite cement. Due to pozzolan quality variations, a safety margin must be respected for the clinker replacement level.

Fly ash is obtained in large quantities from coal-fired power generation. It is principally an aluminosilicate formed of small glassy spheres. This results in very low bulk density (~0.8t/m^3) which impacts transportation and storage costs. A small fraction of ash particles are hollow and fineness usually varies between 200-500m^2/kg (or 45μm residue of 10-40 per cent), similar to that of Portland cement. Ash is usually available as a dry powder suitable for pneumatic transport and is often introduced to blended cements with little or no grinding. Grinding initially results in crushing of the hollow and porous particles before further energy-intensive grinding of dense glassy material begins. The most favourable combination of product quality and grinding energy is usually obtained with combined grinding. In many plants partial combined grinding is implemented by feeding ash to the separator – irrespective of the type of grinding circuit – and only the coarse particles are returned to the mill for grinding.

Silica fume, or 'microsilica', may be similarly processed, being extremely fine, or may be blended with previously-ground OPC.

Limestone – arguably a more sustainable material than clinker – can participate in hydration reactions, but 28-day strengths for mortar and concrete can be lower. The loss in strength can be offset either by a strength increase resulting from the narrower particle size distribution of the interground clinker or by overall finer grinding of the blend at constant kWh/t. For limestone levels between 5-10 per cent, strengths are not normally reduced. In addition, in some mill systems yielding narrower PSDs, inclusion of up to five per cent limestone broadens the PSD and can offset problems of undesired mortar or concrete bleeding and delayed initial set.

(Section B4 provides some supporting data for this chapter.)

Ball mills

In existing plants two-compartment closed-circuit ball mills with separators predominate, although VRMs are preferred for new plants. The first compartment, typically occupying 25-35 per cent of the mill length (less for high-surface area products), is charged with large media of ϕ60-90mm, suitable for grinding raw clinker of up to 30mm in size by impact, employing internal lifting liners that protect the mill shell. A diaphragm separates the two compartments, allowing passage of clinker below 2-4mm in size from the first to the second. The second compartment more efficiently achieves fine grinding with smaller balls in a closer size range (20-30mm) – or even mono-sized (Marsay and Sumner, 2013).

With a range of ball sizes in the second compartment, classifying liners are used to retain larger balls close to the diaphragm, whilst simple wave liners are appropriate when using a single ball size.

Ball mills are designated by internal shell diameter, length and connected drive power. Media quality is important for consistent and efficient performance. High-chrome alloys are now almost exclusively used for balls and liners, and ball usage should not exceed 50g/t cement. 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 annually, to maintain the optimum size profile and remove tramp metal.

The quantity of balls, the type and condition of the shell liners, and the mill speed determine the power drawn by the mill (Christian Pfeiffer, 2011). Mills typically operate at about 75 per cent of critical speed (at which centrifugal force just holds the charge to the shell during rotation), and 25-35 per cent volumetric charge loading. Table 6.1 shows general guidelines for mill operation.

Table 6.1 Guidelines for mill operation

Two-compartment closed-circuit ball mill		
	First compartment	Second compartment
Power draw (kWh/t cement)	9-10	24-25
Liner type	Continuous lifting	Wave or classifying
Charge specific surface (m^2/t charge)	9.5-10.5	35-38
Closed-circuit ball mill preceded by roll press		
Power draw (kWh/t cement)	5-6.5	22-25
Liner type	Continuous lifting	Wave or classifying
Charge specific surface (m^2/t charge)	12-14	38-40

Ventilation of mills has several important functions: removing water evaporated from moist feed or from gypsum dehydration, or injected to cool the mill; transporting material within and out of the mill, especially fine product, and removing heat due to hot clinker and mill power dissipation. A ball mill cannot cope with feed moisture levels above about 3-4 per cent unless fitted with a drying chamber, and as a rule of thumb, every one per cent increase above 0.5 per cent increases energy consumption by more than 10 per cent, especially at high fineness levels.

Airflow has tended to increase since about the 1970s. Normal air velocity is around 0.9-1.5m/s, averaged across the open mill cross-section above the ball charge. Fully air-swept mills, where all the mill discharge is transported to the separator by air and coarse separator rejects are returned to the mill inlet, operate with 5-6m/s. Static pressure drop across the mill during operation is a good indirect measure of sweep.

Cement discharge temperature is maintained between about 90-115°C. If too cool, gypsum is insufficiently dehydrated and product strength is lost. If too hot, excessive gypsum dehydration occurs resulting in false set (see later). Water sprays are used to control temperature, but it is essential that water is evaporated and does not cause cement hydration or build-up on liners or screens. It is easier to spray water into the first compartment but, where maximum cooling is required, it is more effective to spray at the diaphragm, concurrently into the second compartment.

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 down on load after a steady state has been maintained for 8-10h. The fan should stop immediately to avoid sweeping away fines and the mill positioned so that the doors can be opened. An audit of the mill condition involves taking samples of material and media along its length to confirm that there is a progressive reduction of size. Media grading is also determined.

The detailed procedure for the complete mill circuit is described by Strohman (2011).

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

Vertical roller mills

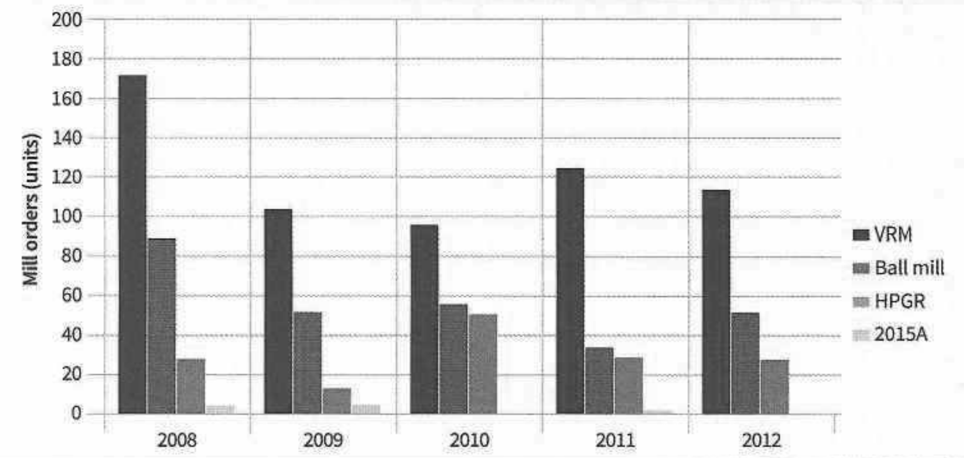
Vertical roller mills (VRMs) were originally used for relatively-soft materials such as raw meal and coal, but in recent years, improved metallurgy for wear parts and modified roll configurations to allow handling of fine material have led to much increased use for cement and slag (Schafer, 2003). Take-up for this purpose by major equipment manufacturers was initially slow until it could be demonstrated that product quality need be little different to that produced by ball milling. Comminution conditions in a VRM are very different to those in a ball mill, as shown in Table 6.2 (Fahrland and Zysk, 2013).

Table 6.2 Comparative performance parameters for the two systems when used for grinding cement

Characteristics	Ball mill (closed-circuit)	Vertical roller mill
Comminution by	Impact and attrition	Pressure and shear forces
Residence time (min)	20-30	<1
Crushings before separation	-	1-3
Circulation factor	2-3	6-20
Wear rate (g/t)	-50	3-6

This originally resulted in steeper PSDs, less-rounded particle shapes and less gypsum dehydration. Producers were concerned that, although cements met national standards, the changed properties and performance would be detected and disliked by major customers. These misgivings were overcome during the 1990s with adjustments to design and operation, and VRMs now compete successfully with other grinding equipment, producing cements with similar levels of water demand, setting times and compressive strength (Mutter, 2013). In addition, when a variety of cements are produced, they are more flexible than ball mills. The largest VRM on order will grind cement at up to 640tph at 3800cm^2/g (or 450tph at some 4400cm^2/g), using six single drive rollers. The flexibility provided by a single VRM for

Figure 6.2 Types of cement mill ordered

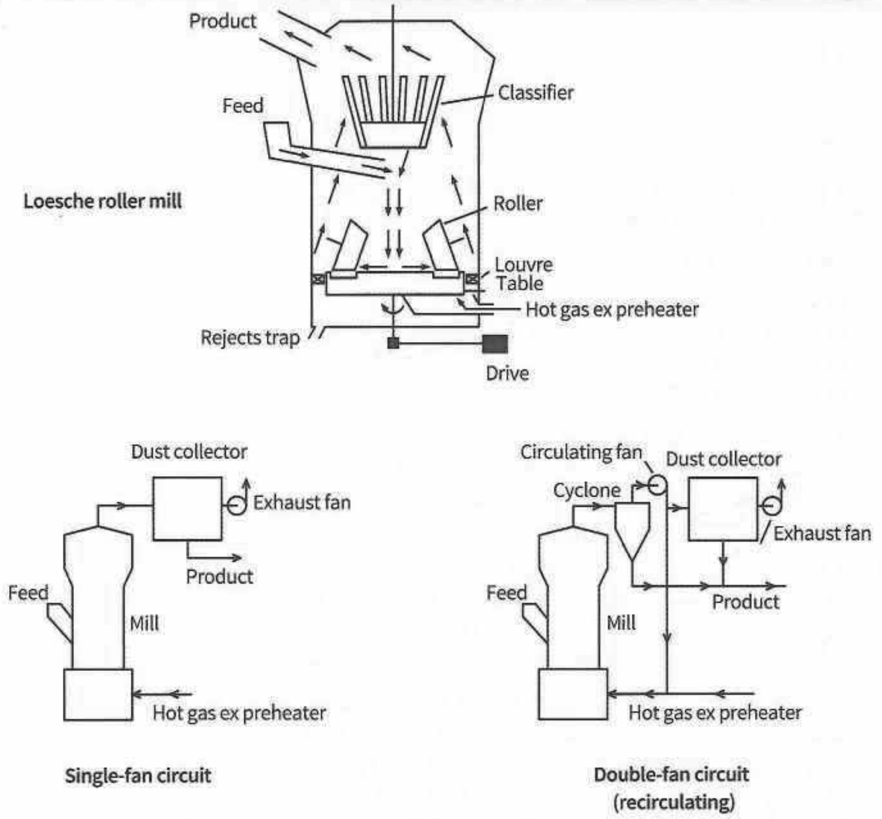


drying, grinding and separating (with no limits for drying) allows quick product changes due to the low residence times encountered.

A 2013 survey by J Harder of orders for new cement grinding mills reported by ECRA (2015) showed the VRM to be way ahead outside China, with 113 mills ordered followed by 51 orders for ball mills and roll presses combined, whereas within China, which accounts for 60 per cent of global production, the majority of cement was produced via existing roll presses.

Circulating load to the separator is normally 200-400 per cent of mill feed rate. However, circulation in the lower body of the mill can be far higher. It is common to have a mill fitted with external recirculation, ie, a transport circuit collecting rejects from the nozzle ring and returning them for regrinding. As a consequence, the velocity and flow of air and rejects through the mill is reduced, and fan power consumption decreases.

Figure 6.3 Vertical roller mill



Often the nozzle ring inlet area is adjustable so that the velocity there can be reduced from some 80-85m/s to 45-60m/s with less exhaust gas flow and lower pressure drop. The device can also optimise distribution of airflow around the mill. Material that 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 grinding surfaces.

There have been innovations in VRM classifier technology. For example, the LV high-efficiency classifier (LV Technology, 2003) gives a higher velocity profile above the table, effectively reducing the concentration of suspended material and the mill pressure drop. The technology is also applicable to vertical coal mills (Nielsen and Bank, 2002) and to ball mills (LV Technology, 2003). Christian Pfeiffer offers a version of its QDK separator, improved via CFD modelling (Woywadt, 2012), noting that significantly coarser classifier rejects produce a more stable grinding bed. Mill throughput increases of 12-30 per cent and system power savings of 1.5-5kWh/t are claimed. Other means of increasing VRM capacity are described by Jung (2004). A recent Loesche innovation is a new-generation 'LDC Classifier' with a patented 'vortex rectifier' to ensure a homogeneous air-stream exiting horizontally direct to the filter, reducing pressure drop and allowing a more compact layout (Loesche GmbH, 2012).

Mills start either with the rollers lifted from the table, or with low pressure applied. In grinding mode actual metal-to-metal contact should be prevented by limit switches or a mechanical stop, and by consistent feed.

It is important that a VRM draws its designed power. This is achieved by adjustment of roll pressure and dam ring height. Excessive rejects may result from an unduly-low dam ring. The inherently-high static pressure in a roller mill system requires tight control of air in-leakage. No more than 10-15 per cent in-leak should be permitted between mill inlet and product collector. This requires careful attention to sealing around the hydraulic cylinder tie rods, table drive, external recirculation and mill feed chute.

There are various designs differing significantly in terms of grinding paths (eg, flat table, grooved table, twin rollers), maintenance of rollers, start-up procedures, etc. These include Loesche's trend towards a larger number of smaller rollers, while thyssenkrupp (formerly Polysius) and FLSmidth offer the Quadropol and OK Mill, respectively (as indicated in Figure 3.1). (The ATOX mill is more probable for raw materials.) Other developments concern optimised drive units, as with the 'Multidrive MPS®' from Gebr Pfeiffer (Woywadt, 2012), with up to six identical drive modules to ensure active redundancy of the drive system and allow an optimum mill speed with greater flexibility. Chinese companies, including Hefei Cement Research and Design Institute (HCRDI) and Sinoma, are attempting to gain market share. Japanese designs from Kawasaki, Onoda/Kobelco, Taiheiyo Engineering and UBE Machinery have been licensed abroad (Harder, 2010). Kawasaki has described further development of its CK model and installations in China (Kawasaki, 2011). Multi-motor drives are advantageous in terms of high efficiency and mill output.

The roller modules used for cement milling refers to two grinding rollers preceded by two smaller 'support' rollers, preparing the feed bed for grinding by the master roller.

Primary VRM controls are:

- mill drive power or mill differential pressure which control mill feed rate
- inlet gas temperature for raw mills (up to 600 °C)
- outlet gas temperature
- outlet gas flow.

Marsay et al (2017) note that cement pre-hydration (resulting in extended initial setting times) can become significant in more recent VRM designs, and indicate how this can be countered by appropriate control measures (studying the internal air flow) and/or using chemical additives.

The material residence time in a VRM is usually less than a minute, compared with several minutes for a ball mill. Control response time should be appropriately faster. In addition, the feed system must be capable of responding to low material flow within no more than 45s or excessive vibration will cause shutdown. The tendency to excessive vibration can be reduced by keeping feed size at less than 100mm and the content of fines (below 1mm) at less than 30 per cent.

The grindability of GBS is up to 30 per cent poorer than that of clinker, while the fineness requirements of 4200-5600cm^2/g are comparatively high and feed moisture content can reach 20 per cent. VRMs offer process simplicity with drying, grinding, separation and material conveyance all taking place in one unit, along with very good energy utilisation and low wear rates (Gebr Pfeiffer, 2012). They can achieve an equally-high fineness of grinding >6000cm^2/g as ball mills. In slag grinding, the external circulating material quantities are higher than for raw material and clinker grinding because of the lower gas velocities in the mill. Tramp metal is removed from circulating material by magnetic separators. Various VRM circuits are encountered for raw meal and cement grinding. Typical energy savings for VRM circuits compared to ball mills are 30 to 40 per cent for cement, or 40 to 50 per cent for slag.

Roll presses

Since their introduction over 30 years ago for slag grinding, roll presses have successfully treated other materials used in cement manufacture. Problems with roller wear have largely been overcome and designs have developed for easier maintenance. They are claimed to be the most cost-effective comminution method (Seeman, 2017).

Roll presses have been used extensively for pre-grinding in a variety of circuits and are increasingly being used as stand-alone cement mills (Liedtke, 2000). Early presses suffered from roll surface and bearing failures, but operating pressures have been reduced, roll sizes increased and metallurgy improved to achieve satisfactory performance. Roll press power consumption is approximately 63 per cent of that for a ball mill for raw grinding and 55 per cent for cement, competitive with VRMs up to around $3600cm^2/kg$ Blaine, for a hitherto sustainable roll diameter of some 2.2m (Holzinger, 2018). The most common circuits are now pre-grinding with slab recirculation and semi-finish grinding. In the former (see Figure 6.4), the compacted slab press discharge is split between recirculation of 80-100 per cent relative to fresh feed and transfer to ball mill for finish grinding. This system allows production increases of up to 40 per cent. Greater capacity increase can be achieved if fines from roll press product are removed in a VS separator with subsequent selection by a high-efficiency separator before finish grinding in a ball mill (see Figure 6.5). The initial V- or 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 and Bembla, 2001). A further development, the VSK separator combines the functions of the VS separator with an integral high-efficiency separator (Süssegger, 2004). KHD's largest press now uses ϕ2.8m rolls and can grind slag to $4200cm^2/g$ at around 160tph.

Figure 6.4 Pregrinding with slab recirculation

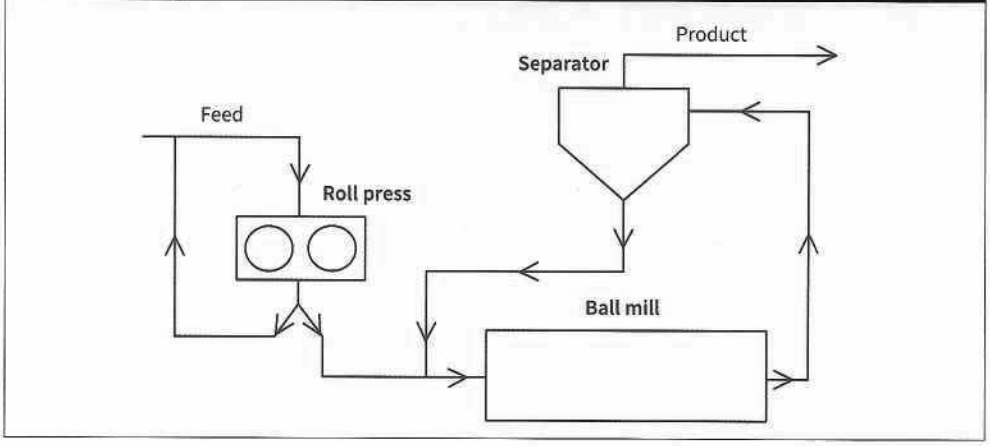
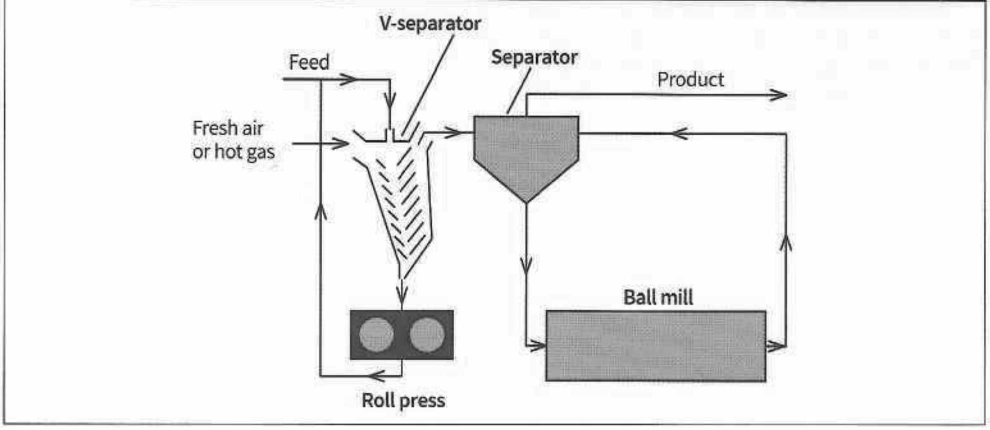


Figure 6.5 Semifinish grinding 'S'



For efficient roll press operation:

- Feed size should be limited to approximately 50mm (not more than 2x roll gap width).
- Tramp metal ingress must be prevented.
- An optimum head of feed must be maintained across the whole roll width 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 (tph)
S = gap between rolls (mm)
L = length of rolls (m)
U = roll circumferential speed (m/s)
D = product density [clinker = 2.5, raw mats = 2.3, slag = 2.45]

(Austin et al, 1995)

Separator technology and drying in an airstream allow grinding of granulated slag with moisture contents of up to 15 per cent. The recent trend is towards more compact single-stage finish-grinding units, mainly assisted by improvements in separators and longer grinding element lifetimes. There are three established European vendors: KHD, thyssenkrupp and Köppern, whilst FLSmidth has resumed with sales to the mining sector already reported. The Chinese companies CNBM, Sinoma (JV with CemTeck), CITIC and particularly Chengdu Leejun have also entered the market.

CMP AG has developed a modified horizontal mill, the 'BETA mill', claiming up to 30 per cent higher efficiencies than a roller press, but industrial-scale application so far seems limited to slag grinding.

Horomill

FCB combined efficient compression grinding with operating simplicity in the Horomill, first demonstrated in 1993. This is a cylindrical mill shell, rotating above critical speed, containing a single idler roller and internal fittings to control the flow of material (de la Fouchardiere, 2003). The roller is in free rotation but is hydraulically pressed against the shell. The mill may operate either in closed circuit or be used for pre-grinding in a ball mill circuit.

More than 40 Horomills came into operation during the first 15 years of production, most for cement grinding. Compared to a ball mill, the equivalent Horomill is approximately one-third the length and there can be energy savings of 30-50 per cent, or compared to VRMs, up to 20 per cent. So far the system seems limited to relatively low throughputs, below some 120tph for OPC at 4000Blaine. However, Fives-FCB now offers the Twin Horomill, consisting of two units operating with a single large classifier and achieving double the previous output. The addition of an 'aerodecanter' flash dryer allows grinding of slag or pozzolan with moisture content of up to 20 per cent, which may be added to the classifier gas circuit, using either clinker cooler exhaust or a hot gas generator (Poittier, 2018).

Mill performance

The overall performance of a milling circuit is best summarised by its specific power consumption at a given Blaine fineness (cm^2/g). Approximate specific power consumption for clinker/gypsum grinding in a ball mill are shown in Table 6.3.

Table 6.3 Specific power consumption for grinding clinker and gypsum

Fineness (Blaine – cm^2/g)	Specific power consumption (kWh/t)
3000	24.4
3200	26.8
3400	29.4
3600	32.0
3800	34.7
4000	37.5

Pre-grinding can reduce combined power by some 20 per cent. Abnormally-high power consumption may be due to mill inefficiency but is also likely to result from over-burned clinker. When comparing plants or proposals it is vitally important to get figures for the complete mill circuit, including all ancillary motors.

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

- kWh/t (@ 3500cm^2/g Blaine) = 33.4 + 0.32C3SN + 0.27C2S - 13.2TEA

where: C_3SN = 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:

- Blaine 1/Blaine 2 n = (tph 2/tph 1) = (kWh/t 1 / kWh/t 2)

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

High mill temperatures exacerbate material agglomeration as well as coating of balls and liners, significantly increasing specific power consumption. In practice, temperatures are often 100-130 ° C. Gypsum undergoes dehydration first to hemihydrate ($CaSO_4 \cdot ½H_2O$) and then to soluble anhydrite ($CaSO_4$). These dehydrated forms dissolve more rapidly and are beneficial in ensuring that sufficient dissolved Ca^{2+} and SO_4^{2-} ions are available to form a protective layer of ettringite and control the initial C_3A reactivity. An inadequate supply of soluble sulphate can result in an irreversible rapid loss of concrete workability known as flash set, accompanied by heat release.

However, if too much dehydrated gypsum is present, it crystallises from solution and causes a plaster or false set, whereafter continued or resumed mixing can restore the initial level of workability. Flash and false set without heat release can sometimes be caused by excessive precipitation of gypsum, syngenite ($K_2Ca(SO_4)_2 \cdot H_2O$) or ettringite ($3CaO.Al_2O_3.3CaSO_4.31H_2O$).

Ideally, cement discharge temperature should be below about 110 ° C to avoid false set, with the level of sulphate present in dehydrated forms at around 0.7-1.3 per cent, ie, about 60 per cent dehydration (Jorgensen, 1988). If excessive levels are detected, the remedies are either to improve mill cooling, reduce cement sulphate, use a gypsum blend with a high level of natural (insoluble) anhydrite, increase clinker sulphate or improve clinker cooling. Too low a level is most likely encountered when clinkers with reactive C_3A are ground at relatively low temperatures in mills equipped with high-efficiency separation. Contributory factors are low milling temperature, high natural anhydrite level in gypsum, steep particle size grading/coarsely ground gypsum, reactive C_3A, excess alkali over sulphate in clinker, slow cooling or reducing conditions in the kiln.

Grinding aids and quality modifiers

Grinding aids (traditionally ethylene glycol-based or triethanolamine-based) may be added to reduce electrostatic agglomeration (Sottili, 2002). The optimum addition rate to minimise cost should be determined, balancing enhanced grinding against power savings, and the cost/benefit ratio should be regularly re-assessed in light of the grinding aid price – though it is common to visit mills that still use the dosage first investigated in vendor tests. Grinding aids also serve to reduce pack set which is a cement-handling problem (see Section A7.22). The output gain in open-circuit ball mills is some 5-15 per cent, which can usually be obtained by circuit optimisation if resources permit.

More recently, composite grinding aids, or 'quality improvers', have been developed that also yield significant strength increase in both mortar and concrete tests (Sumner and Gianetti, 2006 and Sumner, 2010). Some catalyse C_3S hydration, whilst others have retarding properties; some contribute to fluidity (Schibuola et al, 2013). Certain additives provide marked benefits in VRM operation, with 5-30 per cent gains in output, 5-25 per cent reduction in vibrations and a reduction in water demand for bed stabilisation, whilst also improving cement quality (Marsay et al, 2012). ASTM requires grinding aids to meet the performance criteria of Standard C465.

Plant trials to determine or compare the effects of additives require careful planning and conduct. Typical clinker from a single store should be used, and the mill circuit allowed to stabilise during 4-8h operation ahead of recording operating conditions and taking samples for the 'blank' (normal) condition. Further data can be obtained every hour for 4h of stable operation with a new additive – preferably 8h – followed by a repeat 'blank' run for 4-8h. If it is difficult to quickly regulate the mill to achieve the same product fineness with the additive, then it is preferable to aim for fixed separator feed or circulating load and analyse the results appropriately.

Performance monitoring

Mill monitoring should include:

- production rate (tph)
- operating hours (h)
- involuntary downtime (h)
- mill drive power consumption (kWh/t)
- connected power, (percentage of mill motor rating)
- cement temperature (°C)
- grinding aid usage (g/t)
- ball usage (for ball mills) (g/t)
- circulating load (%)
- differential pressure (for roller mill)
- classifier/separator outlet temperature (for roller mill).

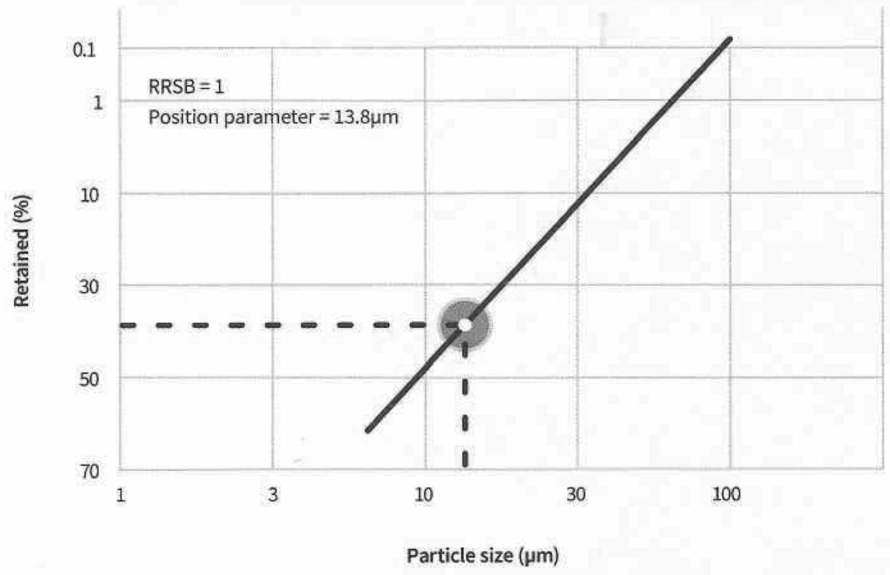
Additional monitoring will be required if pre-grinding is effected, specifically the combined kWh/t of both units will be of importance.

Particle size may be determined by a number of techniques:

- specific surface area (cm^2/g), measured by air permeability through compressed powder
- +45µm residue by wet sieving
- laser PSD systems, which can be reported as the Rosin-Rammler (R-R) function.

The Rosin-Rammler function is described by a plot of ln(1/(1-P)) versus d on log-log axis (ie, cumulative particle size fractions against particle size) gives a straight line and a narrow distribution results in a greater slope (see Figure 6.6).

Figure 6.6 Rosin-Rammler graph



It is considered that particles below 3µm contribute most to one-day strength but also increase water demand, while the 3-25µm fraction governs 28-day strength. Note that hydration only penetrates 3-4µm in 28 days. Particles larger than 25µm make a negligible contribution to strength.

Typical particle size distribution for a cement of 3600cm^2/g Blaine is given in Table 6.4.

Table 6.4 Typical size distribution for 3600cm^2/g cement

Below (µm)	%	Below (µm)	%	Below (µm)	%
180	100	30	81.6	5	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 slope function of 0.965.

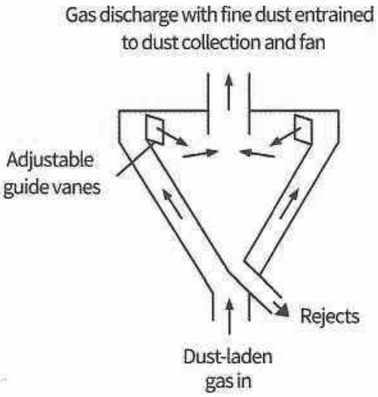
As +45µm particles contribute little to strength, their number should be minimised. At 3600cm^2/g a mechanical separator would typically yield 7-8 per cent +45µm and a high-efficiency separator 4-5 per cent. Fine cement with a narrow particle size range (as from high-efficiency separators) gives high mortar strengths but can also show high water demand, which will result in low concrete strength. Concrete strength at fixed workability, not fixed water:cement ratio, is ultimately critical, and standard mortar strengths are significant for process control and specification only – not directly for customers' concrete product designs. This must be considered in retrofitting a high-efficiency separator to an existing mill, usually justified by an increase in production due to reduced Blaine requirement for a given mortar strength, but the benefits do not always carry through to concrete (Detwiler, 1995[1] and 1995[2]).

6.3 Separators (classifiers)

Several types of separator are employed in mill circuits, with numerous variations of each.

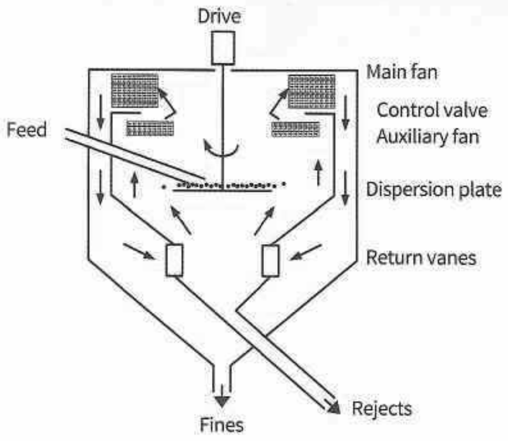
Grit separators (see Figure 6.7) are used to dedust mill air-sweep. They have no moving parts and effect separation by cyclonic airflow induced by guide vanes. Radial setting of the vanes gives minimal separation and near tangential setting gives maximum. Vanes should be adjusted so that dust collector catch has the same 45µm residue as the product. The dust collected can then be conveyed directly to the product. These units are typically used in circuits with high air-sweep.

Figure 6.7 Grit separator



Mechanical (static) separators (see Figure 6.8) are the traditional product classifiers. Material is fed on to a rotating dispersion plate whence it spins 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 decreasing velocity and diversion through the return vanes. Operating adjustments are the number of auxiliary blades, clearance between auxiliary blades and control valve, and the radial position of

Figure 6.8 Mechanical separator



the main fan blades. These determine the material load in the separating zone of the classifier and are critical to separation efficiency. The main fan blades establish the airflow, while the dispersion plate effects the distribution of material into the rising airflow. The height from the return vanes to the dispersion plate defines the classifying zone. For good operation, the optimum mass flow of material per unit volume of air (kg feed/m^3 air) ought to be established for each product fineness.

High-efficiency cyclone separators were introduced to improve on the mechanical separator's low efficiency regarding fines recovery. A simplified process flow for the O-Sepa (see Figure 6.9) is as follows. Material fed on to a rotating dispersion plate 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 airflow. The rotor forms a horizontal vortex which classifies particles between centrifugal force and the inward airflow. The fine fraction exits upwards with the exhaust for subsequent collection as product, while the coarse fraction falls and is discharged from the bottom of the vessel. Fineness is controlled solely by rotor speed: increasing speed increases fineness. Airflow 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 design of the incoming air ducts and guide vanes. The height/diameter ratio of the rotor controls retention time in the separating zone. Later developments of the O-Sepa have reduced airflow rates by around 30 per cent without changing efficiency (Ito et al, 2007).

Figure 6.9 O-Sepa separator

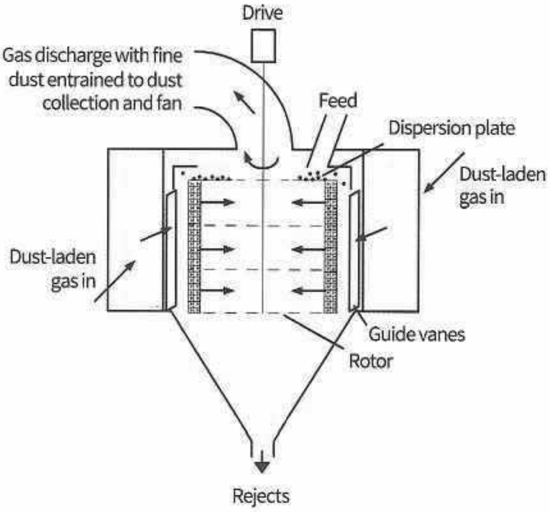
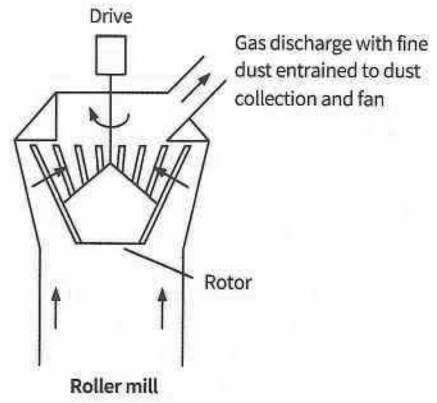


Figure 6.10 Vertical roller mill classifier



Dynamic classifiers, used integrally with a roller mill (see Figure 6.10), involve the upward flow of dust-entrained air into a segregating area above the grinding table. There, with decreasing air velocity, coarse particles fall back to the table while fines leave with the exhaust for external collection. Design developments have yielded a progressively steeper Rosin-Rammler distributions of mill product and an increasingly coarse reject fraction returned to the table, providing a more stable grinding bed. Rosin-Rammler slopes are summarised as follows:

High-efficiency classifiers	yield Rosin-Rammler 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 the LV classifier (see above) has also achieved significant pressure drop reduction, system power reduction and increased throughput on roller mills.

Complete mill circuits can be quite complex and when comparing mill efficiencies, it is particularly important to include power consumption by the separator, fan(s) and and other equipment as well as that of the grinding unit alone. In some cases this additional consumption can even negate the expected saving over the level for an efficient ball mill circuit. For example, a VRM producing 150tph at 3200Blaine might have an apparent advantage of almost 10kWh/t over a ball mill, but be associated with ancillaries that consume over 5kWh/t more than those in the ball mill circuit.

Possible mill circuits are legion, from a single open-circuit mill to a combination of mills and separators (Onuma and Ito, 1994). The situation is rendered particularly interesting when increased performance is required from existing equipment and new units are combined with old. The only generalisation that can be made is the obvious caution that balance and control are essential, and the observation that new installations prefer more simple circuits.

6.4 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 drawn, airflow and mill internal temperature. Classification depends upon classifier feed rate, airflow and either rotor speed or vane/diaphragm setting. Rule-based controllers operate on well over 500 mills, with FLSmidth and ABB taking the lion's share. MPC systems control about 40 more (Haspel, 2009).

Mill feed control maintains the quantity of fresh feed and the proportions of individual components. The feed rate determines the ratio of feed to grinding media in the mill, the optimum steel to clinker mass ratio is typically 8-12.

Mill power control depends upon the weight of grinding media. Periodically, make-up charge is added to maintain maximum power draw.

Mill ventilation in a two-compartment mill with L/D ratio of 3.2 is limited to around 1m/s and is normally monitored by static pressure at the discharge which should be typically -100 to -150mm WG. Ventilation governs 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 directional microphones located externally approximately 1m from each end of the mill and shielded from interfering noise. KIMA, a successful new entrant, fits sound monitors to the mill shell, with wireless connection to the controller.

Control philosophy is primarily to maintain the optimum material level in the mill, with airflow held constant, as judged by fan power. 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. Mill noise level, power drawn and pressure drop are used to detect problems such as plugging in the mill, causing it to fill 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 rate or, where drying is effected, by control of inlet gas temperature.

Classifier control determines product fineness and circulating load (rejects returned to the mill), and usually remains constant for extended periods. In mechanical separators, airflow is determined by the diameter and number of blades on the main fan. Cyclone separators allow independent adjustment of airflow and rotor speed. Use of real-time laser psd data for classifier control can provide significant benefits in terms of mill output and consistency of product size distribution (DeNigris, 2018).

6.5 Cement storage

Combined storage capacity is discussed in Section A6.1. Minimally, there needs to be sufficient silos (or compartments in multi-cell silos) to keep different cement types without simultaneous filling and discharge, and capacity should also be adequate to complete at least one-day strength testing before shipment. With cement storage typically costing between US$200/t (dome storage) to US$450/t (silo storage with piling), there is a strong pressure to minimise both total storage capacity and the number of silos for a given capacity. Dagnan (1986) suggests a guideline of 30 days' production and details the factors involved, though this may now be considered an unjustifiable cost. He also specifically addresses the economics of small-volume special products. Erratic powder flow in silos can lead to failure due to asymmetric stresses and large concrete silos built to older construction codes may need to be post-tensioned to ensure their security (McKay and Durack, 2006).

Cement should, if possible, be below 60°C when conveyed to silos. The storage of cement at greater than 80°C causes gypsum dehydration, and the reaction of the released water with fine cement particles results in loss of early strength and promotion of false set, as well as build-up in silos. After the mill, cement can be cooled if necessary, using a dedicated cement cooler (Kochmann and Ranze, 1997) or by water jacketing a pneumatic conveying line.

Silos for most materials are prone to developing dead zones, 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, 2002). Build-up and the dubious performance of many silo level detectors may lead to erroneous inventories. Methods and problems of level detection such as internal structures, build-up, uneven surface layers and dust are reviewed by Little (2004) and Incontri (2018) describes a new radar technique designed to provide accurate, continuous and reliable data, along with lower costs. Erratic flow of cement is described by Dudley (2002) while Graham (1997) 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 to detect anomalies.

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

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

6.6 Cement dispatch

Shipments of cement range from almost wholly bulk in developed markets to almost wholly bagged in less-developed countries.

Bagging adds approximately US$5-6/t to production cost and can, of course, increase shipment costs depending on bag size and extra handling requirements.

Valved bags made from two-ply kraft paper are common, being strong enough to contain cement and stand up to rough handling, they must be permeable to air while being filled and they are impermeable to moisture thereafter. Intermediate plies may be added, perhaps of micro-perforated polyethylene, though high-permeability grades of kraft paper are making an appearance. Pneumatically-conveyed cement is highly aerated and the air needs to be evacuated for a bag to contain its correct weight. If the bag walls are not sufficiently porous, micro-perforations may be provided. Once the air has escaped, the weight of the cement ensures that the perforations reseal, keeping the bag watertight.

The major cost element in sacking is kraft paper, which has historically been volatile in cost and, occasionally, in short supply. If a plant has a significant bag market some effort is justified in purchasing and stockpiling paper and in optimising bag configuration against breakage (Grundy, 1993). In-plant sack breakage of less than 0.3 per cent should be maintained by careful attention to sack handling and load-out conveyors.

Offenbecher (2007) reported on a VDZ investigation of the degree of moisture protection actually provided by a variety of bag systems, storing the bags on pallets in a warehouse for six months. When stored without additional protection, cement from the bags without an additional moisture barrier showed changes in quality, but no differences were obvious when pallets carrying any type of bag were wrapped in a protective hood.

Where acceptable to the market, woven polypropylene bags offer both strength and economy (Worff, 2000). Plastic bags have become more popular in mature markets during the current century because they increase shelf life and weather resistance, generate less waste than paper and are easier to recycle. They can also be a cheaper alternative and accept printing more easily than kraft paper (van Noort, 2017).

Cement sacks are typically 50kg, 42.6kg (ie, 94lb, which occupies $1ft^3$) or 40kg net weight. However, regulations in the European Union and elsewhere require a maximum weight of 25kg (Anon, 1996), a size which is spreading to some other regions.

Bag weight standard deviation should be determined to minimise overweight. ASTM C150 Section 11.3 specifies that packages more than two per cent underweight may be rejected, and that if 50 random bags 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 one per cent, fewer than one in 20 bags 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 one standard deviation.

Bulk loading of road or rail trucks is normally at about 400tph, and of ships and barges considerably faster. Twenty-five-tonne road trucks can be loaded at six-minute intervals from a single loading spout with appropriate equipment and procedures. Filling spouts are telescopic with conically-tapered nozzles to seal against tank openings. The filler comprises a central cement delivery pipe surrounded by an outer suction tube which vents at least $1000m^3/h$ air and is ducted to a dust collector. Load weight is determined either from tare and gross weights of the truck (preferably on dedicated 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 and Pauer, 1995). 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 Research (2003).

A third mode of shipment is the 'big bag', a 0.5-2t flexible intermediate bulk container (FIBC) 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 their use has been limited to special situations (Lasheen, 2003), or as temporary 'silos' for small batch plants.

Automated dispatch systems as described by Bauer (2004) are increasingly employed and can help to minimise check-in:check-out ("CICO") times for trucks and local congestion.

6.7 Distribution

Cement may be sent to customers from its place of manufacture or from separate grinding plants. It can also be distributed or collected from a remote depot supplied by rail or water – or it may have arrived at a quayside terminal from a manufacturing site hundreds of kilometres away. Depots and terminals should be designed firstly with material throughput and logistics in mind, rather than storage. 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. The following formulae are provided for estimating unit bulk shipment costs (see Table 6.5), though it is noted that there can be a wide variation in costs depending on geographical location for any given mode of transport.

Table 6.5 bulk shipment costs, 2014

23-25t road truck	= US$0.11/t x km (one way)
2000t rail shipment	= US$0.060/t x km
40,000dwt sea shipment (long haul)	= US$0.003/t x km
10,000dwt sea shipment (short haul)	= US$0.03/t x km

On this basis, a 320km (200m) haul would cost approximately US$35.20/t by road and US$19.20/t by rail. This, of course, may not reflect door-to-door cost where road haulage normally offers an advantage. Road transport is usually more flexible than rail for small, irregular or short-notice shipments.

By comparison, a 6000km east-west transatlantic shipment with a 40,000dwt vessel would cost about US$18/t (subject to the specific trade route as well as the normal cyclical variation of shipping costs). It is obvious, therefore, that coastal areas are very exposed to low-cost imports unless protected by tariffs. Indeed, 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. Polomsky (1995) reviews the operation and economics of bulk carriers of greater than 25,000dwt and Thomas (2004) 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: capacities of above 400tph are usually needed, often involving fluidslides. Technology for ship loading and unloading continues to develop (Van Aalst Bulk Handling BV and Dragon Alfa Cement Ltd, 2012) and ports continue to implement projects.

According to Cement Distribution Consultants, world seaborne trade in cement and clinker reached 227Mt in 2017, including 98Mt of domestic deliveries. A total of 44 per cent of this involved self-discharging cement carriers, 41 per cent bulk carriers (Handysize and larger) and the remainder, coastal cargo vessels. A significant volume of cement and clinker nowadays is shipped in partially loaded Supramax and Ultramax bulk carriers which are rapidly replacing the Handymax size vessel, for which most cement terminals and clinker import facilities have been designed. Upgrading these facilities to handle fully loaded vessels is taking place, although still at a moderate pace.

Inland waterways play an important role in some regions, with cost reputed to be only a fraction of road transport costs. Holcim's Ste Genevieve plant is centrally located on the Mississippi River below the locks and dams of St Louis. This ensures cheap transport to depots in cities and distribution points along the Missouri, Mississippi, Ohio and Illinois rivers, supplying customers who account for 27 per cent of US consumption. Similarly, the New Zealand industry has relied heavily on coastal sea transport to supply distribution terminals.

European figures for cement transport in 2007 were US$13.50/t/100km by road and around US$20.20/t to cross the Mediterranean Sea. In addition, transport costs per tonne are lower to ship a large consignment of cement from the north coast of France 5400km across the Atlantic than to truck it 180km to Paris (Cembureau, 2013).

6.8 Quality assurance and customer service

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

It is customary for cement manufacturers to provide technical support for cement users. The more common cement and concrete problems are described briefly in Section A7.22. 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 bag shipments and, possibly, primitive storage facilities between plant and customer, it is important that all bags should be date-marked, albeit in code. It is not unusual for complaints of hydrated or low-strength cement that may be years old. The counterfeiting of cement by substituting ground grey dirt can also occur. Records allowing the tracing of questioned shipments to silo and production data are desirable. Clear date-marking of bags is a legal obligation where regulations demand addition of agents to cement to reduce the level of soluble chromate and cut the risk of sensitisation to 'cement dermatitis' due to contact with moist mortar and concrete.

It has been found valuable to record and analyse all customer complaints, and 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 justifies the cost.

7. Quality control

In an increasingly competitive market where cement companies vie for customer satisfaction and retention, quality control becomes ever more important.

The quality department is responsible for:

- advising procurement staff on selection and specification of raw materials and fuels
- analysis of raw materials and fuels
- mix design, reflecting individual raw material costs, production costs (principally drying, grindability and burnability) and product quality
- selective winning of limestone
- preblending raw materials and blending kiln feed to minimise chemical variation
- process control, which involves raw material moisture, raw meal chemistry and fineness, clinker free lime, proportioning of clinker and gypsum and cement fineness
- 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
- managing the sampling and analysis of purchased materials and environmental samples
- (also miscellaneous other functions such as water treatment and measurements associated with process engineering and pollution control; frequently, the laboratory maintains and reports production and inventory data).

7.1 Sampling

Sampling – the reduction of a large quantity of material to a small portion that 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), for which purpose various devices may be installed (Trottier and Dhodapkar, 2012), or from a stopped conveyor belt where the entire stream should be cut. Reduction involves successive crushing and splitting of samples to achieve the final analytical sample. The ASTM procedures for aggregate are described in D75 and C702. Theoretically, the required sample size is estimated using standard formulae involving predetermined factors for material heterogeneity, particle size, density, shape factor,and required precision (Pitard, 1993). The minimum representative weight (MRW) is given by Pitard as:

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

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

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

- between 0.2-2 per cent of total flow, depending upon heterogeneity and particle size (Labahn and Kohlhaas, 1983)
- minimum sample size as related to maximum particle size – see Table 7.1.

Table 7.1 Guideline for sample size (ASTM C75 - Standard Practice for Sampling Aggregates)

Maximum particle size (mm)	Minimum sample size (kg)
10	10
20	25
30	60
40	80
50	100
60	120
75	150
90	175

The appropriate frequency of sampling from various process streams is discussed by Clark (2008).

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. EN 196-7 applies in Europe.

Sampling procedures for coal are given by ASTM D2234, D4915 and D2013, as well as ISO 18283 (manual methods) and ISO 13909 (mechanical sampling from moving streams).

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

7.2 Chemical analysis

The main methods for chemical analysis now in use are outlined here, but with the wider use of wastes bringing more trace and minor elements to the process, additional methods are being employed at plant level to improve and maintain the best process conditions and cement quality.

X-ray fluorescence

Process control requires online or rapid offline analyses to adjust material feed proportions to the raw mill. X-ray fluorescence (XRF), using either pressed-powder or fused-disk samples, was almost universally employed by the end of the 20th century, but advanced X-ray diffraction methods have since made significant penetration. Numerous XRF instruments are available, either wavelength-dispersive machines (counting elements of interest either simultaneously or sequentially), or energy-dispersive machines. A primary beam of X-rays stimulates secondary X-ray emission from each element present, with intensity proportional to their concentration. The secondary radiations, each with characteristic wavelength, can be separated geometrically using appropriate analysing crystals for individual detection and measurement (ie, wavelength dispersion), or collected as mixed radiation and analysed into energy bands electronically (ie, energy dispersion). The latter mode of operation provides poorer theoretical dispersion but, with good computer algorithms to resolve overlapping bands against a high background, offers a much cheaper, simpler and more reliable system (Field and Hornung 1995).

Each element in the sample attenuates or enhances the X-radiation of others to varying degrees, so that inter-element 'matrix' corrections are necessary and calibrations are not linear. While wide matrix calibration programs are available, those used for process control apply only to a narrow range of composition. When using pressed pellets, discretion should be exercised to avoid inappropriate calibrations - for example, the use of a kiln feed calibration to analyse bypass dust. XRF analysis is less sensitive for lower atomic weights and sodium analysis near 0.1-0.2 per cent levels is frequently marginal. One should mistrust any computer-generated calibration with zero slope and an intercept equal to the previously unvarying result.

X-radiation only penetrates a few micrometres into the sample and powders must be ground to -20µm before being pressed into disks for analysis. Some minerals may render the surface non-representative, eg, quartz, which may be present in large particles, and platy minerals. Fused samples eliminate errors due to particle size and mineralogy and are essential for accurate comparison of samples from different locations, while pressed powder discs are appropriate for materials from a single source.

Pedersen and Wismann (1997) review XRF sample preparation and make an interesting point in estimating relative contributions to total analytical error as shown in Table 7.2. Davidts and Deprez (2018) discuss good practice for the preparation of fused bead samples. Fusion of hot meal, kiln dust and bypass dust may lead to partial volatilisation of Cl and SO_3, and other methods are preferred for volatile balance studies (see below).

Table 7.2 Estimated contributions to total XRF analytical error

Technique	Contribution (%)
XRF measurement	10-15
Sample preparation	25-30
Sample collection	55-65

EN 196-2.2 was the first European standard for XRF analysis of hydraulic cements and is the basis of ISO 29581. A similar standard has been developed for the current edition of ASTM C114, which also includes accuracy and precision requirements for all analytical methods (see below).

Note that XRF is not normally a primary method of analysis and must be calibrated from standard samples, though standardless XRF is useful for approximate analysis of alternative raw materials, or special elements in common raw materials, for which no calibration has been established (Möller, 1995, and Boumans and Blaakmeer, 1995). The actual sample analysed is of the order of microgrammes and requires considerable care in rigorous sample reduction procedures: neutron activation methods that analyse a total process stream are intuitively preferable.

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

Table 7.3 ASTM criteria for precision and bias (Table 1 of C114)

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

** Insoluble residue is the portion which is insoluble in hydrochloric acid. Almost all clay compounds in the raw mix are insoluble, but become soluble after reactions with lime. Hence the residue indicates the efficiency of the burning process.*

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

PGNAA

Prompt gamma neutron activation analysis (PGNAA) uses a source of neutrons to radiate samples, resulting in characteristic secondary gamma radiation. Gamma radiation, being of much higher energy than X-radiation (0.5- 5MeV vs 1-50keV) is less subject to attenuation, so that bulk material (10-100cm thick) can be analysed, where only the sample surface would be sensitive to XRF. The first analysers were applied in 1985, developing into the superior cross-belt design by 1994. The gamma spectrum allows rapid, online analysis of process streams without sample preparation and can analyse Si, Ca, Al, Fe, Ti, Mg, K, and S as well as Cl, N and H. Many plants now use cross-belt PGNAA analysers to control the composition of pre-blend piles and raw mix. In 2007 some 20 per cent of installations were for preblend and 80 per cent for raw mix control (Foster, 2007). Accuracy and precision are similar to that of XRF and sampling error is drastically reduced (Macfadyen, 2000). However, it is still common practice to verify online analyses by laboratory XRF analysis. (Foster and Bond, 2006). Established suppliers, with many hundreds of units installed, include ThermoFisher Scientific, Gamma-Metrics, FTC-ACTech and PANalytical, recently joined by offerings from China.

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, free lime, free magnesia (periclase), clinker phases and other mineral phases in conventional and alternative raw materials. Most other techniques measure concentrations of elements and then report these as oxides.

Powder diffraction used to be of relatively low precision and limited to comparative studies in kiln chemistry (Beilmann, 1993). It later provided a method for rapid determination of free-lime in clinker and ASTM C1365 provides a standard XRD method for phase analysis of Portland cement and clinker. (Note that conventional 'wet' methods of free lime determination also include calcium hydroxide if the cement has been subject to some hydration: XRD will not measure this, which can lead to discrepancies.)

Late 20th century advances in applying total XRD pattern analysis using the Rietveld standardless method have improved speed, precision and accuracy (Fullmann and Walenta, 2003, and Lundgaard and Jons, 2003) and 'on-stream' analysis is also being applied (Storer and Manias, 2003, and Rapson and Storer, 2006). The Rietveld method was developed in 1969 to refine neutron diffraction data and later adapted to XRD, rendering the method quantitative, with the new acronym QXRD. Intensities calculated from a crystalline structure model are compared with the observed X-ray pattern using a least squares refinement, varying the crystal structures parameters and the peak profiles to get the best match between observed and calculated patterns.

The FCT-ACTech on-stream system continuously extracts a sample for grinding to approx 50 per cent <45µm, and sub-sampling to provide a smaller, virtually-continuous representative sample of about 300-600g/min, which is further milled to continuously feed the device. Operating on cement, a complete quantitative picture of the genuine crystal phases can be obtained, including various polymorphs of alite and belite, along with free lime, calcite and forms of calcium sulphate (see Section B7.3). Other suppliers include Bruker and PANalytical, and Rietveld software packages can be uploaded to equipment from some other suppliers, such as Thermo/ARL.

QXRD offers additional structural information, difficult to obtain by other techniques, such as live trends of the state of gypsum dehydration ex-mill, to be used to optimise mill conditions (Storer, 2007).

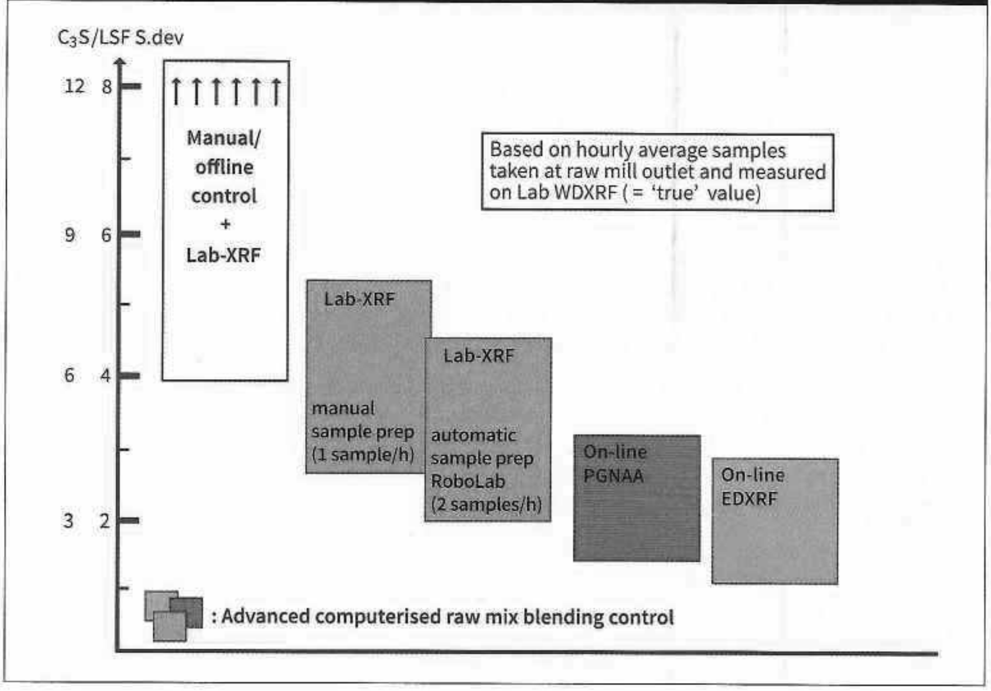
Automation

To meet the requirement of rapid analysis and adjustment for efficient control of a continuous operation, automatic systems for sampling, sample conveying and preparation and for chemical and physical measurements are being increasingly used (Riedhammer, 1993; Möller and Zement, 2004, and Hepper, 1996).

Analytical equipment has become increasingly automated, with interfaces feeding signals to plant information systems. Robust devices have been produced to operate alongside manufacturing plants and transmit information to the laboratory, for example, particle size distributions and clinker free lime. Complete XRF devices can also be so located. Alternatively, analysis of all samples can be centralised, using pneumatic sample transport to take closed containers to the laboratory where one or more programmable robot arms can arrange for their drying, comminution/dissolution and presentation to various analytical devices. (Jacobs and Regis, 2005).

Important systems include FLSmidth's QCX and Polysius' POLAB, both of which continue to develop. Retrofits are likely to be costly but result in better frequency and reproducibility of analyses as well as in direct linkage to plant control and laboratory management systems. Figure 7.1 indicates the improvements in standard deviation reported by FLSmidth Automation (2002).

Figure 7.1 Typical domains for standard deviation of controlled parameters for alternative raw mix control applications as reported by FLSmidth Automation (2002)



Other

For volatile mass balance studies, Cl is best analysed by $AgNO_3$ titration (ASTM C114) and sulphur by high-temperature combustion (ASTM D4239). Low levels of chloride, fluoride, sulphite (in some synthetic gypsums) and sulphate in raw materials can also be analysed as anions in solution by ion chromatography (IC), involving separation by ion-exchange and sequential detection by conductivity cell. Trace metals in waste fuels, raw materials and kiln dust can be analysed by induction-coupled-plasma (ICP) emission spectroscopy in solution (probably after microwave dissolution) (Dirken and van der Broeck, 2004) or atomic absorption (AA) spectroscopy: care is needed in sample preparation to avoid contamination. A corporate central laboratory or a commercial testing laboratory often provides such services, which become more important when using wastes.

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

Plants making blended cements with slag need to perform separate determinations of sulphur in sulphide and sulphate forms (C-595, and C-114; Section 15.2). In this test method, sulphide S is evolved as H_2S from an acid solution of cement and dissolved into a solution for titration against potassium iodate (KIO_3). Alternatively, total S and sulphate S can be determined by other methods and sulphide S is the difference between the two.

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

The laboratory has an important role to play in assuring that process and environmental monitoring are carried out correctly. This involves understanding how the sampling and monitoring equipment must operate to reliably produce representative results, and managing the acquisition and storage of samples.

Cement plants using large quantities of waste materials and fuels need to satisfy regulators and neighbours that they present no dangers to health or the environment. The onus for this may be placed upon suppliers, but it is likely that, at the least, representative samples will have to be taken and stored with integrity. An appreciation of techniques for organic analysis becomes necessary and a good number of plants have established their own in-house facilities, involving equipment such as chromatographs and mass spectrometers, along with staff to operate these. A range of evolving national and regional standards address many of the issues involved, and their guidance must be followed.

7.3 Particle size analysis

There are many computerised techniques for measuring particle size distribution (PSD). Commercial automatic online analysers for kiln feed and cement have been used for several years (Kaminaga, 1997), with dry analysis being the more practical, even though some wet methods may provide better absolute accuracy (Kalkert, 2001). Online analysers are principally justified by shortening the transition time when multiple types of cement are produced and by offering information with which to control classifiers.

Methods use different principles such as laser light diffraction by particles in random motion in a dynamic liquid or air, the settling of particles in a still liquid, the effect of particles on the impedance of a sensing zone of an electrolyte solution, electron or optical microscopic image measurements, or sieving. Results from different techniques are not expected to agree. Due to speed and ease of use, laser diffraction analysers are the most widely applied in the cement industry. However, many variables can affect the PSD curves that they produce (Geurts, Van Loo and Sumner, 1999). These were highlighted by two ASTM 'round-robin' tests and laser PSD analysis is unlikely to be approved as an ASTM standard method until they are resolved. The most important variables are:

- powder dispersion (dry aerosol or ultrasonic alcoholic dispersion; isopropyl alcohol appeared to be the best for liquid dispersion)
- choice of optical models (Mie or Fraunhofer) and the values of particle refractive indices for converting diffraction data into PSD
- solid concentration.

7.4 Thermal analysis

Differential scanning calorimetry (DSC), is not widely encountered at plant level. It provides a simple and fast method to quantify $CaSO_4.½H_2O$ and $CaSO_4.2H_2O$ in cement and determine the degree of gypsum dehydration during milling. Gypsum or gypsum/anhydrite addition rates, and the degree of gypsum dehydration greatly influence cement performance in setting, strength and interaction with concrete admixture. DSC is also useful in quantifying needle-shaped syngenite ($CaSO_4.K_2SO_4.H_2O$) and ettringite ($3CaSO_4.C_3A.32H_2O$) which govern dry cement flowability and pack-set problems and – with secondary gypsum – cause early stiffening of paste, mortar, or concrete. Thermo-gravimetric analysis (TGA) determines losses of sample weight as a function of temperature up to 1600°C, and is used to detect pre-hydration of cement in a mill caused by over, or incorrect, spraying of water. It also determines $Ca(OH)_2$ and $CaCO_3$ in cement or weathered clinker. When troubleshooting volatile build-ups in a kiln system, TGA of materials alone, or in combination with differential thermal analysis (DTA), provides insight to the thermal behaviour of compounds (Miller, 1982). Most corporate central laboratories have TGA and DTA. Sample sizes are fairly small, usually from a few to tens of mg and careful preparation is very important (Bhatty and Milller, 2004). When producing limestone cements or using less than five per cent limestone as a minor additional constituent, it is necessary to demonstrate that the proportion in a cement sample is within the permitted limits and TGA can be used to do this.

It is good practice to specify quality parameters for gypsum supplies, setting limits upon clay/chalk content, and upon the anhydrite to dihydrate ratio. For example, below 20 per cent impurities and an anhydrite:dihydrate ratio of 0.5, 1.0 or 2.0 (or else zero anhydrite, for some sources). It is important to sample deliveries and so be in a position to check, when necessary, that a reasonable degree of stability has been achieved.

7.5 Microscopy

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

The method is described by Ono (1995). A PCA report reviews the method (Campbell, 1992), with further information given in the standard text by Campbell (1999).

The method, in gross simplification, involves crushing clinker and screening to 106-44µm. A 1.715 refractive index immersion oil is used to mount the sample between a glass slide and cover glass for examination at 200-400x magnification using transmitted polarised light. Measurement is made of alite (C_3S) size, alite birefringence, belite (C_2S) size and belite colour.

Belite colour (BC) is observed under plane-polarised light, optionally 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 to selectively dissolve matrix phases to facilitate determination of BC and alite birefringence. Measuring birefringence is rather time consuming and determining size ranges of alite and belite is somewhat subjective. However, meaningful results can be obtained with experience.

Alite size (AS) indicates the rate of temperature rise of kiln charge between ca 1200-1450 °C where C_2S is combining with CaO to form C_3S. Rapid heating is desirable and is indicated by alite of 15-20µm size, while slow heating produces alite of 40-60µm or larger.

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

Numerical ratings assigned to BC are:

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

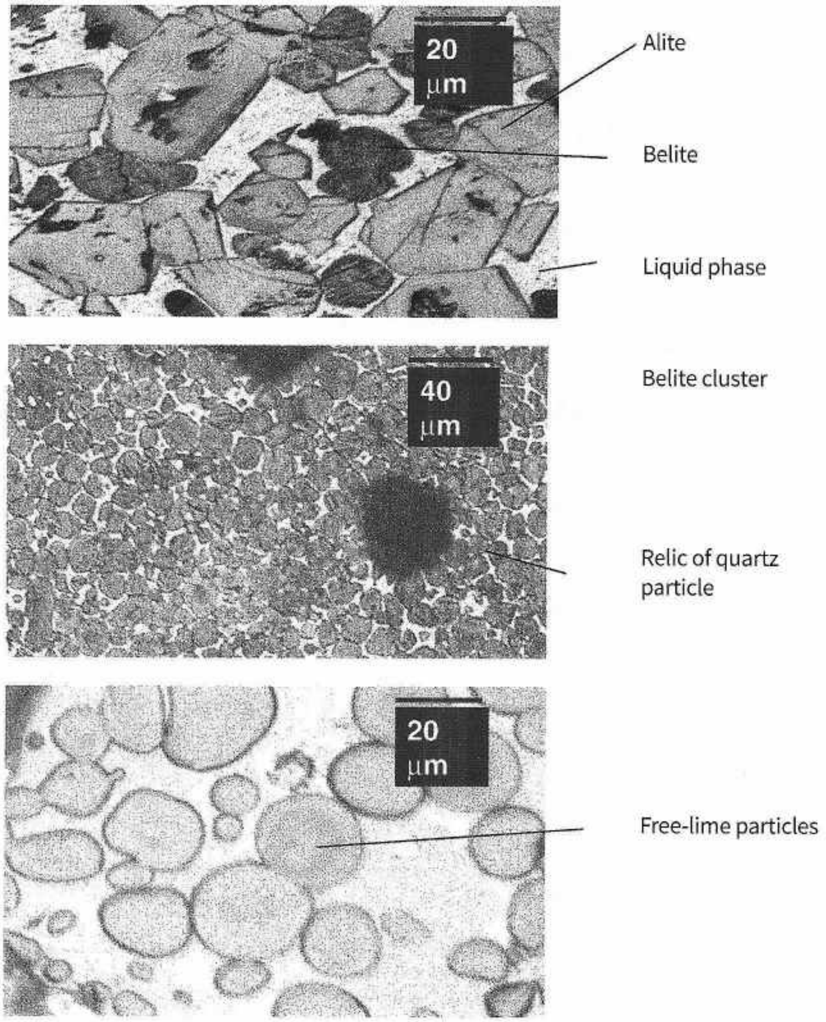
A correlation with 28-day mortar strengths was produced:

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

Ono also related relative clinker grindability to the same set of parameters (Ono, 1981, p198-210).

Belite colour and alite birefringence have much greater influence on strength than alite or belite size. The correlation applies when burning conditions are reasonably constant. However, distortions result from variation of alite and belite, free-lime and alkali sulphate concentrations, gypsum dehydration, and cement fineness. Furthermore, hard burning and coarse kiln feed can cause large alite and belite and thus less-reactive clinker, while under- burning can result in low-quality clinker with very small crystals. Thermal profiles of long kilns produce less-reactive clinker having large alite and belite crystals, while shorter and faster preheater and precalciner kilns make more-reactive clinker, having

Figure 7.2 Clinker micrographs



smaller crystals. High clinker SO_3 tends to increase alite size and clinker dust.

A more demanding parallel technique is to inspect polished sections, which yield much additional data. 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, 1992). The latter arises from coarse silica particles in kiln feed and has an adverse effect upon grindability and 28-day and later strengths. Presence of different polymorphs of alite, belite, aluminate and other microstructures often reflects 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 Campbell, 2004). As mentioned earlier, calculation of potential Bogue compounds from concentrations of the major oxides is misleading. QXRD and microscopic point-count usually give higher alite and lower belite levels than calculated potential Bogue values.

7.6 Virtual Cement and Concrete Testing Laboratory (VCCTL)

Cement manufacture involves extensive physical testing to assure product performance, but conventional tests incur delays of 28 days or longer. Since the 1980s computational models have attempted to predict cement and concrete physical properties and performance. In 2001 a consortium was established between US NIST and international partners to develop a software tool named Virtual Cement and Concrete Testing Laboratory (VCCTL), updated annually. VCCTL aims to help the concrete industry's long-term drive to convert from prescription specification-based production to performance specification-based production. The system is also applicable to composite cements and, through prediction of diffusion coefficients, concrete durability can also be modelled (Bullard, 2014).

The software can create virtual materials, using carefully characterised cements, mineral additives, fillers and aggregates. Further, a user can simulate curing under various conditions and calculate thermal, mechanical, and transport properties as a function of processing.

The Nanocem initiative (www.nanocem.org), founded in 2004, has similar aims, using emerging techniques to gather fundamental information about the development of concrete microstructure at atomic and nano scales, along with data regarding structure of hydrates and gel porosity. Results are integrated to a framework spanning the atomic to the macroscopic scale with a good theoretical framework, using advanced computational methods, established in other fields. Eleven core projects are complete and five are ongoing, covering subjects ranging from thermodynamics to the kinetics of cement hydration, from admixture interactions with cement to concrete durability, additionally, there are "partner projects" covering a wider span. Most results have been published and are listed on the Nanocem website.

These long-term projects facilitate investigation of alternative materials and process changes, rather than replacing conventional routine testing.

7.7 Calorimetry

An isothermal or conduction calorimeter monitors and determines rates of heat release from hydrating cement at constant temperature over a day or more, enabling the study of cement reactivity and factors that influence it. An example of a heat release profile is shown in Figure 10.1. ASTM C1679 provides a 'Standard Practice for Monitoring Cement - Admixture Interactions Using Isothermal Calorimetry'. Developments in using this technique as a standard method for measuring two-, three-, and seven-day heats of cement hydration were reported by Lipus and Baetzner (2008).

Isothermal calorimetry cannot simulate the heat effect of massive hydrating concrete, which is nearly adiabatic, with temperature increases accelerating hydration. Semi-adiabatic and adiabatic calorimeters are more suitable: these are easy to assemble, cost much less than isothermal units and can be used in the field (Springenschmid, 1999 and Wang et al, 2008).

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

7.8 Burnability

The burnability or 'combinability' of kiln feed is determined experimentally by heating pelletised samples for a fixed period of time, eg, 15 or 30min, in a furnace set at different temperatures, commonly between 1350-1550°C, and determining free lime after cooling and pulverising. A programmable furnace approximating the thermal profile of an industrial kiln tends to improve reproducibility. Before changing kiln feed fineness or chemistry, or using a new raw material, the effect on laboratory burnability should be checked, using the correct amount of coal ash in the mix. Even with an ash correction, laboratory clinkering usually produces a higher free lime than burning in a kiln, probably due to the lack of mixing action and the tendency of kiln operators to over-burn. There are formulae for estimating burnability factors, considering only the influence of raw mix chemistry as described in Section B5.3, but these ignore the effects of fineness and mineralogy.

Harrisson (2010) points out that, while conventional wisdom commends soft burning to a reasonably high clinker free-lime level (~5 per cent), this can have unforeseen effects when the LSF level is at or below 100 per cent. In this situation, each one per cent free lime indicates that there has been a failure to produce the amount of C_3S that had been calculated for the raw feed. If the clinker LSF is as low as 94 per cent, the loss of C_3S with each per cent free lime becomes significant.

7.9 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. Hills (2007) reports a comprehensive survey of literature on clinker grindability. The Bond ball mill is widely used as a standard laboratory test. It simulates closed-circuit grinding with a 250 per cent circulating load. Moore (1995) describes the procedure, which yields a Bond Work Index: the kWh/t required to reduce a material from infinite size to 80 per cent passing 100µm (Section B4.8). Section B7.2 lists some values, and a number of factors that affect clinker grindability are given in Sections 6.2 and 7.5.

Other tests in use include those of Schiller and Ellerbrock (1994), and Theisen (1993).

Coal grindability is conventionally tested using a Hardgrove mill (ASTM D409). Section B7.2 lists some data. The Zeisel test, apparently providing useful results for VRM design, applies a different test procedure to the Hardgrove apparatus.

Note that most grinding tests involve screening the product at 45µm or 75µm. 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 close to the test product fineness. It is also possible that certain materials, such as petcoke, may not give rise to the crystalline fractures expected by the tests – a certain malleability may occur which, again, may significantly distort 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 -2.36/+0.3mm 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.

7.10 Physical testing

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

Different test methods, particularly for strength, can give widely different results from a single cement. Therefore, 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 laboratory engage in a programme of comparative testing with others (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.

Much automated physical testing equipment is now commercially available and is reviewed by Morrison and Walden (2006) and Gardiner (2011). The use of microprocessors, computer-controlled systems, robotics and optics allows increased precision and accuracy in testing as well as greater laboratory efficiency, and can improve reproducibility and reduce variability. Other benefits may include improved workflow and reduced turnaround time and training costs. Such automation is increasingly being accepted by the ASTM. An automated Vicat time-of-set test has been approved as an alternative method which must be calibrated against the manual method (ASTM C191), however, no automated version of the Gillmore needle test has yet been approved.

The use of automatic tamping machines and vibration tables is allowed for making cubes for mortar compressive testing (ASTM C 109) – again, calibrated against the manual methods. Automatic Blaine testers have been used for online or offline quality control for many years and ASTM C204 allows the use of devices that have demonstrated acceptable performance. Automatic mortar and paste mixers provide cycles conforming to CEN and DIN methods, and an automatic air-void image analysis method for hardened concrete has been developed, but these are not yet approved by ASTM.

Accelerated test methods have long been sought and CST Instruments Ltd (2017) describe a new hand-held test device for use in the field that measures electrical conductivity after 50s of hydration, using 15g of cement and 0.5l of de-ionised water. The test result, along with parameters from the cement datasheet, is used to predict compressive and flexural strengths at seven and 28 days. The UK Accreditation Service (UKAS) and others have confirmed that the results are typically within 5-7 per cent of those measured under EN 196-1 standards, so aiding the development of new concrete mixes and allowing faulty batches of concrete to be rejected or diverted to less critical work.

7.11 Process control analysis

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

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

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

For convenience, when monitoring deviations and calculating statistical variation, process control usually employs composite terms such as Bogue compounds, lime saturation factor, silica ratio, etc, rather than individual oxides.

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

■ quarry drill cores	chemical analysis, including S and Cl, every 2m
■ blast hole cuttings	chemical analysis of composite of each blast
■ imported raw materials	chemical analysis of shipment composite
■ raw mill product	chemical analysis of one- or two-hour composite – 75µm screening of eight-hour composites. Raw mills are increasingly controlled by online analysis
■ kiln feed	chemical analysis of four- or eight-hour grab samples
■ hot meal to kiln	K_2O, Na_2O, SO_3, Cl and LoI (800 °C) on four- or eight-hour grabs
■ clinker	complete chemical analysis including SO_3 and free lime on two- or four-hour grabs and on daily composites
■ cement mill product	Blaine and SO_3 on two-hour composites, complete chemical, gypsum dehydration and physical tests on daily composites
■ cement shipped	7- and 28-day strength testing on two or three grabs per week per national standards.

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

7.12 Chromate passivation

Since 2005 an amendment to European Union Directive 76/769/EEC has required that cements, when hydrated, must contain less than 0.0002 per cent-soluble chromium VI (Cr^{6+}). This originated from concern over 'chromium dermatitis' or 'cement eczema' (Kerton, 2003). No doubt other regions of the world will follow suit in due course, although only a very small proportion of people are susceptible to sensitisation. Not all of these are allergic to chromium in cement and not all of them do get sensitised. Further, many of those who are affected could have avoided the problem by following guidance provided in product health and safety data sheets for reducing irritation risks. Sensitisation arises from prolonged exposure to very small quantities of soluble chromium and once produced, irritation will persist, even if chromium levels are reduced.

The process involves the soluble form of chromium, Cr^{6+}. Ferrous sulphate added to cement reduces the chromate completely, precipitating Cr^{3+}. An amount of 0.35 per cent by weight of ferrous sulphate, $FeSO_4 \cdot 7H_2O$, is enough to reduce 20µg Cr^{6+}/g cement with no side effects on concrete. The sulphate would preferably be added to cement when there is a pending potential intimate contact with skin, eg, at floor laying, repairs and hand-made casting, but the legal requirement is to add the agent to cement prior to dispatch, so that that the Cr^{6+} content is limited to 2ppm of dry weight for all cement available on the EU market (except if used in closed manufacturing systems). EN 196-10 provides an analytical test method.

Ferrous sulphate is widely used in various hydrated crystal forms, perhaps mixed with a filler to assist handling and dosing. However, it is readily oxidised, compromising its long-term efficacy. High doses increase water demand and can be involved in surface staining of concrete. In recent years enhanced grades have been introduced, characterised by better flowability, lower acidity, better packing behaviour and superior shelf-life. Alternatives are available but tend to be more costly (Stopps, et al, 2011).

7.13 Cement quality

Standardised test methods specified for a range of cement and concrete properties are outlined by Rivera (2018). These comprise the compressive strength of cement-sand cubes or prisms; tensile strength and fineness; consistency, setting time and flow; soundness and expansion.

To produce high-quality cement of low variability, it is essential to have uniformly-burned clinker of consistent chemical composition. Strength development of mortar and concrete depends upon six principal cement 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.

Achieving a target mortar or concrete strength is the culmination of establishing the correct chemical and physical composition of raw materials, mix design and process control, and should not be related solely to finish milling parameters (Gebhardt, 1995).

However, poor control of cement milling can negate all the earlier measures due to problems with fineness, PSD, sulphate level and the forms of calcium sulphate present and their degree of dehydration. The simple means of increasing early strength in laboratory tests is to grind cement finer. However, this may have a negative effect upon concrete strength due to increased water demand.

C_3S primarily governs strength gain up to 28 days while C_2S has more effect later. Work reported by Hewlett (2003) indicates that at six months, strength is largely independent of C_3S:C_2S proportions. Higher SO_3 tends to increase 28-day strength and higher alkalis tend to increase early strength and reduce 28-day strength.

Multiple regression analyses (MRA) using data from round-robin tests on a large suite of all types of Portland cements from different sources, provide the data in Table 7.4, showing the effect of changing a quality control parameter on C109 mortar cube strengths at different ages.

Table 7.4 Correlations of C109 cube strengths with various parameters

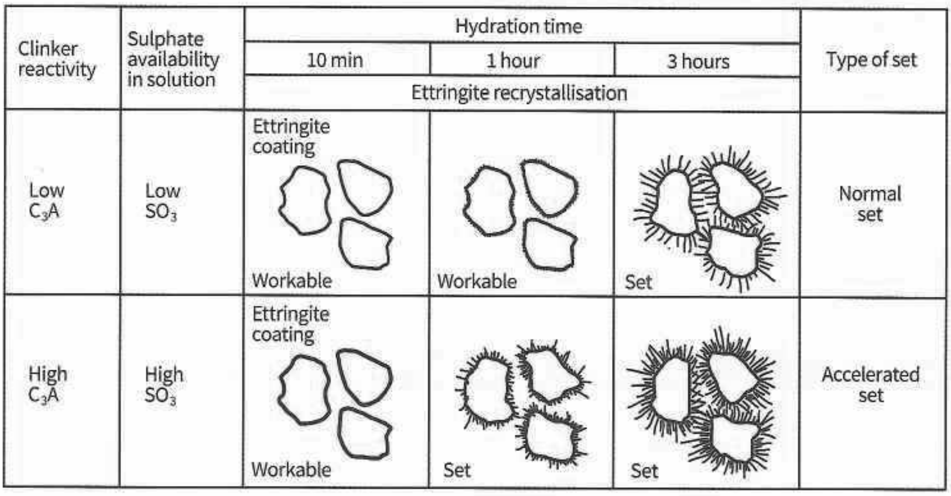
Parameter	Change	Effects on C109 mortar cube strengths (MPa)			
		1-day	3-day	7-day	28-day
Blaine (m^2/kg)	+20	0.9	0.9	0.6	0.5
-45µm (%)	+2.0	0.0	0.4	0.7	0.8
Na_2O eq (%)	+0.1	1.2	0.8	0.4	-0.8
SO_3 (%)	+0.5	0.9	1.0	1.2	1.0
Free lime (%)	+0.3	-1.2	-0.5	-0.4	-0.6
C_3A (%)	+1.0	0.0	0.1	0.0	0.0
C_3S (%)	+5.0	1.5	0.7	0.0	0.0
LoI (%)	1.0	-0.8	-1.0	-1.2	-1.0
Mortar air (%)	1.0	0.0	0.0	-0.3	-0.5

The above correlations do not include the effects of clinker burning conditions, clinker mineralogy, forms of calcium sulphate and particle size distribution. These parameters are also significant but are not routinely monitored and controlled at all cement plants. Note that similar exercises on different sets of data yield different coefficients, therefore it is important to obtain and use results obtained from data that are relevant to the problem being addressed. Each manufacturing plant should analyse blocks of, say, 40 sets of data to determine a robust correlation that can be used for predictive purposes and then maintain a watch on the coefficients to determine if updating is required.

7.14 Setting time

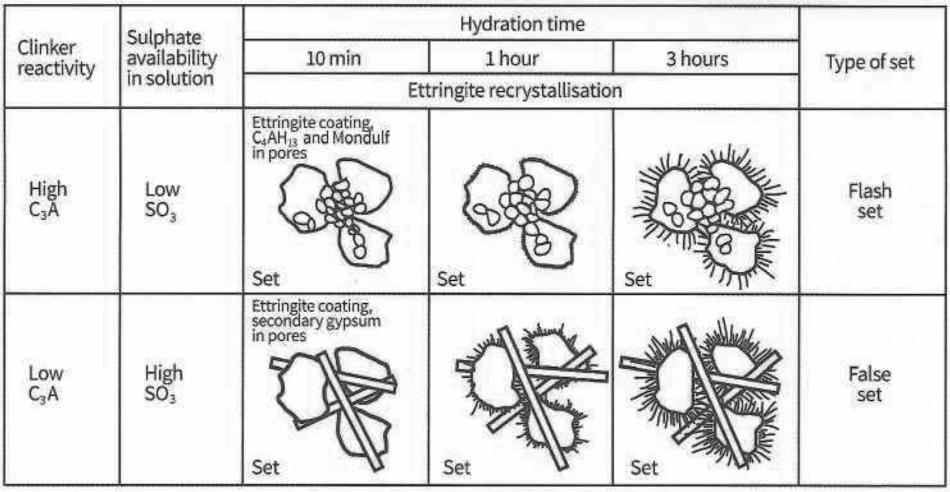
Setting of cement is caused by the development of cross-linking structure of hydration products shortly after the dormant period (see Chapter 10). This is illustrated by Figure 7.3 from Locher et al (1980). Good control of setting is achieved by matching C_3A reactivity with soluble-sulphate availability.

Figure 7.3 Cross linking of hydration products



Unbalance between C_3A reactivity and sulphate availability can cause flash set or false set as shown in Figure 7.4.

Figure 7.4 Flash and false set



Clinker reactivity	Sulphate availability in solution	Hydration time			Type of set
		10 min	1 hour	3 hours	
		Ettringite recrystallisation			
High C_3A	Low SO_3	Ettringite coating, C_4AH_{13} and Mondulf in pores Set	Set	Set	Flash set
Low C_3A	High SO_3	Ettringite coating, secondary gypsum in pores Set	Set	Set	False set

As discussed in more detail under 'Mill Performance' in Chapter 6, the primary factors that affect normal setting time are free-lime and the forms of calcium sulphate (see Section B7.4). The secondary factors are fineness and C_3A content.

The effects of other parameters on Vicat initial setting time (VIST) can be appreciated by the following correlation, which indicates a significant effect from free lime:

VIST (min.) = 216 - 44 x (per cent free-lime) - 0.125 x (m2/kg) - 2.22 x (per cent C_3A)

7.15 ASTM Cement types and specifications

There are numerous national and international cement specifications together with the American Petroleum Institute specification for oilwell cements (API Specification 10A / ISO 10426-1), which has virtually universal recognition, even though now duplicated in the ISO 10426 series. The European specification for construction cements, EN 197, is quite different from those of the ASTM C150, C595, C1157, C91, C1328, C1329, CSA A3001, A3002. The respective cement types and associated testing methods are listed below.

It is a curiosity that the trading of clinker cannot relate to a general specification. Chemical ranges on the clinker and physical performance of the cement it produces, 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, off-specification 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 widely disparate results.

Table 7.5 ASTM C150 specification for Portland Cement

Type I	General purpose
Type II	Moderate heat/moderate sulphate resistance
Type III	High early strength
Type IV	Low heat of hydration (rare)
Type V	High sulphate resistance

A final designation, A, can be appended to most of the type numbers to indicate that air is entrained. (The optional label of "LA" for low alkali was dropped in 2014, as the previous edition of this handbook appeared.)

Table 7.6 Associated ASTM test methods and 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)
Values should always be checked against the current edition of each standard	

Table 7.7 Chemical limits associated with ASTM C150					
	I	II	III	IV	V
	IA	IIA	IIIA		
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
Insoluble residue – max (%)	0.75	0.75	0.75	0.75	0.75
C_3A – max (%)	8	15			
C_4AF+2C_3A (%)				25	
Alternative					
C_3A – max (%)				7	5
C_4AF+2C_3A (%)				25	
C_3S – max (%)				35	
C_2S – min (%)				40	
Optional					
C_3S+C_3A – max (moderate heat) (%)		58			
$Na_2O+0.658K_2O$ – max (LA) (%)	0.60	0.60	0.60	0.60	0.60
Values should always be checked against the current edition of each standard					

Table 7.8 Physical limits associated with ASTM C150 *(Note: A cements have lower strength requirements)*

	I	II	III	IV	V
	IA	IIA	IIIA		
Air content – max (%)	12	12	12	12	12
(A) – max (%)		22	22	22	
– min (%)		16	16	16	
Fineness – min (m²/kg)	260	260	260	260	260
Autoclave exp – max (%)	0.80	0.80	0.80	0.80	0.80
Strength – min (MPa)					
1-day			12.0		
(A) 1-day			10.0		
3-day	12.0	10.0	24.0		8.0
(A) 3-day	10.0	8.0	19.0		
7-day	19.0	17.0		7.0	15.0
(A) 7-day	16.0	14.0			
28-day				17.0	21.0
Gillmore, IST/FST – 60/600 minutes for all types			Vicat, IST/FST – 45/375 minutes for all types		
Values should always be checked against the current edition of each standard					

Typical cement compound concentrations for the five cement types are shown in Table 7.9.

Table 7.9 Typical cement compound compositions

Cement type	ASTM C150/C150M-12	C_3S (% by mass)	C_2S (% by mass)	C_3A (% by mass)	C_4AF (% by mass)	Fineness (m²/kg)
I	General purpose	57	15	9	8	384
II	Moderate sulphate resistance	57	17	7	10	377
III	High early strength	56	16	8	9	556
IV	Low heat of hydration	42	32	4	15	340
V	Sulphate-resistant	58	18	4	12	389

Additions were formerly limited to limestone, water, calcium sulphate (and air entraining agent). At the option of the manufacturer, process additions (primarily grinding aids) that meet C465 may be used. C465 requires comparative testing with/without additives and the difference shall not exceed:

1. water requirement for normal consistency shall not increase by more than one per cent
2. setting time (Vicat initial) shall not increase/decrease by more than the lesser of 1h or 50 per cent
3. autoclave expansion shall not increase by more than 0.10 per cent
4. grand average of strengths at all ages shall not decrease by more than five per cent
5. difference in SO_3 <0.3 per cent
6. difference in Blaine <13m²/kg
7. drying shrinkage <0.025 per cent.

Under C465, various additives such as blastfurnace slag have also been used and recent editions clarify this practice. Traditional organic processing additions can be used in amounts up to one per cent by mass and need to meet C465 requirements. Inorganic processing additions (eg, slag, ash, pozzolans) can also be used in amounts of up to five per cent, but if more than one per cent is used, testing in accordance with ASTM C465 is required.

White cement is a special Portland cement with very little ferrite phase (less than one per cent) and relatively high aluminate and silicates contents. Its whiteness results from keeping iron, manganese and chromium at very low concentrations, using gas or other low-ash kiln fuel, burning clinker under reducing conditions with water quenching and grinding the cement to a high fineness. Cement whiteness is impaired if Fe_2O_3 is >0.50 per cent, Mn_2O_3 >0.02 per cent and Cr_2O_3 >0.01 per cent. The use of pure sand and limestone and the low clinker liquid phase require clinker burning at 1600 °C, usually with a mineraliser (CaF_2). 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, 2000, Clark, 2001 and Schulz, 2003).

Various pastel-coloured cements can be made by mixing or, preferably, intergrinding pigments with white cement (Bensted, 1993). Darker colours such as red and brown may 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 per cent by weight of cement, are the oxides of iron, manganese, chromium and cobalt, as well as carbon black. Grey cements of equal quality may produce concrete of darker or lighter colour, depending upon the levels of iron and other elements that are present.

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 per cent limestone is incorporated together with an air-entraining agent. Masonry cement is typically ground to 5000-6000cm^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.

7.16 European EN 197 cement specification

Table 7.10 EN 197 – 1:2011 Cement – Part 1: Composition, specifications and conformity criteria for common cements

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

CEM III	Blast furnace cement	CEM III/A	35-64	35-65	0-5
		CEM III/B	20-34	66-80	0-5
		CEM III/C	5-19	81-95	0-5
CEM IV	Pozzolanic cement	CEM IV/A	65-89	11-35	0-5
		CEM IV/B	45-64	36-55	0-5
CEM V	Composite cement	CEM V/A	40-64	18-30 + 18-30	0-5
		CEM V/B	20-39	31-50 + 31-50	0-5

Siliceous fly-ash is <10 per cent CaO; calcareous fly-ash >10 per cent CaO.
Limestone L has <0.5 per cent organic carbon; LL has <0.2 per cent organic carbon.
Composite cements include both slag and silica-fume/pozzolan/fly-ash.
Values should always be checked against the current edition of each standard

Table 7.11 EN 197.1 physical requirements

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

N = Ordinary cement   R = High early strength cement
Values should always be checked against the current edition of each standard

Table 7.12 EN197.1 chemical requirements

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

Values should always be checked against the current edition of each standard

Table 7.13 Test methods associated with EN 197

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

Values should always be checked against the current edition of each standard

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

This standard specifies the scheme of evaluation of cements against their corresponding product specifications in EN 197-1. The standard 28-day strength classes generally have upper limits as well as lower. A cement manufacturing facility must implement a documented production quality management system quite similar to ISO 9000 but with additional auto-control of released cement. Auto-control rules are laid out for sampling, preparation, retention, testing and statistical evaluation of results. Criteria are set for conformity of cement sampled at the point of loading to the applicable standards: 95 per cent of results should be above the auto-control minimum strength and 90 per cent below the maximum. Specified corrective actions must be taken in case of nonconformity. If in conformity, the product is certified and the CE mark is applied to the product. ASTM C183 is somewhat similar to EN 197-2 but not as comprehensive and not mandatory.

7.17 Composite cements (intergrinds or blends)

As discussed in the previous chapter, along with natural pozzolans, there are two by-product materials that are commonly blended or interground with cement under ASTM C595: pulverised coal fly ash (pfa) and ground granulated blastfurnace slag (GGBS). Silica fume, a byproduct of electric-arc manufacture of ferro-silicon and silicon metal, is also used to a much lesser extent, and rice husk ash may be an alternative to pfa in rice-growing countries.

In many cases, composite cements appear slightly pale and yield concretes with a lighter colour: this is not indicative of strength and surface finishing techniques can play a more prominent role in governing appearance.

The non-clinker materials are not themselves cementitious but latently hydraulic, ie, reacting with lime released during cement hydration to form compounds with cementitious properties. They are widely available and exploited in certain areas but not worldwide. Snellings et al (2012) provide an extensive review of the types of naturally-occurring mineral that exhibit pozzolanic properties, and their evaluation is discussed by Morrical et al (2011). Although generally slower to develop strength than straight Portland cements, the composite cements reduce concrete porosity 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.

Some minerals are calcined to produce "artificial pozzolans", and Norwegian studies have demonstrated scope for alternative chemical treatment of olivine and serpentine, avoiding the use of fuel. (Justnes, 2009).

Although the term 'pozzolan' is often used generically in standards for any active mineral addition, pozzolans should ideally be distinguished from hydraulic materials such as ground granulated slag. These latter react directly with water to form cementitious compounds, while pozzolanic materials, in the presence of moisture, react with calcium hydroxide (either a reaction product or added directly) to form compounds possessing cementing properties. Blastfurnace slag, for example, is hydraulic if activated by sodium hydroxide, silicate or calcium sulphate.

ASTM C595 requires that the manufacturer shall, on demand, state the source and amount of pozzolan added. Under both ASTM and EN standards, a merchant is free to buy pure Portland cement and blend in appropriate minerals to produce composite cements that comply with one or more of the specifications.

Although C595 refers only to blended cements, a common practice is to add the same pozzolanic materials at the concrete batching plant. Pozzolanic materials thus employed are covered by ASTM C618 (pfa and natural pozzolan) and C989 (blastfurnace slag). EN 15167 covers the use of ground slag as a concrete addition, and EN 13263 covers silica fume. (Note that unlike Europe, in the USA the term 'slag cement' usually means 'ground granulated blastfurnace slag', not a composite cement!)

Natural pozzolans are usually soft to grind. ASTM C618 requires that SiO_2+Al_2O_3+Fe_2O_3 >70 per cent and addition rate is 15-40 per cent. A significant safety margin has to be adopted during manufacture to allow for the variability of properties for naturally mined materials. Rapid determination of a hydraulic activity index allows the margin to be reduced. Velásquez et al (2018) indicate how XRD techniques can be modified to determine the amorphous phase in pumiceous materials to provide a more precise indicator than measurement of acid-insoluble residue or quartz content.

Fly ash is classified by ASTM C618 as 'Class F' with SiO_2+Al_2O_3+Fe_2O_3 >70 per cent and 'C' if >50 per cent. Some Class C ashes contain sufficient CaO to be appreciably cementitious. Addition rate is 15-40 per cent of total, and a 45μm residue of below 20 per cent is preferred if the ash is to be blended rather than interground. Ash is increasingly liable to contain unburned carbon due to the use of low-NO_x burners in power boilers. For intergrinding or mixing in concrete, ASTM C618 requires that LoI (effectively due to carbon) should be less than six per cent.

High carbon levels are undesirable for aesthetic reasons, as the surface colour of concrete is affected, and also for technical reasons, as fine carbon interferes with the action of admixtures. Sometimes size separation of ash can improve quality where carbon is mainly in unburned material in the coarsest particles. Other technologies include froth flotation and drying, magnetic separation and electrostatic separation. High-carbon ash may be suitable as a kiln raw material. Several beneficiation processes are in commercial operation.

In addition to carbon content, there is also increasing concern over the presence of ammonia from 'slip' during injection to abate NO_x emissions. Above 50ppm, such ash may be difficult to use in cement and concrete due to the objectionable odour that arises during concrete production and use, and which can persist for many weeks in poorly ventilated indoor locations. Limits for safe ammonia concentrations in ambient air, which vary from one country to another, rarely appear to be exceeded beyond a few hours. But statutory limits tend to drop, and ammonia removal options are on offer.

Fly ash is sometimes classed as an 'artificial pozzolan', and its properties are addressed in EN 450, whilst its addition at the concrete mixer is covered in the UK within 'BS 8500: Concrete - Complementary British Standard to BS EN 206-1'. In contrast to much of the rest of Europe, UK users have preferred classified fly ash, with the coarser particles removed, as this provides greater consistency between deliveries, improved water reduction properties and consequently better strength performance in mortar and concrete.

Typical compositions, with wide variations, are shown in Table 7.14.

Table 7.14 Typical compositions

	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

Slag supplies should have a consistent moisture content, suited to the drying capacity of the equipment in use, and chloride content may need to be limited and monitored for some sources. The activity of slag increases with the ratio (CaO+MgO)/(SiO_2+Al_2O_3) and with glass content – however, the two appear to be inversely related. Addition rate is typically 25-70 per cent.

The use of direct additions to the concrete mixer is governed by different principles or concepts within the CEN member-states. The prerequisite for these is conformity of the addition with one of the harmonised European product standards listed in Table 7.15. As a descriptive regulation – ie, without further tests apart from the attestation of conformity of the concrete – the k-value concept enables part of a stipulated addition content to be taken into account in the (equivalent) water-cement-ratio or the minimum cement (hydraulic binder) content. EN 206 suggests the k-values listed in Table 7.15 when using CEM I or CEM II/A.

Table 7.15 Suggestion for the k-value concept according to EN206

Concrete addition	Permitted cement types acc. EN197-1 (c)	k-value	Additional quantity which can be taken into account to $(w/c)_{eq}$
Fly ash (f) acc. EN450-1	CEMI	0.4*	$f/c < 0.33$
	CEMII/A		$f/c < 0.25$
Silica fume(s) acc. EN13262-1	CEMI and CEMII/A (without CEMII/A-D)	2.0* for $(w/c)_{eq} < 0.45$ 2.0* for $(w/c)_{eq} > 0.45$ except for exposure classes XC and XF, where $k_s = 1.0$	$s/c < 0.11$
Ground granulated blastfurnace slag (h) acc. EN15167-1	CEMI and CEMII/A	0.6**	$h/c < 1.0$

**maximum value*
***recommended value*

Note that objectors suggest that the k-value approach does work for easy cases with limited levels of replacement by well-known reactive additions but fails for more complex cases (eg for ground slags with quality that significantly exceeds the demands of EN15167) and when concrete properties other than strength are of concern (eg durability).

Limestone is another common interground additive. The main technical argument for including minor amounts in Portland cement is that when cement standards have more than one cement strength class, with both upper and lower limits, then the lowest class of cement can be produced at a fineness which confers acceptable workability, finishability and bleeding characteristics to the concrete, compared to those obtained from a coarsely ground non-limestone cement of the same class. Limestone with a high iron content is best avoided, as it imparts a pink colour to concrete. When used at above five per cent addition level, standards, including the 2012 revision of ASTM C595, require a calcium carbonate content above either 70 per cent or 75 per cent, limit the acceptable level of total organic carbon to below 0.2 per cent and the methylene blue adsorption level (an indication of fine clay content) to below 1.2g/100g (Tennis, Thomas and Weiss, 2011).

ASTM C595 Specification for Blended Hydraulic Cements

Pozzolan or slag used for intergrinding must satisfy requirements for pozzolanic activity - primarily that a standard intergrind shall have not less than 75 per cent of the 28-day strength of the control cement mortar.

Types covered are shown in Table 7.16, with chemical and physical limits listed in Tables 7.17 and 7.18.

Table 7.16 Cement types covered in ASTM C595

Type IS	Portland blast-furnace slag cement - up to 95 per cent slag permitted
Type IP	Portland-pozzolan cement - up to 40 per cent pozzolan permitted
Type IL	Portland-limestone cement - up to 15 per cent limestone permitted
Type IT	Ternary blended cement - up to 70 per cent of pozzolan + limestone + slag, with pozzolan being no more than 40 per cent and limestone no more than 15 per cent

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

Table 7.17 Chemical limits for cement types covered in ASTM C595

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

Table 7.18 Physical limits for cement types covered in ASTM C595

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 (min)	45	45	45	45
FST – max (h)	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, later revised to include Portland cements. Adoption was slow, but it is intended that performance-based specifications will eventually displace the present mix of compositional and performance criteria.

7.18 Supersulphated cement

Highly resistant to seawater, sulphate attack and acid attack, this cement is made by intergrinding 80-85 per cent GBS, 10-15 per cent anhydrite and five per cent Portland cement clinker to about 450m^2/kg. Ettringite is the primary hydration product, which develops strength. Therefore, a slag high in alumina (15-20 per cent) is preferred. Some European standards require a minimum slag content of 75 per cent (Hewlett, 2003 and Taylor, 1997).

7.19 Calcium aluminate cement (CAC)

A cement yielding rapid strength development and high durability in aggressive environments can be made using a mix with high alumina and high iron contents. The typical composition of "Ciment Fondu" is shown in Table 7.19.

Table 7.19 Typical Ciment Fondu composition

Compound	Share in cement (%)
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 with bauxite is charged to a reverberatory open-hearth furnace, preheated and calcined by combustion gases before passing into the hearth section of the furnace, where it is melted above 1450°C and homogenised. The molten product is continuously withdrawn by means of a pan conveyor where it cools and solidifies as clinker. The clinker is crushed and ground to 250-400m^2/kg. No set regulator is added during grinding. Bauxite, fuel and power costs are all high. Other high-alumina cements with 70-80 per cent 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. Typical chemical compositions are shown in Table 7.20.

Table 7.20 Typical chemical compositions for CAC

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

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

Before the hydration sequence of aluminous cement became well understood, there were a few well-documented failures of buildings constructed with high-alumina cement concrete beams. The major phase of the cement, CA, initially hydrates into metastable CAH_{10}, and eventually converts into stable C_3AH_6. This conversion is accompanied by a reduction in volume of the hydrate and the evolution of water. This increases porosity of the hydrated mass and reduces strength. Failure can be prevented by following guidelines upon water/cement ratio and minimum cement content. A low water/cement ratio is advised because unhydrated 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.

Aluminous cement provides rapid setting and strength development (one-day strength higher than Portland 28-day), good resistance to abrasion, to high temperatures and to sulphate and weak acid attacks (Clark, 2002). The disadvantage is price, which is 4-5 times that of Portland cement. The primary application of CAC is for refractories. Combined with Portland cement, even more rapid setting can be achieved and secondary uses for CAC are as a component of products used in concrete repair, self-levelling floor screeds, fireplaces, sewer pipe lining, mining and fertilisers.

7.20 Shrinkage-compensating cements (SCC)

Shrinkage-compensating cement (SCC) is used to minimise or eliminate the cracking of Portland concrete due to drying shrinkage. This is achieved by adding more calcium sulphate and an agent that produces controlled expansion upon hydration. The most widely-used expansive agents are calcium sulphoaluminate 3CA. $CaSO_4$ ('Klein's compound' or 'ye'elimite'), CA, $C_{12}A_7$, and C_3A. On hydration, these compounds all expand due to formation of ettringite ($C_3A.3CaSO_4.32H_2O$).

Although three varieties of expansive cement are designated in ASTM C845 as K, M, and S, only type K is widely available in the United States: this contains Portland cement, Klein's compound, calcium sulphate and uncombined lime.

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

The composition of kleinite-expansive clinker can be varied considerably to utilise the raw materials most economically available – usually gypsum, bauxite, limestone or industrial byproducts. The raw mix can be sintered in a rotary kiln at temperatures above 1300 °C. Clinkers made in the USA typically contain 8-50 per cent Klein's compound, with anhydrite, alite, belite and some free lime. (Hewlett ibid, p830; Taylor ibid, p31). If 10-20 per cent of expansive clinker is ground with 75 per cent Portland cement and 10 per cent GBS, the shrinkage of Portland cement is compensated. Both proportioning and curing are critical (Zivica, 1999).

Approximately 90 per cent 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 223R, 2010).

Table 7.21 lists typical compositions of Type K SCCs made in the USA (Taylor ibid, p314):

Table 7.21 Typical compositions for Type K SCCs

Compound	#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, which has limited applications, is made from Portland cement clinker with a high C_3A content.

Several industrial trials have produced cements with a low CO_2 footprint, combining an aluminate cement portion and a belite portion, respectively providing early and late concrete strengths. These use conventional kiln and mill systems in the hope that the technology may be readily transferred to full-scale operations.

7.21 ISO 9001:2000 Quality management system

ISO 9000 deals with the fundamentals of quality management systems, including the eight management principles upon which the family of standards is based. ISO 9001 deals with the requirements that organisations wishing to meet the standard have to fulfil and it is this latter document that principally concerns cement plants requiring quality accreditation.

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

Hoyle (2003) condenses the process into five linked tasks:

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

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

The process has been criticised as wasteful and irrelevant, but a number of claims are made for the efficiencies and cost savings which stem from implementation, though not all of these are 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 pre-dated formalised quality management. More persuasive, however, is a recognised worldwide trend towards adoption of ISO 9000 with the increasing likelihood that major engineers and contractors will require certification of cement manufacturers to meet standard procurement terms. The worldwide total of ISO 9000 certificates is now over 1.6m, including over 0.35m in China.

7.22 Concrete problems

Concrete must possess strength and durability in the hardened state and, after mixing, fresh material must have suitable workability to be placed in formwork and consolidated. Workability is not only concerned with confirming that the mix is cohesive and mobile, so as to be placed and cast into the required shape. It also considers the stability of the mix (avoidance of bleeding and segregation during transport and placing), its compactability after placing and its ability to provide a satisfactory surface finish.

The design of concrete mixes and their correct preparation is an extensive and complex subject, not of direct concern to staff engaged in cement production. The manufacture of cement is more sophisticated than the production of concrete and current cement quality demanded by the market is substantially higher than specification minima. However, while cement as shipped is seldom out of specification, variation in properties due to inadequate materials and process control can seriously compromise concrete quality. Cement manufacture allows 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 results are not usually available until after the concrete has been placed, so that cement variability can only be allowed for 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, materially reducing the cost of production.

Virtually all concrete mixes involve additives to modify cement performance and cement-admixture incompatibility is a potential problem. The true end customers are neither the concrete producers, nor the builders, architects, civil engineers and specifiers who need to understand how to get the best out of this versatile product, but the people who have to live with concrete for many decades after it has been produced. It is believed that close technical communication between cement and concrete producers is invaluable for all parties.

A typical concrete mix may be:

- water: seven per cent (by weight)
- cement: 13 per cent
- fine aggregate, -6mm (dry): 32 per cent
- coarse aggregate, +6mm (dry): 48 per cent
- water reducer : 0.25 per cent 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 and Wilson (2011). Admixtures are considered by Ramachandran (1996). They fall into several groups, which are self-explanatory:

- air-entraining agents
- accelerators
- water reducers ('plasticisers') and high-range water reducers ('superplasticisers')
- set retarders
- foaming agents, defoamers, pumping aids, shrinkage reducers, etc.

Air-entraining agents may be added either by intergrinding in cement or directly to the concrete mix. Desired concrete air content is usually 4-7 per cent by volume, the higher contents being employed where freeze-thaw cycles are expected. Increasing air content, however, yields lower strength. The basic materials are either soaps from natural resin or, to a growing extent, synthetic active ingredients. After site reports of excess air entrainment, laboratory tests demonstrated that, with natural active agents, air content was stable after a mixing time of about two minutes at the most. With synthetic agents significantly longer mixing times were necessary to obtain constant air content. It is recommended that the admixture quantity determined in initial performance tests should be doubled in an additional test carried out over both short and long mixing periods (say 30s and 5min). If there is a substantial rise in air content with the double addition quantity and extended mixing time, there is a high risk of a subsequent rise in air content of fresh concrete during construction work (ECRA, 2011).

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 (ASTM C563) is the level which maximises mortar strength at one day. However, higher SO_3 contents are required to maximise strengths at later ages or at elevated temperatures. Higher cement fineness, C_3A and alkali contents also demand more SO_3 for maximum strength. Concrete containing water reducer usually requires a higher optimum SO_3 (Tagnit-Hamou and Aitcin, 1993). 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, 1995).

In general, concrete 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 per cent and more effective superplasticisers can reduce it by up to 30 per cent. 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.

It can be useful to carry out some tests of cements in mixes that mimic those of interest to major customers. Using different aggregates or producing cement-rich concrete can have a marked effect upon the choice between two competitive cements that have similar performance in standard tests. In the latter case, a relatively high-alkali content can be disadvantageous. What is most useful for major customers, however, is to minimise variability in performance.

Potential problems with concrete are numerous, and their investigation and solution is a specialist task. Many defects result from poor site workmanship, resulting in inadequate compaction and cover of reinforcements, or from lack of attention to architectural design details, perhaps resulting in persistent local drips and surface discolouration from rainwater. However, as cement is frequently, and usually incorrectly, blamed for defects, some common areas of contention may be summarised.

Cement deliveries

'Hot cement' is usually an irrelevant complaint: cement over about 60 °C can be unpleasant to handle but contributes little heat to concrete. (Heat generated by hydration in mass pours is a different issue, reflecting the type of cement.)

Pack-set is a phenomenon of electrostatic charging of particles which, upon storage and compaction (particularly in bulk distribution vessels), results in resistance to flow. Another cause of pack-set is dehydration of gypsum in hot cement resulting in syngenite formation and partial cement pre-hydration. Unlike hydration, however, once pack-set is broken, the cement flows freely. Electrostatic charging is prevented by use of cement grinding aids. A standard method for the determination of pack-set was available from WR Grace and Co, and this has become ASTM C-1565. Thermal scanning techniques (DTA/DSC/TGA) are useful for investigating gypsum dehydration and syngenite formation.

Lumpy or low-strength cement is usually caused by protracted or improper storage allowing partial hydration and carbonation. Lump formation may be associated with the presence of K_2SO_4, and – in contrast to pack set – grinding aids are not likely to alleviate the problem.

Fresh concrete

Flash set and false set both arise from incorrect gypsum dehydration in a hot mill and are discussed in Section 6.2 and in the paragraph on setting time above. Whilst flash set is irreversible, false set is generally of little practical consequence except in a few central ready-mix concrete plants that use very short mixing times, though it is worth noting that aerated or carbonated cement has a strong tendency to false set, along with strength loss, and setting time is often lengthened. The standard ASTM penetration test methods for early stiffening are C359 for mortar and C451 for cement paste. Although C451 is the optional test under C150, it is widely held that it has little relationship to field performance of concrete. C359 using mortar gives better correlation, though it is sensitive to water/cement ratio and to mortar temperature. Note also, that false set or flash set may be caused by certain concrete admixtures, particularly water reducers.

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

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

- insufficient sulphate and high alkali contents in cement
- accelerated formation of ettringite
- an excess of soluble sulphate causing gypsum precipitation
- inadequate C_3A to control sulphate released into solution.

Mature concrete

Low strength of concrete cylinders can result from:

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

Cement content of hardened concrete can be confirmed (ASTM C1084) by arithmetical comparison of CaO or SiO_2 analyses of its constituents.

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

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

Alkali-silica reactivity (ASR), is a reaction between alkali in cement and some sensitive aggregates which has led to the habitual specification of low-alkali cement by engineers in some countries. Reactive aggregates are not ubiquitous and the unnecessary specification of low-alkali cement frequently entails additional production cost and the landfill of process dust. The 0.6 per cent (ASTM C150) limit may be unnecessary for many aggregates and too high for others. Certain pozzolan and slag blends can be effective in reducing the reaction.

Cracks – if deep can be due to:

- high slump concrete with consequent high shrinkage
- poor aggregate gradation
- rusting of re-bar too close to surface
- structural settlement from lack of footings
- restrained concrete or inadequate relief joints
- alkali-aggregate reactivity
- freeze/thaw of non-air-entrained concrete.

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

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

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

Cracks – surface may be due to:

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

Delayed ettringite formation (DEF) has achieved some notoriety. The most common occurrence has been in heat-cured precast concrete components, such as railway sleepers. Ettringite, a normal hydration product, forms early in cement hydration, leading to setting and hardening. Under certain circumstances, principally curing temperatures above about 70°C, ettringite formation is prevented. Cooling followed by prolonged exposure to moisture can cause the slow formation of ettringite crystals causing expansion and cracking of hardened concrete, but structural collapse without warning is most unlikely. DEF can, however, increase the risk of secondary forms of deterioration such as freeze/thaw attack or reinforcement corrosion.

The search for causes has become somewhat emotional, but there is little evidence that this is the result of anything but incorrect concrete practices (Stark and Bollmann, 20001 and 20002). Where curing is likely to take the concrete temperature above 70°C the potential for DEF should be considered. Limiting the internal concrete temperature to 70°C during its early life will minimise the risk of subsequent DEF.

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

Attack on hardened concrete by sulphate from external sources often leads to expansion, cracking and spalling. In more advanced stages of attack, concrete may be softened and disintegrated. Sulphates may come from ground- or seawater, soils, road de-icing salts and from flue gas emissions, and result in both chemical reactions and physical processes. Incoming sulphate reacts with calcium hydroxide to form calcium sulphate, which, in turn, reacts with hydrated calcium aluminates to form ettringite in expansive reactions (Lea ibid, p. 310). Magnesium sulphate is more aggressive than sodium or calcium sulphate since it also reacts with hydrated calcium silicates to form gypsum and magnesium hydroxide. Long-term concrete exposure studies conducted by PCA indicate that sulphate attack under alternating wet and dry conditions is particularly destructive due to crystallisation pressure. In general, Type V Portland cement, blended cements or high-alumina cements should be used to improve sulphate resistance of concrete.

Brown surface discolourations are caused by a very small concentration of Fe^{2+} ions which diffuse from inside concrete through capillary pore solution to the surface and are retained there in the lime efflorescence (calcite crystals). The Fe^{2+} ions can originate from any of the components of concrete. However, in addition to Fe^{2+} ion concentration, many simultaneous conditions are required for discolouration to occur, involving curing conditions, pore size, clinker chemistry, non-clinker cement components and concrete additives, especially alkanolamines (used to reduce pack set and enhance strength). To reduce the potential for discolouration, it is suggested to:

- produce a dense concrete
- use cement containing 30-40 per cent slag
- use cement which has been burned harder to increase C_3S formation at the expense of C_2S and free-lime.

A new test method has been developed to test the brown discolouration potential of cements (Haerdtl et al, 2003). Mapei have proposed a novel blend of alkanolamines that cuts down the level of staining in laboratory tests (Recchi et al, 2013).

Surface dusting of floor slabs is caused by:

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

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

- unsound aggregate
- freezing and thawing
- premature finishing
- excessively fine aggregate (-150µm).

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

Whitehead and Schöne (2013) point out that standards in Ghana were preventing the use of dolomite despite the availability of widespread deposits, partly due to concern over stability in alkaline conditions. Test data have now resulted in amendment of the Ghanaian standard to permit the use of dolomite or dolomitic limestone, requiring that either $CaCO_3$ content is above 65 per cent or the sum of $CaCO_3$ and $MgCO_3$ is above 70 per cent.

8. Maintenance

Maintenance ranges across chemical, electrical, mechanical, civil and structural engineering, involving numerous arcane skills. The activity may be regarded as a production cost, but it is essential if output and quality are to be assured: improving maintenance practices offers scope for significant savings.

No attempt is made here to address the subject in practical detail. Instead, this chapter presents general concepts, recognising that there are more failure modes than old age and that appropriate analysis of equipment can lead to both greater reliability and reduced maintenance cost.

An academic paper by Shafeek (2012) provides a survey of industrial practice as a background to the study of a Saudi cement plant. The cement industry application of planned maintenance is reviewed by Patzke and Krause (1994), condition-based maintenance by Rudd and Wesley (2003) and plant engineering by Guilmin (1994). Meanwhile, useful comments on the current scene can be found in 'Plant Services Magazine'.

8.1 Maintenance benefits and costs

Maintenance typically represents 15-25 per cent of total manufacturing cost, with best practice said to achieve 10-15 per cent or less. Since the 1970s maintenance departments have had to cut costs whilst increasing plant reliability. This has resulted in smaller staff with multiple craft skills, and an increasing use of information and measurement technologies, aided by computer technology.

It can be a 'Cinderella' area of manufacturing, where funds disappear into a 'black hole', with little feel for cost effectiveness or the function's overall efficiency. Maintenance activity is complex and difficult to get a grip on, and its costs do not often get the same level of attention as energy. However, effective maintenance management is an area for gaining significant competitive advantage.

Maintenance obviously costs money, needs people and requires discipline, but there are rewards. Take a 'typical' 0.83Mta kiln: the most important things to avoid are kiln stops. Consider two conditions, 'good' being 50 kiln stops/year and 7900h (90 per cent) run, and 'poor' being 140 stops/year and 7300h (83 per cent) run. Costs arise in three categories: power (assumed to be 50kWh/t for steady operation), fuel and lost production. Of course, some stops may not be due to inadequate maintenance: process upsets or poor inventory control can also be involved. Best operating practice maintains kiln operation at a level of 93-95 per cent YoY and, as cost increases when equipment is idle, plants with the highest run factors can register the lowest maintenance costs.

Some works, especially those with well-planned control rooms, run stricter discipline for switching off auxiliaries when the kiln stops, classed as 'good' power control. In this situation, good maintenance practice would consume 3kWh/t above the base level, compared to 8kWh/t for poor practice. For a plant with 'poor' power control, both these figures would rise by about 25 per cent.

Based upon typical kiln warm-up schedules, the additional fuel consumption in a poor maintenance regime would be about 50kcal/kg, compared to below 20kcal/kg in a good regime. The difference in lost production might amount to 90,000tpa. Adding all these factors, and assuming a sold-out market (and quite moderate profit margin and cost figures), the annual loss due to poor maintenance could easily exceed US$2.5m, without considering knock-on effects upon parameters such as refractory life.

Other areas can be subjected to similar approximate cost analysis, giving a 'taster' of the extremely high costs resulting from insufficient or poor maintenance. Even under low demand conditions the cost of unreliability is high. Yanusa-Katingo and Sinha (2013) examine data for a Nigerian cement plant, confirming that there were rewards waiting to be reaped from resources already invested in a condition monitoring system which had yet to yield sufficiently advance warnings of impending failures.

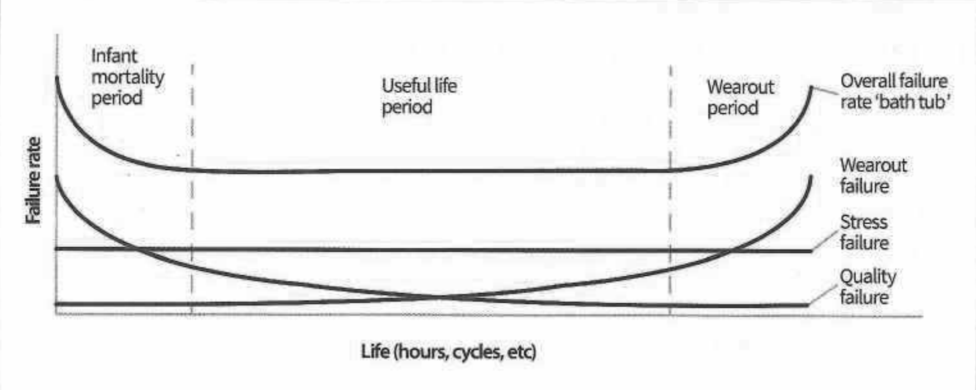
As with other areas of plant operation, maintenance can be outsourced to teams run by an equipment supplier, paying a yearly performance-based fee per tonne of product. Systems include those from ABB (Knabenhans, 2011), FLSmidth and several suppliers of equipment for gas sampling and analysis.

8.2 Failure modes

Maintenance aims to ensure maximum availability and efficiency of equipment using limited resources of manpower, cost and equipment downtime.

'Maintenance' is the preservation of equipment condition, while 'repair' means restoring it to pristine condition; 'patching' is inadequate repair to less than new condition. Historically, cement industry maintenance involved running to failure, followed by repair or replacement. In the 1950s the concept of preventive, or operating-time-related maintenance was developed, attempting to predict equipment life expectancy and initiating 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 equipment items or their components have a high risk of early failure (infant mortality) followed by an extended life with high reliability. Combination of these two observations led to the 'bath tub' concept (see Figure 8.1). More recently it is suggested that there are at least six different characteristic curves.

Figure 8.1 Equipment failure profiles



It became 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, 2000) which recognises that, along with data from non-destructive tests (eg oil cleanliness or sonic probing for micro-cracks), most failures eventually give advance warning through parameters such as temperature or noise and vibration. The optimum maintenance programme should therefore identify and adapt to the failure mode for each equipment item or component. Because much equipment is actually damaged through mis-operation, regular surveillance to detect abnormal conditions early is essential, and this is best performed by operations personnel, appropriately trained and provided with clear visual guidelines for normal operating parameters.

8.3 Computerised Maintenance Management Systems (CMMS)

A single-line cement plant may possess more than 500 motors, together with numerous other items requiring maintenance. Effective monitoring demands computerised data processing and there are now hundreds of proprietary systems in addition to in-house offerings. Common to all systems are:

- an equipment database of descriptive and specification information on all equipment items, including all drawings and specifications obtained at the time of erection or later modifications
- a database for preventive maintenance tasks together with a scheduling function related to operating time or 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.

In addition, it is desirable also to address:

- storeroom inventory management and procurement
- labour and overtime tracking
- safety records.

Essential to the CMMS is an examination of the logic underlying every preventive maintenance task to determine whether it is cost effective and does, in fact, prevent 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 inefficiency and potential inconsistency in multiple data entry.

8.4 Reliability-Centred Maintenance (RCM)

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)
- consequent prevention or control task definition.

It should be carried out by the maintenance team, operators, engineers and supervisors responsible for each major equipment system. FMEA involves the most likely, most expensive and most hazardous modes of failure and, not infrequently, it is recognised that certain failure modes cannot be cost-effectively prevented, either because there is no advance warning or because monitoring is impractical. Such situations are usually remedied by redesign or by changes in operating practice. FMEA also requires the definition of function for each equipment system to avoid such ambiguities as the maintenance department considering that a machine is functioning because it is running, while the production department considers that its manner of operation is defective. Considering historical production achieved, equipment design specifications, standard industry performance and business objectives for the year, management must, therefore, establish for each system the required:

- capacity (eg, production rate) together with feed and product specifications
- reliability in terms of per cent scheduled operating time when the equipment is actually available.

Usually these parameters must be maximised for the kiln but may be relaxed for oversized ancillary equipment. The same priorities should be employed by the maintenance department in allocation of the scarce resources at their disposal (manpower, expense and downtime), and these may vary from time-to-time depending upon equipment and inventory situations.

RCM incorporates both preventive and corrective maintenance activities, along with Predictive Testing & Inspection (PT&I) and other actions, to provide an optimal balance to achieve greater reliability and lower life-cycle costs. It was developed in the aerospace industry and very low failure rates attest to its effectiveness. The concept also underlies the various quality management systems (Ireson and Coombs, 1988). Kirby (2012) describes RCM's history, philosophy, approach and outcomes.

To get the best out of any advanced maintenance philosophy, the outlook of the entire workforce needs to change, so that maintenance comes to involve everyone. Albarkoly and Park (2015) consider the experience of four Libyan cement plants where monthly breakdowns were the norm, due to a general misunderstanding of maintenance and its importance, fostered by apparent disinterest and lack of support from management.

Total Productive Maintenance (TPM) originated in Japan in the 1970s. It is based on "prevention at source", focussing on identifying and eliminating the source of equipment deterioration rather than either letting it fail before repair, or applying preventive/predictive strategies to identify and repair equipment once deterioration has taken hold and necessitated expensive repairs. Instead of enlarging inspection teams, all employees are trained and motivated to be responsible for identifying problems at the earliest possible point in the process so as to minimise rectification costs.

Fraser et al (2011) discuss accounts of "real world" applications of the popular maintenance strategies, noting that they are not necessarily mutually exclusive.

The cement industry has generally accepted that the greatest run factor and maintenance cost efficiencies are achieved by running kilns to failure – subject, of course, to analysis of such online monitoring as is possible (eg, ID fan vibration). Planned maintenance is conventional on all major equipment, with shutdown scheduling being either fixed by a management plan or determined by equipment failure. The specific job list should incorporate feedback on deficiencies from both production and quality departments. Shutdown tasks 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
- inspection of highly-stressed equipment that cannot be inspected during operation should be done at every opportunity (eg, clinker cooler).

Mutter (2013) suggests that the focus should be on reliability rather than maintenance and points out that top-performing producers will, as an absolute minimum, conduct a Root Cause Analysis (RCA) for every kiln stoppage to determine the true cause of failure rather than its symptoms and put corrective measures in place. Without a formal investigation, it is likely that a small group will look at the problem and make judgements, based on no or very few hard facts, thereby reaching the wrong conclusion. RCA is a team-based activity involving those who know the plant's operations and maintenance history as well as those with detailed knowledge of the equipment, working with an experienced facilitator.

8.5 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.

CMMS data are increasingly integrated into enterprise asset management systems that also record stock levels, requisitions and material costs, and also to procurement and quality management software. This enables correctly-informed business-driven decisions to be taken without delay.

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

Equipment downtime is another major maintenance resource which should be used as efficiently as possible. For kilns run to failure, best use must be made of the unavoidable downtime and it is essential to plan in advance for all routine checks and accumulated non-critical repairs (Lowell, 2001[1,2]). This may also involve contingency planning for possible tasks that are not confirmed until access is obtained. Once the required job list has been established and the critical path determined, the work should be prioritised and scheduled to minimise the outage. Ancillary equipment that can be made available for scheduled preventive maintenance can accommodate other considerations. For example, maintenance on mills may await peak power tariff periods, and maintenance on crushers should be carried out when the quarry is not operating. Finally, scheduled maintenance should follow the schedule as closely as possible so that production staff gain confidence that the equipment will be available when needed. Loss of maintenance credibility is a major factor in operators' refusal to shut down for routine attention and a downward spiral into breakdown maintenance.

Inputs to the CMMS come from work orders that should:

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

An important management function is to determine operating strategy with respect to maintenance. Work order coding allows a large number of orders to be classified, 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 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 performance is assessed.

The automatic acquisition, storage and analysis of consistently-measured performance data enables a range of Key Performance Indicators (KPIs) to be calculated and used to justify decisions. These KPIs include Manufacturing Equipment Utilisation (MEU) and Overall Equipment Effectiveness (OEE). OEE is calculated from actual equipment availability, performance, and quality losses relative to maximum expected performance - essentially capturing when an asset is available, the speed at which it operates, and the amount of production failing to meet quality specifications.

Not all of the hours in a working week are available: some must be subtracted for planned maintenance, but after this, more time can be lost to unplanned downtime, and the ratio of actual operating time to planned available time indicates "Availability". Trends in equipment utilisation and unplanned downtime can link these to the reason codes that have been applied to stoppages. Another parameter is "Performance", the ratio of actual weekly production to that expected if equipment operates at its target production rate throughout the week. Trends in these parameters can be called up, corrective actions can be applied rapidly and analysis of historical data allows for root cause analysis of capacity losses and determination of areas for continuous improvement.

OEE gives context among the three variables that make it up. For example, speeding up production at the expense of quality so that overall operational performance is compromised is not truly productive. In addition, more than simple asset availability or production-line efficiency, OEE additionally considers product quality and viability - essentially its reputation in the marketplace. A level of 85 per cent has been suggested as a world-class benchmark (resulting, for example, from 90 per cent availability rate, 95 per cent performance efficiency and a 99 per cent rate of quality), but note that it is a general indicator, not a specific management tool.

8.6 Maintenance organisation

In most other industries, the maintenance department includes both mechanical and electrical functions. Separating these groups and having them both report to plant management is peculiar to, though not universal in, cement making, apparently merely due to tradition. Mechanical maintenance skills primarily involve mechanics (or repairmen) and machinists, while electrical skills involve 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 plant areas to foster increased familiarity with, and 'ownership' of, the equipment, along with championing new maintenance initiatives.

Over the past 25 years there has been much discussion and implementation of self-directed workers. In the cement industry, the most common form has been the work team of 3-5 artisans under a leader who, in turn, reports to a salaried supervisor or manager. This concept can and has worked well, but such teams should never 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.

The 'traditional' structure assumes little cross-skilling with separate production and maintenance reporting lines and co-ordination meetings often at two management levels. Some newer process industry structures have moved towards combined operations management (production/ maintenance) covering day-to-day maintenance with geographically-localised production/ maintenance teams. (A complementary central engineering function concentrates on maintenance standards, strategy, training, planning, major repairs, specialist resources, capital projects, etc).

Operations management, whilst perhaps naturally lending itself to electrical/mechanical/production cross-skilling and a total quality multi-disciplinary team approach, needs to develop from a well-managed and controlled maintenance function whereby skill levels and engineering standards are not prejudiced by the new organisational structure. In practice given excellent communication between maintenance and other functions, traditional and 'operations management' structures can be equally effective. Communication is key, a fact not always appreciated by newly-imported senior managers!

8.7 Role, planning and control

Traditionally, maintenance responds to problems and failures identified by operators. Increasingly, however, with more sophisticated monitoring techniques, the maintenance department detects abnormal equipment conditions as well as correcting them. This may involve a dedicated inspector, though a significant amount of diagnostic data can be electronically collected and analysed.

Analysing the proportion of maintenance time spent on breakdown work is a useful measure of the effectiveness of a preventive programme. In badly-maintained facilities, up to 80 per cent 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 root causes, otherwise the department will be organised for perennial breakdowns, becoming entirely reactive, resulting in either excessive manning levels or excessive overtime, and parts inventories will expand to cover any eventuality. With carrying costs accounting for 10-25 per cent 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 until it impacts earnings. Risk analysis can be effective to establish optimum stock levels, if integrated with analysis of equipment failure and procurement cycles. Considerable savings in inventory cost may be achieved if suppliers agree to carry stock in the plant and charge for it only when used. There may also be potential for selling back surplus stock.

Technical strategy is concerned with selecting the optimum balance between different categories of maintenance, such as the following three, taking account of the prevailing market demand, the balance between direct and 'hidden' costs and the type of failure mode:

- Operate to Failure (OTF)
- Fixed-Time Maintenance (FTM)
- Condition-Based Maintenance (CBM).

All of these can be applied offline, when shutdown opportunities present. But FTM and CBM are preventive in nature and CBM is also corrective.

Managers need to decide which approach to apply to each plant item and, in the case of FTM and CBM, how frequently these techniques should be applied. In many cases, the choice of technique for a given component will be obvious from experience. In less obvious cases, the following considerations can be helpful in deciding upon the most cost-effective approach:

Failure characteristics

- mean life (mean time to failure)
- uncertainty of item life
- detectability of failure (Is inspection possible? What technique is required? What advance warning of failure - lead time - will this give?)
- replacement time
- Can the item be reconditioned/repaired?

Cost characteristics

- consequence of failure (in terms of lost output, quality, associated damage, maintenance resources)
- cost of preventive actions (maintenance resources, lost output).

Safety and environmental implications

- consequence of failure in respect of safety of personnel and environmental impact.

Spares holding policy is inextricably tied to the above strategy formulation process. For example, the OTF approach may require complete plant assemblies to be held in stock (eg, a complete gearbox) whilst FTM or CBM techniques may require only individual spares items (eg, bearings, seals and gears) to be held or ordered on condition ('just in time').

Prioritisation of maintenance tasks is essential to allow timely equipment inspection and repair before failure. Resources will never be considered adequate, so that priorities must be set by the plant manager, maintenance manager and 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. Ancillary equipment can often be allowed to shut down without permanent loss. Redundancy reduces priority
- Equipment failures that impact the ability to load out product and satisfy customers are a priority.
- Safety items must immediately be either corrected or temporarily neutralised (eg, by locking out or roping off).
- Environmental or regulatory items may sometimes be deferred if an action plan is agreed with the regulator.

Some important and essential inspections do not concern equipment with moving parts but relate to the integrity of structures, where failure would impact severely upon staff safety and continuity of production. A prime example is keeping records of inspection of concrete and steel silos at intervals of 2-5 years.

Priorities and their justifications should be communicated to shift supervisors 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 workforce with a varying load of priority work.

8.8 Mobile equipment maintenance

Vehicle maintenance requires specialised skills not normally associated with conventional plant maintenance and is traditionally managed by the quarry department. With increasing equipment size, the number of units is frequently inadequate to allow efficient management of a dedicated team and equipment, so that many plants use external contractors for all but the routine checks and adjustments carried out by operators. Frequently such resources belong to the equipment supplier, who can offer efficiencies of familiarity, technical support, parts inventory and the ability to apply varying levels of resource according to need, now able to use data generated on-board and transmitted for analysis by experts in a distant location. In-house facilities may remain for remote locations or an abnormal fleet composition.

However, wherever maintenance is conducted, a system must be established to track the maintenance and service history of each item of mobile equipment and this may be either the quarry department's record-keeping programme or the maintenance department's CMMS.

Operating hours are logged at each routine inspection so that preventive maintenance activities can be initiated when appropriate. Mobile equipment is subject to an increasing risk of wear related to operating hours, while the condition of machinery can be closely monitored through analysis of lubricating and hydraulic fluids that contact various components. Oil monitoring is 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 nature of failure types to be managed.

8.9 People and indicators

A maintenance supervisor manages all maintenance staff, overseeing all their work as well supervising planning, logistics and administration of the department. The supervisor needs excellent interpersonal and management skills, and an extensive knowledge of the systems, legal requirements, equipment and installations involved in cement manufacture.

Maintenance can be one of the most difficult working environments, providing a service to others, often under considerable pressure. It is a fast-moving, dynamic and stressful environment where people must be able to work well under pressure.

The Society for Maintenance and Reliability Professionals, based in Atlanta, proposes a range of metrics, including the following:

Table 8.1 Maintenance metrics

Measurement	Goal
Work order backlog	Planned: 6 weeks Ready to schedule: 2-3 weeks
Overtime	<5 per cent
Preventative maintenance compliance	>90 per cent
Schedule compliance	>90 per cent (report why schedule was broken)
Planned maintenance ratio	85 per cent
Reactive work	<10 per cent
Mean time between failure	The higher, the better (start on critical assets)
Budgeted vs actual	9.5-10.0 per cent of budget

Source: Society for Maintenance and Reliability Professionals

Jusko (2013) quotes maintenance guru Jess Shivers as listing these 10 maintenance metrics to drive better decisions.

1. Maintenance cost per unit of production – determine the right balance between running equipment to failure and spending too much.
2. Maintenance cost as a percentage of replacement asset value – a world-class percentage is less than two per cent.
3. Preventive maintenance compliance – 100 per cent compliance is the aim, but often only about 60 per cent of tasks are completed, and only around a quarter of those are carried out correctly.
4. Planned versus unplanned maintenance – a world-class level is 90 per cent planned.
5. OEE or availability – aim for 85 per cent OEE and/or 95 per cent availability. (Although neither is strictly maintenance-oriented, both are influenced by maintenance.)
6. First-pass yield or first-pass good (the ratio of finished product emerging from a process stage to the amount entering) – typically considered a quality metric but can be influenced by maintenance.

7. Stores inventory value – a benchmark level is 0.5-0.75 per cent of replacement asset value. Further, inventory turnovers should number 2-3 per year, rotating the inventory to reduce aging.
8. Backlog of work orders in worker-weeks – because companies calculate the measure differently, it is hard to identify an optimal number. If the correct practices are in place and the backlog is high, staffing may be an issue. On the other hand, operations decisions can drive the metric, in which case it could could be a maintenance/operations partnership issue.
9. Schedule compliance or "Did we do what was on the maintenance schedule?": aim for 80-85 per cent.
10. Critical events – not a benchmark metric, but something for a corporate manager to watch. This is about big issues, such as a two-hour downtime on a production line or four hours on a packing line, and should spur a root-cause examination.

9. Environment and pollution control

Environmental regulation is, of course, very much a matter of national and local ordinance. However, most nations and larger corporations aspire to keep ahead of the game, and certain generalisations can be made about air, water, solids and noise pollution. Cement plants are primarily concerned with emissions to air.

The European Union (EU) conveniently produces 'Best Available Techniques Reference Documents' (termed 'BATRef') for various activities, including one that covers 'The Production of Cement, Lime and Magnesium Oxide'. The latest text, resulting from a 2008 consensus between industry specialists, equipment suppliers and regulators, was adopted in 2010, and reissued in a new format (not affecting details for cement production) in 2013. It provides a useful compendium of knowledge, experience and cost data gleaned from over 200 plants across western Europe. Work on a new edition will start soon.

9.1 Dust collection

The principal types of dust collector used in cement plants are bag filters and electrostatic precipitators (ESPs).

Bag filters ('fabric filters') comprise filters of woven fabric or needlefelts, most often cleaned by reverse air flow or by a reverse compressed-air pulse. Modern pulse-jet designs – equipped with silencers – show lower pressure drops than reverse-air systems, improving fan power consumption. Needlefelts are said to better resist pulse-jet cleaning stresses. Modern synthetic fabrics include materials that can operate up to around 280°C. Typical values of filter rating or 'face velocity' are between 0.5-2.0m/min. Separation efficiency can be above 99.9 per cent, and emissions of less than 5mg/Nm^3 are achieved by well-designed and well-maintained units (dry gas, 273K, 10 per cent O_2). Besides dust, the fabric also removes substances adsorbed on particles, such as dioxins and metals, if present, with suggested potential for impregnation with catalysts to enhance performance. Kiln exhaust filters can contain thousands of tubular bags and ensuring a long bag life is an important cost factor.

Successful applications of bags woven with a surface PTFE membrane have been reported and a number of suppliers offer this enhanced medium (eg, Al Matrafi et al, 2008). The backing medium has to be compatible with the mechanical, thermal and chemical requirements, and has an inert and microporous structure upon which dust is collected – in contrast to the 'depth filtration' seen for uncoated textiles.

A potentially useful pending development is a flexible metal fibre filter material, resistant to 1100°C (ZKG 7/2004, p36), and sintered lamella and ceramic fibre media have also been investigated. Rigid elements resembling elongated thimbles are available, made from Ca-Mg-Si fibres, stable up to 850°C, and well suited for dedusting cooler exhaust air. (Lauer, 2017.)

Harfmann (2018) provides a useful table of numerous filter applications in a cement plant and the associated dust loads and temperatures, suggesting suitable polymer materials to achieve below 10mg/Nm^3 in each case and discussing design parameters such as face velocity.

While the risk of explosion with unburned fuel or CO is obvious for electrostatic precipitators (ESPs), a similar risk attaches to bag filters unless anti-static bags are employed and earthed.

ESPs comprise an array of discharge electrodes at 50-100kV negative potential and earthed collecting plates. Pressure drop is about 15-20mmH_2O and power consumption 0.2-0.3kWh/1000m^3. Efficiency is typically over 80 per cent per field so that a four-field unit should capture above 99.9 per cent of dust. Efficiency varies with particle size, and was historically described by the Deutsch equation, derived in 1922 and involving the effective migration velocity ω (m/s), the collecting plate area A (m^2) and the gas flow rate V (m^3/s):

- Efficiency, $\eta = 1 - e^{-\omega A/V}$

However, modifications were produced during the 1960s to better address more stringent limits and problems collecting ash from low-sulphur coals. The Matts-Ohnfeldt equation has worked well for power generation and cement kilns (Parker, 2003).

- $\eta = 1 - e^{-k}$

  where: $k = [\omega_k A/V]^{-n}$

  with ω_k being a 'precipitation constant' for the particles concerned (m/s), and n a constant close to 0.5 (the slope of a graph of effective migration velocity against field length).

Migration velocity reflects dust resistivity, particle size, field intensity, gas viscosity and other design parameters, and for a dry-process kiln is likely to be 0.07-0.10m/s, with dust resistivity between 10^7-$10^{11}\Omega$cm. H_2O, Cl and SO_3 reduce basic dust resistivity. Gas temperature should be 120-150°C, though ESPs can be designed for higher levels. Modern units are very reliable and guaranteed to achieve below 10mg/Nm^3 but are still liable to total electrical shutdown due either to kiln interlock or to electrical failure, and this must be acceptable under emission regulations. Richter et al (2003) review developments in the design and control of ESPs, and Vandelli (2003) describes hybrid dust collectors where a precipitator is followed by a fabric filter.

A large ESP can be as efficient as an equivalent bag filter, with similar capital cost, and lower operating and maintenance costs. However, a perceived sensitivity to CO trips (see below) has driven a preference for bag filters on kiln exhaust duties, due to concern over potential emission of noxious compounds from waste fuels.

Large bag filters and ESPs collect mill product as well as cleaning gases. With extremely-high inlet dust loadings, preliminary 'knock-out' entry sections are provided to limit the particle burden upon the main unit and flow vanes ensure an even distribution of gas and dust across the system. Cyclones may play a role here, being typically 95 per cent efficient, dropping to around 60 per cent for particles below 5µm. Performance is inadequate for current emission standards, but they can still partially clean process gases that are ducted to other equipment, perhaps for drying or partial inertisation duties.

Conditioning or cooling of exhaust gas may be necessary before dust collection. Kiln exhaust is often used for drying raw materials, which cools the gas and raises its humidity. Inlet gas temperature should be below 170°C for ESPs and for most baghouses; for the former, a moisture content of about 15 per cent (v/v) is required, as often encountered after a raw mill. It is important, however, that the gas should not pass through the dewpoint before release, especially if significant SO_2 or Cl is present to cause corrosion. When the raw mill is off, a conditioning tower (CT) cools and moistens kiln exhaust by water sprays – it may also be used for cooler exhaust. The alternative is to add tempering air, but this significantly increases total gas flow and fan power consumption, and would lead to de-rating of the kiln. CT operation is notoriously problematic due to the dirty atmosphere that the sprays encounter and to the large turndown ratio needed for control (Richter and Taylor, 1999). It is important to achieve good atomisation and spray dispersion throughout the gas stream, whilst avoiding a 'wet bottom'. CFD modelling of one tower (Sharifadeh et al, 2007) demonstrated that a swirl device intended to improve droplet distribution (and replacing a perforated plate that had required shaking to dislodge adhering dust), was in fact causing spray impingement on the inner wall, leading to build-up. Cold air bleeds are commonly provided to protect bag filters against severe gas temperature excursions and air-to-air heat exchangers can reduce the temperature of clinker cooler exhaust prior to filtration.

Gravel-bed filters were formerly used on clinker coolers, tolerating temperatures up to 450°C and temporary overloading, but they are inefficient and now rare.

9.2 Pollution control

Water

Water discharge has not been a concern beyond handling normal internal waste and stormwater, (with potential for leaching from stockpiles), and spillage, especially near fuel stocks. The temperature of discharged cooling water may also need control and in some circumstances its pH as well. The environmental concept of water stress is now better understood. Water conservation, water footprinting and water management are a growing concern, with water ranking alongside carbon as a finite resource requiring careful management. We can expect indicators to measure performance in water management, measurement and reporting, and guidance on good practice. In 2013 the WBCSD launched a 'Global Water Tool' for the cement sector (www.wbcsd.org/work- program/sector-projects/water/global-water-tool.aspx). This helps companies to map water use and assess risks by identifying areas of scarcity and integrating data for groundwater, surface water and precipitation at plant locations to identify water stress areas.

Solids

Apart from expected factory debris (eg packaging, pallets, scrap, spillage sweepings, vacuum machine discharges, etc) the main solid waste of concern is usually kiln brick. Now that chromium is rarely a component, used refractory may be incorporated to kiln feed (after crushing and grinding) or landfilled. Note that if the 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 is dust, if a bypass or final ESP field is discharged to relieve a volatile cycle. In the past landfilling involved minimal expense, but increasing regulation encourages methods to reduce dust generation or recycle it. Besides being partly decarbonated, bypass dust contains alkali chlorides and sulphur compounds as well as intermediate products such as C_2S and CA. Utilisation depends upon composition. In Europe the majority goes as a minor additional constituent in cement, following EN 197-1. Other possible uses are in agriculture, roadworks or sewage sludge conditioning or as a raw material for glass production. Remediation technologies involve either leaching or heat treatment, but few are presently economically and environmentally acceptable.

Material spillage collected by fixed or mobile heavy-duty vacuum systems (Haugen, 2006) is returned to the process.

Noise and vibration

Noise originates primarily from mills, fans/compressors and screw/drag-chain conveyors, and workers need ear protection in affected areas. Noise at the property boundary is best considered at the design stage and can be mitigated by enclosure, insulation and by silencers for large fans (Fuchs, 1993). Soundproof doors must remain closed unless access is needed. Enclosed buildings for noisy equipment require ventilation, and the ventilation fans require silencing. Although most kiln exhaust fan problems occur with short stacks, some ducts and tall chimneys resonate unless silenced. Audible alarms on reversing vehicles can produce many complaints from neighbours at night, and appropriate traffic management and layout of roads can eliminate these. Noise sources, regulation and control are discussed by Dupuis (2003 and 2004).

As well as noise, there can be repercussions from quarry blasting (Section A2), and heavy machinery such as VRMs require isolating foundations to avoid vibrations that can escape the plant boundary.

Regulations

Air pollution control is becoming progressively more onerous worldwide as regulators lower limits on particulates, CO, SO_2, NO_x, etc and add new prohibitions on metals, dioxins and other trace chemicals.

Particulates

Dust suppression is important where dry materials are handled or mobile equipment uses unpaved surfaces. Various engineered and chemical-spray systems are reviewed by Carter (1995). Archer (2003) describes fugitive dust control methods for crushers, screens and transfer points. Small bag filters are ubiquitous for such duties, yet frequently little care or attention is lavished upon doors, ductwork, fans and dust handling equipment. This topic is addressed by Winston and Hollinghead (2007), pointing out that it is often not the filter or its media that cause poor performance. Respirable dust level at locations within and at the perimeter of plants is subject to regulation in many countries including the United States (Cecala et al, 2000[1] and 2000[2]).

Particulate limits for kilns are now typically well below 50mg/Nm^3. Where exhaust opacity is used to monitor emissions, detached plumes resulting from hydrocarbons or ammonium, chloride or sulphate compounds combining after emission can present a problem not addressed by conventional dust collection. Considerable understanding of detached plumes has been acquired by afflicted plants and some information has been published (Wilber et al, 2000 and Tate, 2003).

The EU no longer proposes emission limits unless waste fuels are used. Instead, the BATRef volume reports that typical clean gas dust contents (daily average values) attained with ESPs range between <10-30mg/Nm^3, while for fabric filters the range is from <10-20mg/Nm^3. The conclusion is that ESPs, bag filters and hybrid filters are all BAT (Best Applicable Techniques), with associated emission levels of <10-20mg/Nm^3, as the daily average value, noting that "when applying fabric filters or new or upgraded ESPs, the lower level is achieved".

The Directive on Industrial Emissions (2010/75/EU) came into force in 2013 and imposes limits on kilns that incinerate waste. These are mostly consistent with the BAT 'associated emission levels' listed here, and exceptions are noted in the following paragraphs.

The US EPA issued proposals for particulate emission limits in 2010 as part of a broader scheme (Williams, 2010), and after several legal battles, these were implemented from September 2015. The limit for existing installations is 0.04lb/st of clinker or 0.02kg/t. This is equivalent to a concentration of 9mg/Nm^3 assuming a dry-process exhaust gas volume of 2300Nm^3/t, as indicated in the EU BATRef document - though some suggest a 10 per cent lower volume. The figures claim to reflect a desire to limit emissions of heavy metals, other than mercury (see later). The emission limit for new installations is 75 per cent lower. Table 9.1 lists the above emission values, and others that are discussed below.

Table 9.1 EU and USA emission limits

Pollutant	Emissions (dry gas, 10 per cent O_2 at 273K)	
	EU 'BAT-associated emission' [1]	USA limit [2]
Dust	<10-20mg/Nm^3 (lower level expected for new units)	9mg/Nm^3 (2mg/Nm^3 for new plants)
NO_x (expressed as NO_2)	<200-450mg/Nm^3 [3]	325mg/Nm^3
SO_2	<50-500mg/Nm^3 [4]	90mg/Nm^3
TOC	10mg/Nm^3 [5]	24ppmv [6]
PCDD/CPDF	<0.05–0.1ng PCDD/F I-TEQ/Nm^3 [7]	<0.20ng TEQ
Hg	<0.05mg/Nm^3 [8,9]	12µg/Nm^3 [10]
Cd + Tl	<0.05mg/Nm^3 [9]	-
Sum of [As, Sb, Pb, Cr, Co, Cu, Mn, Ni, V]	<0.5mg/Nm^3 [9]	-
Inorganic chloride compounds expressed as HCl	<10mg/Nm^3	3ppmv
Inorganic fluoride compounds expressed as HF	<1mg/Nm^3	-
Ammonia	<30-50 [11]	-

[1] *Daily average*

[2] *Assuming 2300Nm^3/t exhaust gas volume (as in 2013 EU BATRef document)*

[3] *Upper level is 500mg/Nm^3, if the initial level after applying primary techniques is >1000mg/Nm^3*

[4] *50mg/Nm^3 when burning waste (unless demonstrating that emissions do not arise from combustion)*

[5] *When burning waste (unless demonstrating that emissions do not arise from combustion)*

[6] *Total hydrocarbons*

[7] *Over a 6-8h sampling period*

[8] *Values >0.03mg/Nm^3 must be investigated*

[9] *Sampling period of at least 30min*

[10] *5µg/Nm^3 for new kilns*

[11] *If ammonia is injected for SNCR purposes*

Gases

The formation of gaseous pollutants during combustion is outlined in Section A4.7.

Carbon monoxide emerging from the kiln or precalciner furnace arises from incomplete combustion of fuel and may be accompanied by unburnt carbon or hydrocarbons. CO surges usually result from unsteady operation of combustion systems. Control of CO levels is critical for ESPs to ensure that concentrations stay well below the lower explosive limit. If the CO level rises above a certain level, the practice was to trip the electrical system to eliminate explosion risk. This leads to apparently uncontrolled particulate releases, though the ESP body does still abate emissions when de-energised. Dust emissions below 100mg/Nm^3 have been reported from one site. A better set of actions involves reducing the dead time for gas analysis and then initially cutting off fuel supply, tripping the ESP only as a last resort. Section 6.2.6 of the EU BATRef document provides detailed guidance.

CO problems may originate from solid fuel feeders, which must prevent surges resulting from hold-ups or blockages, especially with moist material. Annual frequencies of CO trips are minimised at German cement plants, in one case falling to only 1-6 in 2002-4, with total durations between 1-6min.

As well as forming as a result of oxygen deficiency or poor fuel/air mixing, CO can also arise from the presence of organic carbon in raw materials (Sadowsky et al, 1997). Present at up to 1.4-6g/kg clinker in some places, 99 per cent of this oxidises below 680°C, and 10-20 per cent of the oxidation results in CO emission of up to 1000mg/Nm^3 - sometimes more - irrespective of the excess O_2 level.

Most CO emissions continuously measured from 40 German rotary kilns were in the range 200-2200mg/Nm^3 as the yearly average.

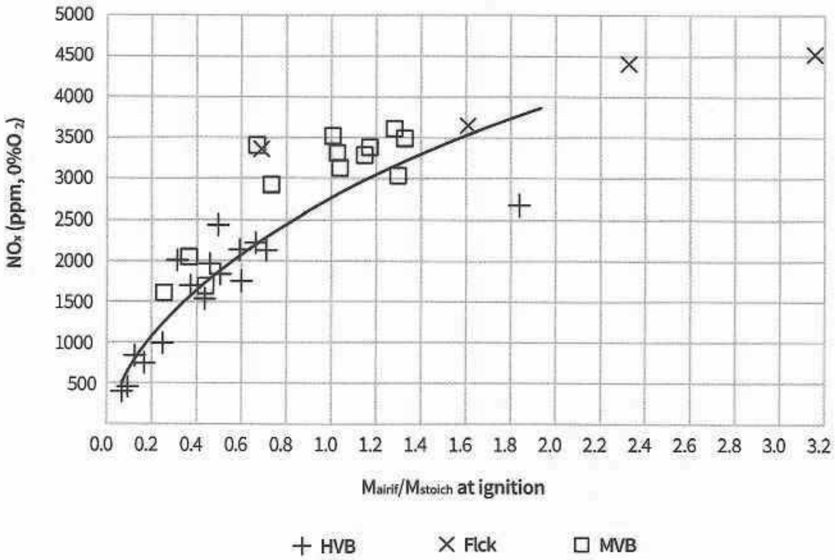
Oxides of nitrogen

Unabated NO_x emissions for normal operation may be 1000-2000mg/Nm^3. Primary techniques for abatement include flame cooling (eg, by water injection), low-NO_x burners, use of mineralisers to reduce clinkering temperature, process optimisation to avoid over-burning, staged combustion in precalciners (and some variants of lump fuel addition to a preheater kiln), while secondary measures are SNCR and SCR. Various techniques can be used simultaneously.

Low-NO_x burners

In a low-NO_x burner, the primary air proportion is reduced to some 6-10 per cent of stoichiometric requirement and axial air is injected at a high velocity, ie, >250m/s. Coal is normally blown through a central channel at a lower velocity, 20-30m/s, while a third channel is used for swirl air, with swirl induced by vanes and/or a bluff body. The net effect is to produce very early ignition - especially of fuel volatiles - in an oxygen deficient atmosphere, tending to reduce NO_x formation. Figure 9.1 shows results for high volatile bituminous coal, flexicoke and medium-volatile bituminous coal produced by the 1992 CEMFLAM research programme, funded by the cement industry and conducted at the International Flame Research Foundation (IFRF). This produced a comprehensive analysis of NO_x formation in cement kiln flames. The effects of burner design and fuel type were extensively tested and documented.

Figure 9.1 Effect of ignition on NO_x formation



M_{airif} = mass of air entrained into flame jet region (at point of ignition)
M_{stoich} = stoichiometric mass of air required for combustion (at point of ignition)

Staged combustion

Fuel can be introduced in a reducing atmosphere in preheaters and precalciners. The strategy is to decompose a portion of the nitrogen oxides generated in the kiln. Typically the vitiated kiln exit gases provide the most effective reducing atmosphere, as shown in Figures 9.2 and 9.3. Further reduction may be achieved by introducing the calcining fuel in direct contact with the raw meal, which acts as a heat sink and minimises local temperatures to produce 'flameless combustion'. Some modern, well optimised plants achieve emission levels below 500mg/Nm³ with multi-staged combustion.

Figure 9.2 KHD Pyroclon calciner

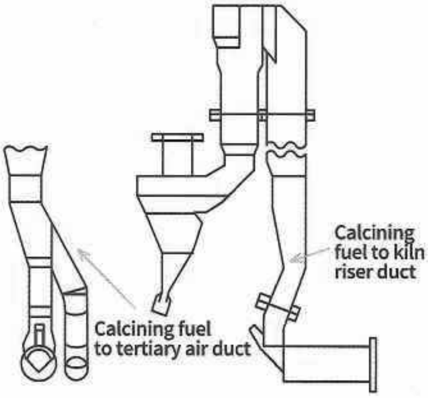
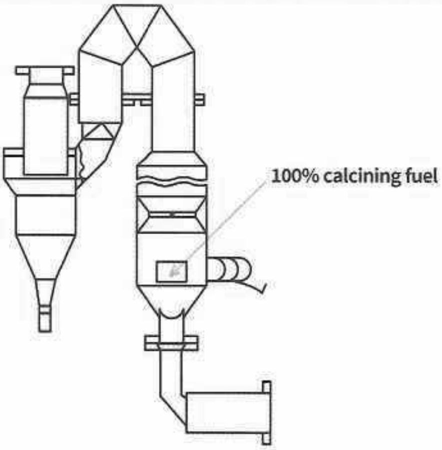


Figure 9.3 FLSmidth ILC calciner



SNCR and SCR

Selective non-catalytic reduction (SNCR) involves injection of ammonia solution, urea or related substances to a preheater in a temperature range of 830-1050 °C. A favourable ammonia stoichiometric distribution and good atomisation are important. Simultaneous use with staged combustion requires that temperatures, residence times and gas atmosphere in the reaction section are adjusted to suit one another.

Selective catalytic reduction (SCR) can be a viable option and has received full-scale demonstrations on cement plants in both 'high-dust' and 'low-dust' configurations, also reducing emissions of VOCs and other traces. Kline (2013) reviews recent developments and Lisberger et al (2013) describe application of a 'semi-dust' system developed by Scheuch GmbH. Schultz (2012) describes developments in FLSmidth's SNCR and SCR techniques involving ammonia lancing and novel catalysts.

Activated carbon filters (installed for other purposes) may also abate NO_x.

EU BAT-associated emission levels are <200-450mg/Nm^3. Existing kiln system design, fuel mix properties and raw material burnability can influence the range achieved. SNCR levels below 350mg/Nm^3 are reached with favourable conditions and in 2008 the lower value of 200mg/Nm^3 had been reported for three plants with easily-combined mixes. The limit in the pending US EPA rule is 1.5lb/st clinker or 0.75kg/t, equivalent to a concentration of 325mg/Nm^3 (see Table 9.1).

Helmreich and Anderl (2017) point out that, whereas the original SNCR units could reduce emissions by around 50 per cent, new installations achieve 75 per cent, principally by careful choice of the urea/NH_3 injection points and set-up of of each lance on the basis of temperature and mixing conditions. They go on to point out that an expected future insistence on the lower emission figure of 200mg/Nm^3 when using waste fuels will almost certainly result in increased ammonia slip from SNCR. This has two main causes: injection at too low a temperature or uneven distribution of the reagent and/or NO_x across the duct. Further measures are listed by Giavazzi and Maiocchi (2018).

They suggest that urea-based systems are often expensive to run and achieve poor efficiency. 'Ammonia water' (a 25 per cent solution in the EU, or 19 per cent in the USA) is 2.0-2.5 times more effective than a 40 per cent urea solution, with less water consumption and fewer problems with calciner build-ups. Delivery, storage and use of anhydrous ammonia is now rarely preferred because it involves the highest safety risk, permitting costs and a number of regulatory issues.

Sulphur dioxide

SO_2 is produced in the kiln both by oxidation of fuel S and by decomposition of sulphates, and is almost totally scrubbed by K_2O, Na_2O and CaO in the preheater, unless present in excess.

However, sulphides and organic sulphur in raw materials oxidise in the preheater and largely exit with exhaust gas, where they can be captured in the raw mill system, if necessary supplemented by scrubbing. Hydrated lime injection at a suitable temperature is effective and alternative sorbents can be used. Capture levels of 60-80 per cent are reported for EU kilns. The presence of moisture aids removal and typically $Ca(OH)_2/SO_2$ molar ratios of between 3-6 are necessary. The raw mill can trap some SO_2, with adjustment to moisture level, temperature, retention time and material fineness. For initial SO_2 levels <1200mg/Nm^3, it is possible to achieve a reduction to 400mg/Nm^3. Higher initial SO_2 levels would involve using significant amounts of lime that might enhance sulphur cycles and cause process upsets. Wet scrubbers can be applied and Schultz (ibid) describes developments in FLSmidth's techniques, such as diverting a small fraction of lime-bearing gas from the calciner to the upper cyclone stage to absorb SO_2, and a gas suspension absorber that can remove 95 per cent SO_2 along with much Hg and PCDD/F. EU and USA emission values are listed in Table 9.1.

Redecam (2018) suggests that a circulating fluidised bed system, proven in other industries, can be an attractive alternative to wet desulphurisation, and also removes HCl, HF, Hg and other metals.

Unfortunately, regulators cannot control natural phenomena such as volcanoes. In 1991 Mount Pinatubo alone is estimated to have emitted an amount of SO_2 equal to between 10-100 times worldwide annual production (Economist, 1992). (Neither are politicians greatly enthusiastic about regulating NO_x and SO_2 from vehicle exhausts and boilers in domestic and commercial premises, given that many voters would directly feel cost impacts!)

Trace elements and compounds

Volatile organic compounds

Of the organic constituents contained in raw materials, well below one per cent is emitted as volatile organic carbon compounds (VOCs). The VOC content of preheater exhaust gas typically lies between 1-60mg/Nm^3, measured as Total Organic Carbon (TOC), though in rare cases it may reach 140mg/Nm^3. (VOC emission due to kiln combustion is very rare, except, perhaps during upset conditions.)

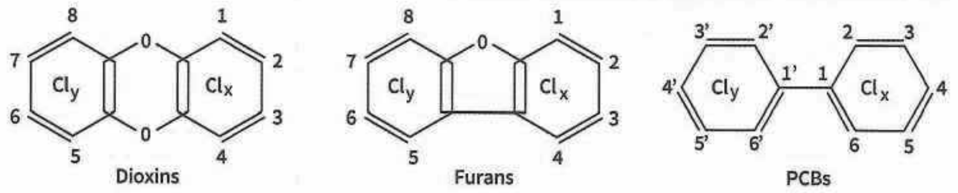
Certain plants have raw materials containing up to three per cent kerogens, or may dispose of oil-contaminated soils. If fed conventionally to the top preheater stage, these tend to distil at intermediate temperatures and exit with the flue gas – if they do not explode in the ESP (Ryzhik, 1992). The resulting pollution is frequently in the form of a detached plume or blue haze. To avoid this and make use of the heat potential, such materials should be injected at above 800°C, usually to a one-stage preheater with a short kiln. The high temperature exhaust may then be used for drying or for power cogeneration (Onissi and Munakata, 1993). If the hydrocarbons occur only in a minor constituent of the raw mix, this may be ground separately and fed to the kiln riser.

Philip (2013) describes successful use of a Regenerative Thermal Oxidizer (RTO) to control odours, CO and organic components. This involves a combustion chamber maintained at above 800°C: the raw meal organic content can be adjusted to supply sufficient thermal energy to maintain this temperature. Ammonia injection provides simultaneous reduction of NO_x. In operation, CO, TOC and odour emissions are reduced by over 95 per cent and NO_x by 50 per cent, with ammonia slip of 10mg/Nm^3.

Dioxins and furans

These are groups of chlorinated organic compounds with similar chemical structures. Emissions of dioxins (polychorinated dibenzo-p-dioxins) and furans (poly-chlorinated dibenzo-furans) are of particular concern to regulators where waste fuels are burned. Any chlorine input to a combustion process in the presence of organic material may potentially result in formation of PCDDs and PCDFs. The basic structures are shown in Figure 9.4, which also includes PCBs (polychlorinated bi-phenyls), some of which exhibit dioxin-like properties or can be oxidised to form furans under certain conditions.

Figure 9.4 Basic structures of dioxins, furans and PCBs



Each can be chlorinated at any of the suitable positions on the aromatic benzene ring

It is generally accepted that PCDD/Fs can result from a combination of formation mechanisms, depending on kiln and process design, combustion conditions, feed characteristics and the type and operation of emission control equipment. In certain conditions PCDD/Fs can be formed in/after the preheater and in the air pollution control device if chlorine and hydrocarbon precursors from the raw materials are available in sufficient quantities. However, formation and subsequent emission require the simultaneous presence of six factors:

- hydrocarbons
- chlorides
- a catalyst (some reports suggest that Cu^{2+} and Fe^{2+} can catalyse)
- an appropriate temperature window between 200-450°C with a maximum at 300-325°C
- a long retention time in the appropriate temperature window
- molecular oxygen present in the gas stream.

There are 75 PCDDs and 135 PCDFs, all with different toxicities. Many regulators recognise a scheme of international 'toxic equivalents' (I TEQ).

It is important that, as gases leave the kiln system, they cool rapidly through the critical temperature range – which is what happens in practice as the incoming raw materials are preheated. Detailed investigations and measurements demonstrate that PCDD/F emissions are low, even when wastes and hazardous wastes are burned:

- Most kilns can meet an emission level of 0.1ng I-TEQ/Nm^3 if conventional primary techniques of material selection and process control are applied.
- Using wastes as fuels or as raw materials at the main burner, kiln inlet or precalciner does not appear to influence or change emissions of persistent organic pollutants.

Heavy metals

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 during the manufacture of cement from natural minerals most metals are retained in clinker.

A trace metal mass balance protocol is described by Gossman and Constans (2003). Using alternative raw materials and fuels requires monitoring both of inputs (Dirken, 2003) and of their distribution between cement, kiln dust (if discharged), and gaseous exhaust (Gossman and Constans, ibid).

Ba, Be, Cr, As, Ni, V, Al, Ti, Ca, Fe, Mn, Cu and Ag are non volatile and their only emissions are at a very low level, with dust.

Sb, Cd, Pb, Se, Zn, K and Na and their compounds are semi-volatile and condense as sulphates or chlorides between 700-900°C, travelling back to the burning zone and building up an internal cycle to the point where an equilibrium is established between input in raw materials and fuel, and output in clinker.

Tl is a metal that is itself volatile and can yield volatile compounds such as TlCl which condense at between 450-550°C in the upper stages of the preheater, where they can accumulate and circulate: partly internally, and partly externally to the raw mill and dedusting plant.

Hg is a another metal that is itself volatile and can yield volatile compounds which pass for the most part through the kiln and preheater, and which are only partly adsorbed by dust, depending upon temperature. To prevent a long-cycle increase in Hg emissions, it may be necessary to limit the concentration of the 'external cycle', by continuously or intermittently extracting part of the dust collected in the filter system.

Besides elemental mercury, reaction products like $HgCl_2$, HgO, HgS, $HgBr_2$ or $HgNO_3$ may be formed.

Kiln dusts contain small amounts of compounds of metals such as arsenic (As), cadmium (Cd), mercury (Hg), lead (Pb), thallium (Tl) and zinc (Zn).

BAT for minimising emissions in the EU involves selecting materials with a low content of relevant metals, especially mercury, using a quality assurance system to guarantee characteristics of wastes, and using effective dust removal techniques. The BATRef associated emission levels are listed in Table 9.1.

The USA approach is different: with new emission standards that came into force from 2015, emissions from existing kilns are limited to less than 27.5kg/Mt of clinker produced, and new kilns have to keep below 11.5kg/Mt of clinker produced (Satterfield, 2013, and ECRA, 2015). The intention is that emissions of other heavy metals will be contained by a stringent limit upon dust emissions. Sorbent adsorption may be necessary to remove Hg at some locations and there are concerns that sorbents could end up in the cement product and interact adversely with some admixtures.

The entry into force of the Minamata Convention in August 2017 obliges member states to reduce Hg emissions, which will require more widespread adoption of the practices used in the EU and USA, perhaps supplemented by additional measures. Knotts and Gueniou (2017) describe the use of a polymer sorbent that has been tested on a cement plant. Satterfield (2017) outlines the the properties of various activated carbons that influence their efficacy in mercury removal. Considered guidance has been produced by the World Business Council for Sustainable Development (WBCSD)(2016).

Hydrochloric and hydrofluoric acids

These emissions are regulated, as are those of 'gaseous inorganic chlorine and fluorine compounds'. Such compounds are, in fact, either not emitted at all or are only found in extremely small quantities in studies of European cement plants. When detected, they are generally attributable to ultrafine fumes of alkali chlorides in the stack gas. Between 90-95 per cent of fluorine entering a kiln system is captured in clinker and the remainder is bound with dust as calcium fluoride. Emissions of gaseous fluorine compounds, and of HF in particular, are virtually precluded. Ultrafine dust fractions that escape the gas filter may simulate low contents of gaseous fluorine compounds in measurements. Regulators, however, still impose limits (see Table 9.1) and demand costly sampling and monitoring.

Ammonia

NH_3 emissions can arise from raw materials and may reach as much as 200mg/Nm^3. Additional NH_3 can arise as 'slip' from the SNCR technique for NO_x abatement, where regular monitoring of NH_3 emissions is essential. Regulation of all the above varies with jurisdiction but has, unfortunately, frequently lost sight of the value of the cement kiln to destroy organic wastes. Also, the legal position has become confused by considering waste burning kilns as incinerators, by considering the products of hazardous waste-burning processes as themselves automatically hazardous, and by assumptions that any measurable toxic metal is dangerous even when at a concentration below its natural occurrence in the earth. Numerical values of emission limits drop as time passes and the list of substances regulated has grown longer, while many regulatory limits have dropped simply because analytical detection thresholds have become lower.

Monitoring

Manual measurement of particulate levels is a protracted procedure involving isokinetic sampling and may be a regulatory requirement. Continuous monitoring of particle concentration uses light diffraction or attenuation across the stack (Stromberg and Puchta, 1996). Gas analysis for process control and emission monitoring involves a wide range of proprietary instruments (Gumprecht et al, 2003). For occasional spot measurements, a small syringe pump with absorption tubes provides a simple, accurate and low-cost method (eg, MSA, Draeger, Sensidyne). Care is needed in avoiding loss of some substances in sample lines or containers, and specialist advice may be needed for studies of trace organic compounds. A range of standards for environmental sampling and analysis has appeared, and is growing. Unfortunately, some regulators require monitoring halfway up the exhaust stack, imposing added complications and expense: such demands should be resisted and pragmatic solutions agreed.

There is an increasing demand for continuous emission monitors (CEMs). A wide variety is available, with many approved by regulators. There is much to be said for cross-duct and in-situ devices, provided that they avoid inter-species interference, as they remove all problems arising from operating and maintaining sampling systems.

The Central Pollution Control Board of India has published comprehensive guidelines for CEMs (2017). The following is a summary account, probably already out of date: not all devices are necessarily proven for cement kiln use.

Table 9.2 Continuous emission monitors

Measuring principle	Type	Gases measured
Chemiluminescence	E	NO, NO_x, NO_2
UV fluorescence	E	SO_2 (H_2S and total reduced sulphur separately)
NDIR (gas filter correlation)	E	CO, CO_2, SO_2, NO_x, N_2O
Fourier transform IR (FTIR)	E/C	CO, CO_2, SO_2, NO_x, NO, NO_2, N_2O, NH_3, HCl, HF, CH_4, H_2O, VOC, O_2
NDUV	E	SO_2, NO, NO_2, NH_3, Cl_2, CS_2 and more
Differential optical absorption spectroscopy	C	CO, CO_2, SO_2, NO, NO_2, HCl, HF, NH_3, VOC, H_2O and more
Flame ionisation detector	E	Total hydrocarbons
Tuneable laser diode	C	CO, CO_2, H_2O, NH_3, HCl, HF, CH_4
ZrO cell	I	O_2
Paramagnetic	E	O_2
Photoacoustic spectroscopy	E	CO, CO_2, NO, NO_2, NH_3, SO_2, HCl, HF, VOC, H_2O

E = extractive   C = cross-duct   I = in-situ

As monitoring of more trace components is required, there is a growing trend to install multi-component analysers. Many systems, especially those based on Fourier transform infrared spectroscopy (FTIR), can measure a range of different components and so offer a degree of 'future-proofing', which is attractive to operators when considering the capital costs of installation.

For Hg, all monitoring devices record 'total Hg' based upon complete reduction of all compounds before analysis by cold vapour photometry. Practical experience with continuous monitors in Europe demonstrates that many additional maintenance steps are needed and that, additionally, it is often necessary to modify commercially-available devices to make them suitable for individual applications. Based upon these experiences, new mercury CEMs have been developed (ECRA, 2011 and 2015). Features key to success include heating surfaces that can contact sample gas, to prevent mercury adsorption, and using sample line materials with extremely weak mercury vapour interaction. Sampling and reduction must occur at temperatures well above the dew point so that no condensate is formed before all mercury is reduced to the elemental state. Avoiding the use of gold collectors for mercury preconcentration gives a rapid and continuous response, alongside eliminating possible problems with the 'poisoning' of gold surfaces.

Various certification schemes are recommended, such as the UK Environment Agency's 'MCERTS' monitoring certification scheme, which is used to approve instruments, people and laboratories. These schemes provide frameworks covering matters such as the standards of performance that the equipment must meet, the relevant level of staff qualification and lists of accrediting laboratories and inspecting sites aligned with relevant standards, such as EN 14791 (for oxides of sulphur), EN 14792 (oxides of nitrogen), EN 15058 (carbon monoxide), ISO 15713 (hydrogen fluoride), EN 14789 (oxygen), EN 14790 (water vapour), EN 16911 (velocity and volumetric flow), EN 14181 (Quality assurance of automated measurement systems) and EN 15267 (Air quality: Certification of automated measuring systems). A range of other relevant standards from various authorities is listed by the WBCSD (2012).

9.3 ISO 14000

The ISO 14000 family of standards address various aspects of environmental management. Some focus on management systems, while others focus on specific aspects such as lifecycle analysis, communication and auditing. Sadly, many systems turn out in practice to be more concerned with providing an electronic or paper trail for auditors, rather than promoting the virtuous spiral of improvement that was envisaged at the outset. However, environmental awareness of staff does increase, which should benefit the planet. The suite of standards comprises:

ISO 14000 family of environmental management standards

- ISO 14001:2015 Environmental management systems - Requirements with guidance for use
- ISO 14004:2016 Environmental management systems - General guidelines on principles, systems and support techniques
- ISO 14005:2010 Environmental management systems - Guidelines for the phased implementation of a system, including the use of environmental performance evaluation
- ISO 14006:2011 Environmental management systems - Guidelines for incorporating ecodesign
- ISO 14015:2010 Environmental management - Environmental assessment of sites and organisations
- ISO 14021:2016, ISO 14024:2016, ISO 14025:2006, ISO 14026:2017, ISO 14027:2017 Environmental labels and declarations
- ISO 14031:2013 provides guidance on how an organisation can evaluate its environmental performance.
- ISO/TS 14033:2012 Environmental management - Quantitative environmental information - Guidelines and examples
- ISO 14034:2016 Environmental management - Environmental technology verification (ETV)
- ISO 14040:2006 Environmental management - Life cycle assessment - Principles and framework
- ISO 14044:2006 Environmental management - Life cycle assessment - Requirements and guidelines
- ISO14045:2012 Ecoefficiency assessment - Principles and requirements
- ISO 14046:2014 Environmental management - Water footprint - Principles, requirements and guidelines
- ISO/TR 14047:2012 Environmental management - Life cycle impact assessment - Examples of application of ISO 14042
- ISO/TS 14048:2002 Environmental management - Life cycle assessment - Data documentation format
- ISO/TR 14049:2012 Environmental management - Life cycle assessment - Examples of application of ISO 14041 to goal and scope definition and inventory analysis

- ISO 14050:2009 Environmental management – Vocabulary
- ISO 14051:2011 Environmental management – Material flow cost accounting – General principles and framework
- ISO 14052:2017 Environmental management – Material flow cost accounting – Guidance for practical implementation in a supply chain
- ISO 14055–1:2017 Environmental management – Guidelines for establishing good practices for combatting land degradation and desertification – Part 1: Good practices framework
- ISO/TR 14062:2002 Environmental management – Integrating environmental aspects into product design and development
- ISO 14063:2006 Environmental communication guidelines and examples, helps companies make the link to stakeholders
- ISO 14064 parts 1, 2 and 3:2006 International greenhouse gas (GHG) accounting and verification standards
- ISO 14065:2013 Complements ISO 14064, specifying requirements to accredit or recognise bodies that undertake GHG validation or verification
- ISO/CD 14066:2011 GHG – Competency requirements for greenhouse gas validators and verifiers document
- ISO/TS 14067:2018 – Carbon footprint of products
- ISO/ATR14069: 2013 GHG – Quantification and reporting of GHG emissions for organizations (Carbon footprint of organisation) – Guidance for the application of ISO 14064–1
- ISO/TS 14071:2014 Environmental management – Life cycle assessment – Critical review processes and reviewer competencies: Additional requirements and guidelines to ISO 14044:2006
- ISO/TS 14072:2014 – Environmental management – Life cycle assessment – Requirements and guidelines for organizational life cycle assessment
- SO/TR 14073:2017 Environmental management – Water footprint – Illustrative examples on how to apply ISO 14046
- ISO Guide 64:2008 Guidance for addressing environmental aspects in product standards. Aimed at standards developers but also useful for designers and manufacturers
- ISO 19011:2011 An auditing standard, equally useful for EMS and quality management system audits
- ISO/IEC TS 17021 (various dates) – Parts 1-7, 9-10 – Conformity assessment – Requirements for bodies providing audit and certification of management systems.

9.4 Sustainable development and climate change

The World Business Council for Sustainable Development (WBCSD) 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.com). In 2002, 10 major cement companies committed to the Cement Sustainability Initiative or 'CSI' (www.wbcsdcement.org) comprising: climate protection, resource efficiency, employee health and safety, emissions reduction, local impacts and internal business practices. They have since been joined by others and external (neutral) consultants have been paid well to create a combined systematic approach to this challenge and gather data on a non-competitive basis in the 'Getting the Numbers Right (GNR)' initiative.

This is a business reality for many plants. For those interested, the concept is well summarised by Kerton (2003), while Portland Bill (2007) and Bätzner (2004) present a more critical view. Rumph (2001) provides examples of materials that have disappeared from everyday life due to past concerns over costly litigation (asbestos) or are being banned (CFCs), and wonders if poorly-based LCA studies may likewise condemn cement. The industry needs to keep legislators and lobbyists correctly and constantly informed to avoid premature industrial death due to hasty and poorly-drafted legislation!

Over the first 10 years, the following significant documents were produced and more have followed, for example, covering management of water and biodiversity. A full list is available at the website, www.wbcsdcement.org/index.php/publications.

- a common reporting protocol for employee safety and safety programmes to educate contractors and drivers
- a CO_2 and Energy Protocol (the "CSI Protocol"), now available in version 3.1, and used worldwide by more than 960 plants
- the world's first global database of CO_2 emissions, reporting on about 1000 facilities worldwide (with a review by Fonta, 2013) and input to the first industry-specific technology roadmap (a path to the future), produced with the IEA (WBCSD-IEA, 2009)
- strict quality guidelines for the co-processing of waste and the use of raw materials
- a guide to emissions monitoring and reporting, alongside a two-year study with UNEP of persistent organic pollutants (POPs), mercury and micro-pollutants
- guidelines on environmental and social impact assessment (ESIA) for quarrying as well as guidelines on quarry rehabilitation
- an assessment of concrete recycling practice.

The concept is broad and in most substantive instances, such as energy efficiency and emissions reduction, simply restates practices which were common pursuits for many years. Particular emphasis is placed upon the driving of climate change by human contributions to greenhouse gases – especially combustion products. Levels of atmospheric CO_2 continue to rise, but some still question how much influence human activities can have, despite the fact that they have reached an unprecedented level.

However, a weight of prominent scientists are behind the concept and it is prudent to take the opinion seriously, given the consensus that the debate is over and that the evidence is overwhelming for eventual disaster from global warming. Some suggest that this is not the case and that much sophistry and misrepresentation is employed to make the case. Lomborg (2001), for example, showed in detail how he believed that this was done. Nevertheless, a remarkable degree of scientific rigour and collaboration has been established and the fifth report on the scientific basis of climate change from the IPCC (International Panel on Climate Change) (2013) provides greater confidence than earlier conclusions, though doubts remain in a few quarters. The next IPCC report is due in 2021.

Meanwhile, the IPCC has issued a special report 'Global Warming of 1.5 ˚C' in October 2018, calling for more stringent and rapid measures to be introduced before 2030, to keep global warming below 1.5 ˚C.

Carbon dioxide and carbon trading

It is reported that cement manufacture results in emission of about 30Gta of CO_2, or five per cent of man-made emissions. Fuel combustion produces around 40 per cent of the CO_2 resulting from cement production, whilst limestone decarbonation accounts for some 50 per cent, the balance being attributed to electricity and transport. Improved energy efficiency and the use of non-fossil fuels help to alleviate the problem, but the impact of the increased acceptance of composite cements can outweigh these, provided that sufficient sources of good-quality extenders are available. Devising more easily-combined kiln feeds and producing products of more consistent quality also contribute to reducing the industry's carbon footprint. It will also be useful to better quantify the CO_2 uptake provided by partial recarbonation of concrete after demolition and the contribution that the correct use of concrete can make to maintaining a reasonable indoor climate for dwellings and offices. Various technologies for CO_2 capture and/or use are being developed and results from pilot installations are awaited. One promising option is that of algae growth in closed reactors illuminated by UV radiation.

Cembureau (2013) has carefully outlined the role of cement in the 2050 low-carbon economy. Their report provides useful analysis of the need to take action on five parallel routes to progress towards an ambitious goal of reducing CO_2 emissions to 20 per cent of their 1990 level.

Carbon trading is one important political/regulatory mechanism that aims to drive down emissions from various industrial sectors. The European Emissions Trading System (ETS), with its current third phase having concluded in 2010, is the best-established version of this. Earlier phases have produced bonus dividends for EU cement as output levels have been depressed and surplus emission allocations have been sold (Kerton, 2011). Phase three offered less generous allocations, but its effects have not been dramatic: there was still a 1.5bnt surplus of CO_2 permits at the end of 2016 (Sandbag, 2017). Cement and lime is the only industrial sector that receives sufficient allowances to meet its needs (probably because allocations are adjusted to alleviate concern over "carbon leakage" due to vulnerability to imports), but in any event, other sectors have sufficient surpluses to see them through to 2020.

The ETS and the likelihood that other regions will follow suit renews interest in energy efficiency as reflected by interest in waste heat power generation, not only for its own sake but due to electricity price hikes. Harder (2013) reported that 865 systems would be in operation by the end of 2013. A total of 98 per cent used the conventional steam Rankine cycle, reaching efficiencies of only around 20-25 per cent compared with the 60 per cent observed for power stations, due to the lower operating temperatures that prevail. Promise is shown for the Kalina (ammonia/water) cycle (Anon, 2012; Mirolli & Gibbons, 2013) and Rankine cycles using organic fluids, which are more suited to the cement plant waste gas temperatures. The main market is for retrofit to existing kilns, as new plants tend to have more preheater stages and lower exhaust temperatures.

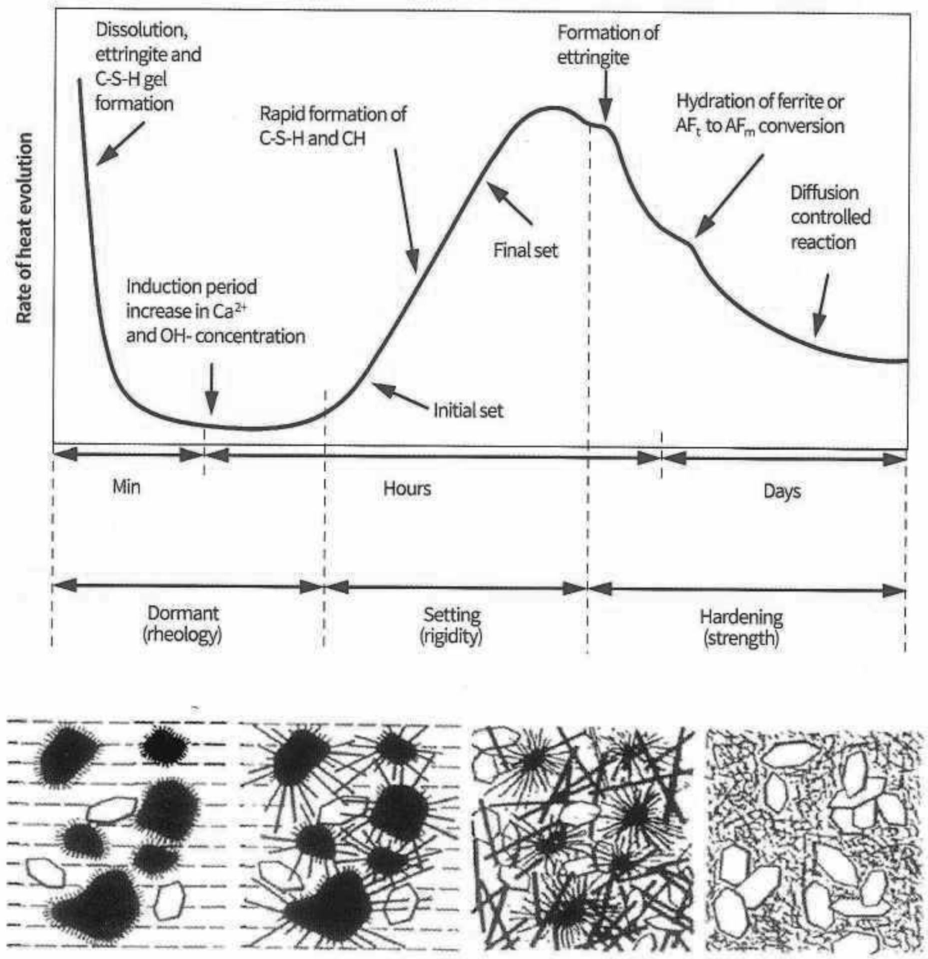
10. Hydration of Portland cement

Portland cement sets and hardens due to exothermic hydration reactions that produce substances including colloidal gels and sub-micron crystals. The crystals may be fibres, needles and plates that form an interlocking microstructure observable with a scanning electron microscope (SEM). Both chemical and physical processes are involved in the standard explanations of cement setting and strength development.

The reactivities of cement phases with water are in the order aluminate > alite > ferrite > belite. C_3A is the most soluble major compound 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.

Hydration can be approximately divided into four stages (described below and illustrated in Figure 10.1).

Figure 10.1 Rate of heat evolution and structure development of Portland cement hydration



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 on the surface of cement particles. Alite dissolves slightly and a calcium silicate hydrate gel coating of uncertain composition (C-S-H) forms on surfaces, blocking further reaction and initiating a dormant period. This initial period is characterised by heat release. A deficiency of sulphate in solution leads to uncontrolled hydration of C_3A to $C_4A.14H_2O$ and $C_2A.8H_2O$ that crystallise to cause flash set. An excess of soluble sulphate leads to precipitation of gypsum, syngenite or ettringite in pore solution, causing false ('plaster') set. An outline of sulphate properties and their effects upon paste and concrete performance is given earlier (Section A7.14).

10.2 Induction or dormant stage

This stage lasts 1-3h while the hydration of all clinker phases progresses slowly. Initially, 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. 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. 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.

10.3 Acceleration stage

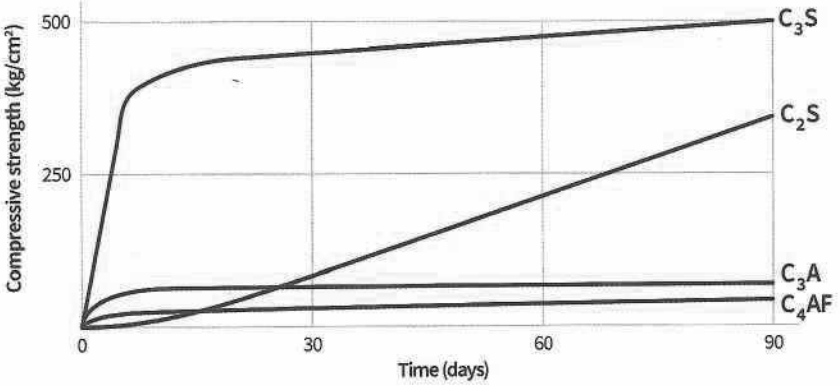
This stage begins with initial set and continues with rapid hydration of alite into C-S-H and $Ca(OH)_2$. These reactions are accompanied by intense heat release which reaches a peak approximately 12h into hydration. As gypsum is depleted, more ettringite is formed, subsequently dissolving and reacting with $Al(OH)^{4-}$ to form monosulpho-aluminate ($C_3A.CaSO_4.12H_2O$). Final set is reached before peak heat release from alite hydration.

10.4 Deceleration stage

This spans many days and is characterised by low-heat evolution and a decreased overall rate of reaction as the species become depleted and diffusion slows with decreasing porosity. Belite becomes the primary hydrating phase.

Hydration of cement typically involves combined water of about 22 per cent 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 causes strength loss. The relative contributions to strength development are shown in Figure 10.2.

Figure 10.2 Strength gain of cement compounds



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. However, when composite cements are used in very dry climates, care may be necessary to avoid concrete drying out prematurely and failing to cure fully.

Other influences

Some minor constituents may play a significant role throughout hydration. If sufficient free lime is present, its expansion during hydration can cause concrete cracking and strength loss (unsoundness). Unsoundness may also result from excessive amounts of certain forms of magnesia (MgO) in the cement. This reacts very slowly, and unsoundness may only appear after many months, or even years. An autoclave soundness test is required in ASTM C150/C150M-18 to detect cements with excessive amounts of CaO or MgO. Hydrated lime ($Ca(OH)_2$) and MgO quenched in a glassy form will not expand.

Alkalis exist in clinker either as oxides dissolved in major compounds or as sulphates, which act as accelerators (increasing early strength gain but also marginally reducing later strengths). They may also affect the performance of admixtures.

Cement fineness has a direct effect on the hydration rate. Finer cements hydrate more quickly, with more rapid heat generation and greater strength gain, particularly during the early period of hydration.

Some concrete products are cured at high temperatures. Reactions are accelerated and reaction products are more crystalline.

Heat of hydration

The rise of temperature in a concrete mass can be considerable during the first few days of hydration. The amount depends upon cement content: hydration rate (determined by cement fineness, ambient temperature and admixtures), cement composition, aggregate volume, thickness of the concrete member and rate of heat loss to the surroundings. (Temperature continues to rise in mass concrete over longer periods.) In cold weather, the heat may help protect concrete from freezing and increase the rate of hydration. Non-clinker components can help to control heat gain during hot weather or in mass concrete.

Drying shrinkage

After hardening, concrete begins to shrink as water not consumed by hydration but needed for good workability and finishing, leaves the system by evaporation – initially from capillaries and later from gel pores. In general, the higher the additional water content, the higher the shrinkage potential. The most influential factor is the type of coarse aggregate used: hard, dense aggregate is able to restrain shrinkage of the cement paste. Other factors having significant impact include the use of shrinkage-promoting admixtures (such as accelerators), the use of dirty aggregate which increases water demand and using a cement with high shrinkage characteristics.

11. Plant reporting

11.1 Definitions

Assessment of equipment and operations involves numerous terms and numbers, many prone to varying definitions. Sometimes a rigid protocol is imposed by regulatory authorities, by a company policy, or by membership of a group of installations that has agreed to share data – and sharing is only productive when common definitions are used.

Plant capacity – Annual capacity can relate to various assumptions for kiln operation and cement intergrinding. A reasonable standard is the designed, or established, daily clinker production assuming 90 per cent annual run factor and five per cent cement additives:

- Annual cement capacity (tpa) = clinker (tpd) x 365 x 0.9 / 0.95

However, the share of additives can be much higher than five per cent. Some elaboration is required for plants which have excess cement milling capacity for producing composite cements, or receive clinker from elsewhere as well as producing their own, or show a deficit due to shipping clinker for remote grinding.

Kiln run-factor – Definitions include fire-on time and running time exclusive of planned shutdowns: feed-on time is suggested as the most significant parameter and should be expressed as a percentage share of 8760 hours per year, ie, 365 days.

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

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

For the above parameters (and others) it is instructive to note the best month's performance and the best week's, as well as the annual average – and then seek explanations as to why the best week is not regularly reproduced. The material in various reports should enable explanations to be advanced and remedies to be proposed: if this is not the case, then the reporting structure and management probably require improvement.

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

Dry tonnes are used for production and raw materials inventory, though reversion to wet tonnes may be necessary in the quarry. Also, certain materials such as coal are usually bought with a standard moisture and adjustment is required to reconcile inventory with purchased quantities and consumption.

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

- Then, for example, if there is 80 per cent limestone in the raw mix and a kiln feed/clinker factor of 1.6:
  1 eqt limestone = 950kg x 1.6 x 0.8 = 1216kg
- Also, if raw mill operation costs US$3.00/t of kiln feed ground:
  unit cost of raw milling = US$3.00 x (950 x 1.6)/1000 = US$4.56/t cement

The system requires adjustment when composite or masonry cements are produced, differing significantly from 95 per cent clinker.

It is also important to bear in mind the various indicators required in TPM schemes to maximise OEE, as discussed in Section 8.5.

11.2 List of reports

Minimal reports to plant management for monitoring operations include:

Daily

- production report – production, downtime, utilisation and inventory by area (milling, burning, etc).

Monthly

- production report
- quarry report – production figures and details of drilling, blasting, loading and hauling
- process summary – operating data and efficiencies for each area
- downtime report – total downtime and detailed breakdown for each area
- shipping report – 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 – raw material, process and product analyses, and statistical variation
- mobile equipment report – availability, fuel consumption and details of major downtime
- manufacturing cost summary – total unit cost and detailed breakdown by area, by individual equipment and by grouping (power, fuel, labour, etc)
- inventory schedule – valuing product, process, fuel and warehouse inventories
- order status – itemising deliveries which have been rescheduled or are overdue
- manpower report – comparing actual numbers with establishment by department, and including overtime, hiring and terminations
- safety report – detailing all accidents and total days worked (to month-end) since last lost-time accident
- projects report – covering planning, ordering, progress and budget of capital projects managed by plant staff.

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

11.3 Inventories and feeders

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

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

11.4 Downtime reporting

One of the most useful and revealing reports of plant operations is the downtime summary. It is believed that suspension preheater and precalciner kilns should be available to operate not less than 90 per cent or 330 days/year (Buzzi, 2003). The dictates of thermal efficiency and of sophisticated process control have introduced large numbers of ancillary equipment items and control signals, failure of any one of which may cause shutdown. 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 only the main reason need be recorded, though other tasks should be noted to explain anomalous totals. For example, if a kiln is shut down for four weeks due to high cement inventory, and major rebricking and maintenance are completed at the same time, the ascription should be voluntary downtime, but the subsequent low annual involuntary downtime must not be incorrectly construed.

Any voluntary kiln 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 feed, or for lack of space for clinker and cement. 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.

11.5 Miscellaneous reporting

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

Operator's data logging is useful even when figures are 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 shutdowns (with detailed explanation), alarm faults, requirements for maintenance or process investigation, process inventory situation, personnel accidents and any other matters requiring 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.

11.6 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	h	tph	kWh/h	kWh/t	kcal/kg	DT (h)	DT reason
Primary crusher				-	-	-		
Crusher-dryer				-	-	-		
Raw mill						-		
Kiln				-	-			
Finish mill						-		

11.7 Typical process summary

Crushing and drying	Month	Year-to-date	Previous year
Primary crusher (tph)			
Crusher – dryer (tph)			
Operating time (%)			
Fuel usage (kcal/kg$_{clinker}$)			
Production – limestone			
– shale			
Feed material moisture (%) – limestone			
– shale			
Product moisture (%) – limestone			
– shale			
Raw milling			
Production (t)			
Operating time (%)			
Production rate (tph)			
Mill (kW)			
(kWh/t)			
Feed moisture (%)			
Meal fineness (% <90μm)			
Meal moisture (%)			
Material usage (%) – limestone			
– shale			
– silica			
– iron ore			
Steel usage (g/t)			
Burning and cooling			
Production (t clinker)			
Operating time (%)			
Production rate (tph)			
Specific heat (net kcal/kg$_{clinker}$)			
Coal (tph)			
Precalciner fuel (%)			
Kiln exhaust gas (°C)			
Preheater exhaust gas (°C)			
Average clinker free-lime (%)			
Feed end (%) – LoI			
– SO_3			
Coal (%) – moisture			
– ash			
NCV – air-dried basis (kcal/kg)			

Finish milling	**Month**	**Year-to-date**	**Previous year**
Production – total (t)			
Type 1			
Type ...			
Operating time (%)			
Production rate (tph)			
Mill (kW)			
total (kWh/t)			
Type 1, etc			
Gypsum usage (%)			
Grinding aid usage (g/t)			
Steel usage (g/t)			
Electrical power summary (kWh/eqt)			
Crushing and drying			
Raw milling			
Blending			
Burning and cooling			
Finish milling			
Packing and shipping			
Utilities and miscellaneous			
Total			
Quality summary			
Kiln feed, Type 1, C_3S			
C_3A			
SR			
Clinker, Type 1, C_3S			
Free CaO			
SO_3			
Cement, Type 1, C_3S (%)			
C_3A (%)			
SO_3 (%)			
Blaine (cm^2/g)			
>45µm (%)			
Initial set (h:min)			
Strength (MPa), 1-day			
3-day			
7-day			
28-day			

11.8 Typical equipment downtime report

Kiln	DT (No)	DT (h)	Share of DT (%)	% h	
			Month	Month	Year
Voluntary - inventory control					
Involuntary - no kiln feed					
Misc operating					
Feed system					
SP plug					
Kiln mechanical					
Power off					
Control system					
Dust collector					
Refractories					
Cooler mech					
Cooler elec					
Fuel system					
ID fan					
Scheduled					
Subtotal					
Total					

Note: kiln downtime is usually counted when feed is off for more than one minute.
Similar tabulations can be made for other major process equipment

11.9 Plant manning

There is a wide range of plant manning depending upon social conditions, education levels, labour cost, plant complexity, availability of contract services, etc. Following is a suggested staffing for a modern, automated, single-line plant that is part of a multi-plant company with central administrative, marketing and engineering services. Assume quarry operates two shifts x five days/week and bagging operates one shift x five days/week. Assume 80 per cent bulk shipping with automated truck load-out.

	Managers	Supervisors and professional	Hourly
Plant manager	1		
Secretary			1
Administration manager	1		
Secretary/receptionist			1
Accountant		1	
Clerical (purchasing, HR, accounting)			3
Mining manager	1		
Foreman		1	
Drill/blast operators			2
Mobile equipment operators			10
Crusher operators			2
Helpers			2
Garage foreman		1	
Mechanics			6
Production manager	1		
Process engineer		1	
Automation engineer		1	
Shift supervisor		4	
Operators			4
Helpers			4
Labourers			12
Packhouse manager	1		
Operators			4
Helpers			6
Maintenance manager	1		
Mechanical engineer		1	
Electrical engineer		1	
Supervisors		4	
Maintenance clerk			1
Warehouse clerks			2
Mechanics			15
Electricians			6
Instrument technicians			4
QC manager	1		
Technicians			7
Environment, health and safety manager	1		
Helpers			2
Subtotal	**8**	**15**	**94**
Total			**117**

12. Accounting

Various audiences use different forms of account to examine aspects of business performance, for example, managers, creditors and investors. Each has no use for parts of the accounts that some others consider essential.

12.1 Cost or management accounting

A detailed cost accounting system is essential to identify and control manufacturing costs and compare actual costs with budget. This should encompass each item of equipment, purchased materials and a breakdown of manufacturing overhead costs. Detailed costs should then be summarised monthly as unit costs (per tonne cement) under various headings, eg:

Table 12.1 Headings for cost summaries

Cost by process area	Cost by natural expense
Quarrying and crushing	Operating salaries
On-site raw materials to raw milling	Operating and service labour
Raw milling and additives	Supplies, rentals, services
Blending	Grinding media
Burning and cooling	Refractories
Finish milling and additives	Purchased raw materials
Bulk handling and load-out	Fuel
Manufacturing overhead	Power
Subtotal	Maintenance, including repair parts and labour
Bag premium	

This allows rapid scrutiny of cost trends and abnormalities for further investigation, if necessary, in detailed accounts.

Financial and management accounting are both required, and serve different functions. Often, rules governing financial accounting distort information, making it unsuitable for management and cost control. Managers need financial data to assess operational performance (examining return on capital employed, margin, asset turnover, debtor days); whereas investors wish to assess profitability (using ratios such as earnings per share and dividend yield to cover); and creditors look for indicators of future stability (such as current ratio, acid test, interest cover). The differences between financial and management accounting can be summarised as follows:

Table 12.2 Comparison of financial and management accounting

	Financial accounting Published statements according to GAAP*	Managerial accounting Information for decision-making, planning and control
Users of information	External users: stockholders, analysts, lenders, union, regulator	Internal users: managers at all levels in the organisation
Time period	Past: historical transactions	Past and future: many future estimates
Formats	Standardised	Customised: for different decision contexts
Key information feature	Auditable: comparable and precise	Useful: improved decisions/control; involves subjectivity
Unit measured	Macro: organisation as a whole	Micro: organisational sub-units such as divisions, products, customers
Regulation	Mostly mandatory: follow GAAP	Mostly optional: benefits must justify preparation cost; not GAAP

**GAAP = Generally Accepted Accounting Principles*

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

Table 12.3 Different processes from revenue to operating profit

Financial accounting	Management accounting
Revenue less manufacturing costs (variable and fixed manufacturing)	Revenue less variable costs (variable manufacturing, variable SG&A*)
= Gross margin less SG&A* (variable and fixed SG&A)	= Contribution margin less fixed costs (fixed manufacturing, fixed SG&A)
= EBITDA	= EBITDA

SG&A = Sales, General & Administrative

Management accounts should help managers make good decisions and the systems used to create them can usually be abandoned once they have served their purpose.

Manufacturing costs, also referred to as cost of goods sold (COGS), represent direct materials, direct labour and plant rent. They can be classified as 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, US$/t calculated from (total overhead cost/quantity produced). When multiple products are being made (eg, Type I, II, III, oilwell cement, etc) overhead allocation becomes more arbitrary, resulting in cross-subsidy of one product by another using political rather than technical considerations.

With computerised 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 a product's cost as the sum of the activities that occur to make it. This is not a new concept but rarely seen in the cement industry due to the relative similarity of 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 (indirect costs associated with maintaining a workforce and equipment), overtime, maintenance and supplies, external labour and services, and other plant overheads such as rent. The inclusion of depreciation in fixed costs is not universal so that care should be exercised in comparing manufacturing costs.

Full absorption cost is 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 and 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 amortisation) is used interchangeably with operating profit.

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

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

  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 effectively fail to apply these equations. One example would be converting a kiln to a lower-cost fuel which derates it (eg gas, high moisture, high sulphur). Assuming the kiln is induced-draught (ID) fan limited, such a step 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 maximising 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 to achieve a higher production rate. Indeed, oxygen enrichment and using 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 strategy used to be based on the argument that while we lost a dollar on every tonne, 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

- improving customer relations, 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 light of previous actual costs as well as 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. These will be due either to change in volume or in cost/price and, whether favourable or adverse, must be explained and corrected if necessary.

Volume effect is the effect of a change in plant production on sales revenues and costs. This is normally defined as:

- net volume effect = volume sales (US\$) - volume cost (US\$)

  where:
  volume sales (US\$) = current year (CY) sales @ prior year (PY) (US\$) - PY sales
  volume change (%) = CY sales @ PY (US\$)/PY sales
  volume cost (US\$) = volume change (%) x PY cost

Price effect is the effect of output prices on total sales revenue, defined as:

- price effect = CY sales - CY sales @ PY (US$)

Cost inflation is the effect of changes in input prices on total cost, defined as:

- cost inflation = CY Cost - CY cost @ PY (US$)

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:

- CY operating profit = PY 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 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 previously. 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 cannot.

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 that 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. Databases are available for identifying duplications.

While automatic order generation is desirable, any items costing more than some predetermined minimum amount (say, US$1200) 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 US$2-3/t of annual clinker production.

Major spare parts should be kept under review in liaison with the maintenance team. 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 girth gears and fan impeller shafts require judgement to balance their high inventory cost against potential production loss. If possible, equipment should be standardised so that one spare can be borrowed from a less critical unit (eg, finish mill to raw mill). A record should be kept of which gears have been turned and how many hours they have operated. 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, US$60,000) that 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 US$2-4/t 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, US$2.5m) are conventionally considered separately.

A **long-term plan**, usually for five 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 needed for estimating future benefits and, therefore, the IRR (see below) of an investment. Plans 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 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.

12.2 Investment justification

Plant investment may be justified by:

- safety
- environmental regulation or permit compliance
- payback
- replacement.

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 (discussed below).

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 ('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 two or three years.

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

Present value reflects the notion that a dollar today is worth more than a dollar in the future. It takes into account reduced future purchasing power 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 recognises 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.

- $PV(CF_1) = \sum_{t=1}^{T} \frac{CF_1}{(1+k)}$

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

- $NPV = -I_0 + \sum_{t=1}^{T} \frac{CF_1}{(1+k)^t}$

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, eg, debt and equity, and tax rate. Firms in the cement industry typically carry a lot of debt, mostly rated just above basic investment grade. Whilst large multinational groups can achieve WACC levels between 8-10 per cent.

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

The **internal rate of return** (IRR) is the discount rate at which the NPV is zero. 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):

Table 12.4 Sample calculation IRR and payback

Capital investment		US$4,000,000
Avoided cost or benefit/year		US$1,000,000
Operating cost	(US$200,000)	
Depreciation (say 10 years)	(US$400,000)	
Pretax savings		US$400,000
Less tax (say 35%)	(US$140,000)	
Add back depreciation		US$400,000
After-tax cash flow		US$660,000
Then payback = US$4,000,000/US$660,000 IRR		= 6.1 years = 10.3%

Some projects are essentially discretionary (paving, office refurbishment, etc) where benefits are virtually impossible to cost justify. Another class of projects (online analysers, blending upgrades, etc) involves 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 one per cent capacity increase or a two per cent 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 and Pindyck, 1994). One of the most useful applications of this approach is the decision to invest in a single pyroprocessing 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 the results 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 recognise and value real options implicit in corporate investments will make executives unnecessarily timid in the face of high-risk, high-return opportunities (Anon, 1994). Conversely, excessive valuation of real options can lead to acceptance of bad projects.

12.3 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 [Section A5.3], the use of calcined raw materials [Section 2.1.] and using activated pozzolanic extenders to effectively reduce the clinker factor and hence, increase plant capacity).

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, downtime 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.

12.4 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:

- [cost 1/cost 2] = [size 1/size 2]^R

An R-value of 0.6 – 'the six-tenths rule' – may be used for first approximations, though comprehensive lists of values for various types of equipment and tasks can be found online. 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 to 10 per cent of equipment cost may be assumed. Another useful reference source are the regularly updated series of Price Books, International Price Books and Estimating Costs Guides from Spon Press (London), which give unit and comparative costs for various engineering disciplines, world regions and countries. Time indices are also useful for updating older cost data: these include the Marshall & Swift Equipment Cost Indexes and the Engineering News Record and the Chemical Engineering Plant Cost Index, available in various formats.

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 turnkey price, there will be scope changes and additions which demand contingency funding. Hollmann (2012) provides a review of five decades of experience in cost estimating and suggests that unreliable methods are still used, despite the evidence against them.

Chinese cement equipment suppliers now provide more equipment than those from any other region, with similar standards of quality, reliability and performance. However, a question which faced strategists planning to set up a new production line at the start of the century was whether or not to purchase Chinese equipment. Several suppliers were offering capital costs about one-third less than those from established European or Japanese suppliers. After the first few projects had been completed, some said that significant foreign resources had to be invested in Quality Assurance monitoring and inspection (especially of forged and cast components) and in overall project management (with supervision and inspection of all aspects of installation) to ensure that:

- Equipment could exceed its nominal rated capacity by 15-20 per cent, as customarily expected from established suppliers.
- Full output could be rapidly attained accompanied by low maintenance costs and good plant availability.
- Designed fuel and power efficiency were achieved.
- There were no delays in furnishing key spare parts.

An element of risk was evident, which could be accepted when there was no immediate market demand for plant output, or when established suppliers had full order books and were unable to promise timely delivery and/or demanded too high a price. Calculation of the IRR for a range of scenarios helped to decide the circumstances when the risk (if genuine) was worth taking, or if it was worth putting extra effort up-front into project supervision and quality control.

12.5 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 shareholders' equity (retained earnings) is often provided.

Balance sheet

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

Table 12.5 Example of balance sheet

Assets – current	Liabilities – current
Cash	Accounts payable
Accounts receivable	Wages payable
Inventories	Taxes payable
Other current assets	Other current liabilities
Assets – non-current	Liabilities – non-current
Land & buildings	Deferred taxes
Plant, property and equipment	Long-term debt
Intangible assets	
Other non-current assets	
	Liabilities – owners' equity
	Capital (common) stock
	Retained earnings

Assets (resources of the company)	= Liabilities (external claims against the company)	+ Owners' equity (shareholders' claims against the company)

With some variation in terminology, income statements – or profit or earnings statements – summarise the situation over a period.

Income statement

Table 12.6 Example of income statement

Sales revenues less: cost of goods sold
Gross profit less: selling, general and administrative expenses research and development, and engineering expenses
EBITDA less: depreciation/depletion/amortisation 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

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, investing and financing.

Table 12.7 Cash flow categories

Operating 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 – involving 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 – 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 must 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 scrutinise 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 that 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 is not the result of a sudden occurrence. More often, it is a sudden recognition of a long-standing 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 expenditure 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, discussed above.

Accounting ratios

These are derived from company financial statements by analysts wishing to make comparisons between:

- companies
- industries
- different time periods for one company
- a single company and its industry average.

Ratios generally are not useful unless they are benchmarked against something else, like past performance or another company in the same industry.

Operating ratios

- *Profitability*

Return on capital employed – Net profit before interest/capital employed
(shows ability to convert invested funds into profit)

Net profit margin – 100 x Net profit before interest/sales
(shows % of profit generated from each US$ invested)

Asset turnover – Sales/net assets
(shows how many times annually a sale has been made from each US$ invested)

- *Asset turnover*

Stock turnover (days) – Stock x 365/net assets
(indicates how long it takes to convert stock into a sale)

Debtor turnover (days) – Debtors x 365/sales
(indicates how many days total credit is allowed to customers)

Creditors turnover (days) – Creditors x 365/cost of sales
(indicates how many days total credit is being taken from suppliers)

Fixed asset turnover – Sales/fixed assets
(shows proportion by which a year's sales are greater or less than the total fixed asset base)

Financial ratios

- *Gearing*

Gearing – 100 x debt/capital employed
(shows the ratio of debt to equity, a measure of leverage or risk)

- *Liquidity*

Current ratio – Current assets/current liabilities
(shows by how much the creditors are secured in the short term)

Acid test – Current assets less stock/current liabilities
(shows by how much the company could match immediate creditor claims)

13. Technical and process audits

The **technical audit** is a review of all items of equipment to determine operating capacities and efficiencies with relation to the original design and 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 propose means of eliminating them (McLeod, 1996, and Zelenovskiy and Piper, 2012).

'De-bottlenecking' is a cost-effective technique for increasing productivity.

The **process audit** is an analysis of operations aiming to increase production, reduce unit cost and improve product quality by establishing and controlling optimum mix parameters. This programme must be implemented on a comprehensive basis rather than by troubleshooting 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 mortar).

A first set of savings can be expected, arising from sustained improvements in profits achieved over a short period (12-18 months), with insignificant capital expenditure. These are attained by adjusting operational protocols and improving maintenance. On the back of these, a second set of prioritised investments can be identified, each with payback within 2-5 years.

13.1 Historical performance

Operational and stoppage figures for the past two or more years are collected and reasons for stoppages are analysed in terms of category (mechanical/electrical/instrumentation/refractory/ other), duration and frequency to identify origins in order of severity. The performance of each department is also studied. Simultaneously achieving the best performance for each department would show a considerably-higher operating level for the whole plant. This level should be achievable with a systematic approach.

13.2 Kiln specific fuel consumption

Specific fuel consumption is arguably the most important operating parameter. Most kilns are limited by ID fan capacity, which restricts the amount of fuel that can be burned and hence the amount of clinker produced. Any reduction in specific fuel consumption will produce the potential to increase the production rate (four per cent reduction in specific fuel will allow about three per cent more production. The discount is due to calcined CO_2 from increased production using some of the gas handling capacity).

A fairly modest average specific fuel consumption for an early four-stage cyclone preheater kiln is 850kcal/kg. Modern large precalciner kilns coupled with high-efficiency coolers are capable of achieving a fuel consumption below 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 temperature some 20°C 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.
- Each additional cyclone stage above four improves energy efficiency by approximately 20kcal/kg.
- Bypasses waste high-grade heat – a 10 per cent gas bypass equates to heat loss of approximately five per cent for a cyclone preheater kiln and three per cent for a precalciner kiln.
- High primary air from a directly-fired coal system foregoes some heat recuperation 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 B5.7, adjusted for the previously-mentioned factors. Significant abnormal heat loss is almost invariably associated with preheater and/or cooler exhaust. As can be seen from the typical heat balance of Section B5.7, 420kcal/kg of a total heat consumption of 850kcal/kg is for chemical reaction and 430kcal/kg is due to heat loss, of which 325kcal/kg or 75 per cent is due to losses through kiln and cooler stacks. The challenge is to reduce these exhaust gas heat losses.

The four most common causes of higher-than-target fuel consumption are:

1. **Low output on the kiln.** Heat losses (/t) increase if the system runs below optimum output.

2. **Poor cooler efficiency** due to poor air distribution, poor maintenance, bad design or some combination

3. **Poor heat transfer** in the tower, leading to increased preheater exit temperatures due to:
   - poor material distribution across one or more stages
   - material dropping down to a lower stage rather than being picked up by the gas stream
   - incomplete combustion in the calciner
   - calciner oxygen level too low.

4. **High number of unplanned stops** which consume additional fuel during re-heat when no clinker is produced.

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

LSF	95-97 per cent
S/R	2.4-2.6
A/F	1.5-1.8
Liquid (1450°C)	23-26 per cent.

Laboratory grindability tests should be made for each raw material (Kannewurf, 1956). Laboratory combinability tests can also be helpful (Kirsch, 1994) as, to a lesser extent, can empirical formulae such as those given in Section B5.3. Large unreactive particles should be avoided, if possible, as grinding to reactive size, say <30µm, by simple raw milling will result in excessive power consumption, overgrinding the softer components and increasing cyclone preheater dust loss.

High exhaust heat loss may be mitigated in four key ways:

1. Reduce losses due to CO formation

- Ensure good combustion in the main burner to avoid local reducing conditions in the presence of excess oxygen (Section A4.1).
- Arrange sufficient air-fuel mixing and retention time in the calciner for complete combustion.

2. Reduce preheater exhaust gas quantity

- Maximise heat recuperation from the cooler.
- Minimise excess air without compromising combustion.
- Avoid over-burning with consistent kiln feed chemistry, and constant feed and fuel rates. Note that every one per cent free-lime below optimum wastes up to 15kcal/kg .
- Minimise false air at kiln seals and preheater ports.

3. Reduce preheater exhaust gas temperature

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

4. Reduce cooler exhaust

- Maximise heat recuperation by control of air flow, clinker distribution and clinker granulometry.
- Minimise false air into kiln hood and discharge seal.

The specific fuel efficiencies of 700kcal/kg and below achieved by modern pyrosystems are largely due to their high-efficiency coolers, and inclusion of fifth and sixth preheater stages.

Note that, if kiln exhaust heat is required to dry raw materials, improving heat exchange in the preheater may be counter-productive. However, there is still a benefit in optimising 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, 1987) that the effective heat loss varies with the source of loss due to the grade of heat involved. For a four-stage cyclone preheater with a two-stage grate cooler such variations are shown in Table 13.1.

Table 13.1 Variation of effective heat loss with source

1kcal/kg heat loss from		Compensation required (kcal/kg in additional fuel energy)
Preheater exhaust		0.87
Radiation from	fourth (top) stage	0.22
	third stage	0.44
	second stage	0.76
	first (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.

13.3 Cement mill specific power consumption

A cement mill is usually limited by drive power so that any improvement in mill system energy efficiency (kWh/t) translates intotranslates into increased production capacity as well as a reduced unit power cost. A reasonable standard efficiency for ball mills is 32kWh/t at 3600cm^2/g, which can be adjusted for other finenesses as shown in Section A6.2.

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 mill internals
- inadequate or poorly injected grinding aid.

The need to balance mill and separator was addressed in the Grace 'Umbrella Technique', a valuable tool for optimising mill circuits (Welch, 1984).

13.4 Other systems

Kilns and finish mills are the highest priorities for process optimisation as they are the highest energy consumers, and usually the units limiting plant capacity. However, apart from their potential for energy savings, 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
compressors	utilisation and efficiency of air compressors is improved by lowering system pressure drops, eliminating leakages and using the air only for proper purposes; also, operating times could be optimised, so reducing power consumption
large fans	another major power consumer, with performance linked to material throughput, circulating loads, specific heat consumption and other operational parameters. Harder (2010) usefully reviews over 30 recent articles about a variety of successful interventions.

13.5 De-bottlenecking

This descriptive term is a mandatory focus for 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 so that a comprehensive plan for an optimum capacity increase may be designed and implemented. Handley (2009^1 and 2009^2) argues for a fresh approach, allocating costs to different plant items when prioritising which to optimise, given their sometimes considerable impact upon profitability. Use of criteria such as cost (or profit) per bottleneck-hour can provide useful food for thought and a parallel procedure can be used to look at limitations that remain after de-bottlenecking due to the least profitable groups of products or customers.

Raw material supply

Many wastes can replace traditional raw materials and reduce cost or allow capacity increase. Quartz sand can cause both milling and burnability problems, while 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 synthetic gypsum 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 may be used to reduce fuel cost. Each opportunity must be considered in the light of local supply cost, compatibility with the process and the presence of possible contaminants including alkalis, heavy metals and PCBs. Whole tyres are quite commonly added to the kiln riser. However, their combustion is intermittent and requires either that a high feed-end oxygen level is maintained, thereby derating the kiln, or acceptance of high CO spikes in the preheater exhaust. As discussed earlier, pyrolysis is a technically-attractive development but does increase the capital cost and complexity of tyre burning. Alternatively, a pre-combustion chamber could be added.

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 now limit output (see Section A3.1). More commonly, drying capacity is the limitation. Remedies include 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/s. An increase of 20 per cent is acceptable and can be achieved by installing a larger or more efficient ID fan. However, this will increase pressure drop by 40 per cent. Replacement of preheater cyclone vessels is possible but expensive, but (as discussed earlier) modification of upper stage vortex finders can result in pressure drop reductions of 30 per cent at a modest cost.

Rotary kiln

Kilns are usually oversized and can accommodate specific loads of 1.8-2.6tpd/m^3 for preheater kilns and up to 6tpd/m^3 for precalciner kilns. A common upgrade is to increase speed to about 4rpm. The ID fan will usually need upgrading.

Clinker cooler

The first purpose of the cooler is heat recuperation and the most common upgrades for grate coolers involve replacement of under-grate fans with higher static pressure units that 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 to improve heat recuperation and bed distribution (Section A5.9). It is essential for satisfactory grate cooler performance that clinker is reasonably granular and free of excessive fines.

The second function of the cooler is to cool clinker before discharge. The acceptable 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 (Section A5.9). While the BMH G-cooler is probably the more expensive option, it does avoid the extensive kiln downtime inevitable with a major cooler modification. Moreover, the unit is both efficient and requires minimal maintenance.

Finish milling

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

Coal milling

Roller mills can be upgraded by replacing a static classifier with a dynamic classifier. 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 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, shaft size or 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.

Baghouse

Gas flow can be increased by changing filter material (eg, to membrane-coated fabric), using pleated bags to increase cloth area, or 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. Conveyor replacement gives the opportunity to change from, say, air slide to screw, or from pneumatic conveying to mechanical and so reduce either power or maintenance cost.

13.6 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 completed.

A programme to audit the performance of projects should be considered one or more years after commissioning. 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 internal audit procedure.

However, it must be recognised 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 carried out during the annual outage when the preheater will be cleaned, false air reduced and cooler efficiency recovered, so the production rate usually receives a temporary boost. Therefore, it may well be spurious to claim an increase in production rate from immediately before the outage to immediately after the outage as wholly due to a specific modification.

13.7 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 that 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 that 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 cement type, potentially resulting in costly liability and loss of customers
5. Transgression of regulation – failure to comply with operating permits or emission regulations, potentially resulting in significant fines or even plant closure

6. Safety – failure to maintain safe equipment and practices, leading to 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 – use of equipment that 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 increases stripping, recovery or comminution cost, or increases cost of corrective additives

11. Design provision for natural disasters – probability of earthquake, lightning strike, wind, tidal wave or flooding for which inadequate or out-of-date design standards have been used.

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 downtime may over-run

17. Project technical risk – possibility that committed investment may not yield projected benefits due to inherent system inadequacy or through being the first to adopt an emerging technique.

Market risk

18. Competitor action – opening of new plant or terminal within market area or irresponsible price cutting by existing competitor to gain market share

19. Alternative product – Portland cement is essential for some applications and a long lead time would be required for any alternative to be accepted, but using more steel, timber, plastic or brick can become fashionable, as can asphalt paving.

Intellectual property

20. Proprietary cost advantage – the possibility that a competitor discovers, or gains control of, a patented process or piece of equipment that provides a significant manufacturing cost.

14. Plant assessment list

An assessment provides a snapshot for future reference – a document that provides a basis for identifying operations or equipment relevant to the flow of materials through the plant and which may have energy- or waste-saving potential, or which may present opportunities for throughput to be increased. The list here provides headlines against which text and numbers may be entered and which may well be accompanied by flowcharts. Such a data list is a necessary preamble to an external plant audit or a technical due diligence.

14.1 General

Plant ______________________________ Date ______________

1. Property map
2. Land title
3. Surrounding land use
4. Plant lay-out
5. Plant age
6. Space and scope for expansion
7. Foundation conditions, earthquake zone
8. Regulatory agencies and 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 and consumption
20. Availability of waste materials
21. Cement market and seasonal variation

14.2 Administration and commercial

1. Manufacturing costs by area and natural expense
2. Manufacturing cost by cement type (bulk/bag)
3. Price by cement type (bulk/bag), price trends, price regulation
4. Data processing, information systems
5. Employee numbers (including permanent contract labour)
6. Organisation chart
7. Employee benefits/housing/transport
8. Safety records
9. Purchase procedure, local/import
10. Warehouse value
11. Capital investments approved or planned

14.3 Communication with stakeholders

1. Register of community complaints
2. Register of customer complaints
3. Register of workforce grievances and disciplinary procedures
4. Records of meetings with staff association/trade unions
5. Records of meetings with community representatives
6. Record of media coverage
7. Register of services to the community and outreach initiatives

14.4 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

14.5 Drying

1. Raw materials moisture
2. Drying method

14.6 Raw milling

1. Raw materials storage capacity
2. Mill type and dimensions, rated/absorbed kW
3. Separator type
4. Mill capacity, rated/actual
5. Mill drive and mill circuit power consumption (kWh/t)
6. Feed and product fineness
7. Discharge conveyor

14.7 Blending

1. Silo type, dimensions, capacities
2. Blending ratio
3. Power consumption (kWh/t)
4. Discharge conveying

14.8 Kiln

1. Kiln feed conveying and metering
2. Kiln type and dimensions
3. Kiln capacity, rated/actual
4. Run factor and downtime analysis
5. Kiln seals, tyre and shell condition
6. Kiln gas analysis
7. Kiln brick life and 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 and dimensions, clinker breaker
16. Clinker discharge temperature
17. Coal mill/raw mill/tertiary take-off locations
18. Heat consumption (kcal/kg clinker)
19. Auxiliary drive
20. Preheater discharge temperature, conditioning tower, kiln derating with raw mill down

14.9 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 and capacity
6. Coal fineness

14.10 Clinker

1. Clinker conveying from cooler
2. Clinker types
3. Storage capacity covered/outside

14.11 Finish mill

1. Mill type and dimensions; rated/absorbed kW
2. Mill capacity rated/actual
3. Mill drive and mill circuit power consumption (kWh/t)
4. Separator type
5. Product fineness
6. Gypsum type and purity
7. Grinding aid; water spray
8. Cement discharge temperature
9. Cement conveying

14.12 Cement

1. Cement types
2. Silo capacity

14.13 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

14.14 Packing and distribution

1. Shipment by month
2. Shipment – share bulk vs bagged (%)
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, palletising

14.15 Emission abatement measures (dust, NO_x, SO_2, etc)

1. Types
2. Capacity and condition

14.16 Maintenance

1. PM schedule
2. Predictive maintenance
3. Shop facilities
4. Availability of cranes, machining and motor rewinding
5. Maintenance man hours planned vs breakdown

14.17 Process

1. Heat consumption, total and non-kiln (kcal/kg)
2. Power consumption, total (kWh/t)
3. Process engineering resources and records
4. Equipment redundancy
5. Plant bottlenecks
6. Emission limits, dust
7. Emission limits, NO_x, SO_2, CO, etc
8. Process control systems

14.18 Materials analysis

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

14.19 Plant capacity summary

	Capacity (tph)	Capacity (eqtpa)
1. Crushing		
2. Raw milling		
3. Coal milling		
4. Kiln		
5. Finish milling		
6. Packing		
7. Load-out		
Note: 'eqtpa' refers to 'cement equivalent tonnes' as described in Section A11.1		

14.20 Storage capacity

	Capacity (t)	Capacity (days usage)
1. Storage capacity of limestone available to raw mill		
2. Storage capacity of clay available to raw mill		
3. Storage capacities of other raw materials		
4. Storage capacity of blending/kiln feed silos		
5. Storage capacity of clinker (excluding outside)		
6. Storage capacity of cement		
7. Silo storage capacity maintained for:		
a. limestone crushed and correcting materials		
b. coal crushed		
c. gypsum		
d. slag / fly ash		
e. cements of different types		
8. Types of store involved		

15. Cement plant construction and valuation

15.1 New plant construction

In general, new greenfield cement plants cost in the range of US$140-230/t of annual cement production, depending on scale and location. A decade ago, Indian and Chinese plant construction costs were about one-third lower than for systems from European suppliers, but those established companies have eaten away much of that advantage by having equipment manufactured and fabricated in the Far East.

Some items of cement plant equipment that do not require complex engineering have tended towards becoming commodities and price erosion is expected to continue.

Some brownfield plant expansions can cost US$60-200/t but can cost much more, depending upon the complexity of layout required to fit items into an existing site and the usable excess capacity of existing ancillary equipment and storage. Recent examples include:

- US$85 – Chinese plant in Egypt, 2018
- US$167 – Nigeria, 2015
- US$225E – USA, 2016
- US$140 – Chinese equipment, Tanzania, 2017.

Permitting of heavy industrial facilities has become increasingly difficult and expensive, so that considerable intangible value may be attached to an existing operation, its reserves, permits and market, even if the equipment is antiquated and inefficient.

One other development has been the progressive construction of a plant by stages, which may be spread over a number of years. In this model the developer initially establishes a simple terminal with cement silos, packing and loading facilities, perhaps adding a cement grinding mill in a subsequent phase. In either case the intention is to take a market position with lower financial exposure and risk. This can then be followed at a later date with the full kiln line.

Obviously, profitable markets draw in low-cost imports and unless protected geographically, by tariffs or other regulation, high margins tend to be ephemeral. Also, the lead time for new capacity and the cyclical nature of most markets can enhance both risk and reward for building new plants.

15.2 Project management

Worldwide cement consumption reached 4.14bnt in 2017, reflecting an annual average growth rate of four per cent over the past decade. The rate of growth is expected to drop by about half over the coming years, as utilisation of non-clinker components become more popular and demand in China contracts. The resulting capacity additions will be in the region of 125Mta, worldwide, each year. At a nominal US$150/installed tonne, new plant and expansion activity will amount to around US$18.75bn per year.

It is noted that managers sometimes reach decision-making level without previous direct experience of project management. Likewise, entrepreneurs occasionally consider investment in cement facilities without prior exposure to the industry. Problems can then result from a mutual lack of understanding between such owners and their consulting engineers. This section is offered as an introduction to the subject and is not directed to those who have already gone prematurely grey building cement plants.

Standard capacities of new production lines are presently between 1-3Mta or 3300-10,000tpd of clinker. Smaller production units as low as 0.1Mta are built, but these will have a higher unit cost of construction and operation. Small plants are usually only justified where the market is limited and geographically isolated or for purely political reasons.

A project normally starts with a concept, primarily the entrepreneur's concern, and proceeds to professional design, execution and commissioning. Such functions are usually contracted to specialist engineering consultancies, though the owner may, of course, elect to create and manage the necessary resources 'in-house'. The skills involved are very specialised and the 'in-house' route should only be considered by owners who have, or intend to develop, extensive and continuous project experience. This approach is normally taken by large cement producers who do have permanent project supervision staff. The risks facing the uninitiated are potentially catastrophic.

Many contracts with consulting engineers are based upon the Fédération Internationale des Ingénieurs-Conseils (FIDIC) 'Client-Consultant Model Services Agreement'. It is important to apply other current standard forms of contract during appropriate stages of a project.

Project structure

Each project is quite distinct due to market, plant location and terrain, raw materials, fuels, infrastructure, services and product mix. The selection of technology and equipment depends upon local conditions, climate, material and utility costs, and maintenance capability. Thus, despite the obvious attraction, it has seldom, if ever, proved possible to take the detailed design of one plant and reproduce it elsewhere, although Dangote Group (Nigeria) is attempting large elements of this approach in Africa.

In the cement industry, there are four major project types that can impact capacity:

- greenfield plant
- plant expansion by installing a complete new production line on a brownfield site
- major capacity increase by:
    - addition, or major modification, of equipment such as a preheater and the related auxiliaries such as coal grinding, cooling, etc
    - changing the process from wet or semi-dry to dry.
- minor upgrade in capacity such as replacement of a preheater ID fan, or 'de-bottlenecking' a conveying system or feeder.

This section primarily addresses greenfield plant expansion and some larger capacity increase projects.

Execution of the project may be effected in a spectrum of ways from turnkey to multi-package:

- **Turnkey or design-construct project** – One contractor commits to complete execution of the project and to meeting performance guarantees. The advantages are a fixed budget and schedule, and relatively little involvement by the owner, but a sound contract is essential with detailed scope and clear specifications. Disadvantages are the relatively long time to prepare and analyse turnkey bids, and the price premium demanded by the contractor to cover risk.
- **Two-phase turnkey project** – Two individual contracts are signed with the equipment supplier: the first to expedite equipment delivery and the second to facilitate accurate quantities and designs for civil work execution and erection.
- **Semi-turnkey project** involves a single contract for equipment supply, erection, installation and commissioning but separate contracting for civil work.
- **Guaranteed maximum price (GMP)** involves a single contractor who receives concept, specifications and scope. A GMP is negotiated involving a fixed fee for design work and profit, and reimbursement of site costs up to the GMP. This system can make for fast implementation but usually requires that both sides are experienced and have mutual trust.
- **Multi-package project** – The owner or engineer performs detailed design, procures all equipment and contracting in multiple packages, takes responsibility for interfaces, and co-ordinates the activities and schedules of the multiple parties. Advantages are potentially fast implementation, potential cost savings and the ability to select the best equipment from various suppliers. The main disadvantages are budget uncertainty further into the project and acceptance of more risk.

Obviously there are a number of choices involved in selecting the optimum project structure, depending on the project owner and the in-house resources available.

Generally, most projects are organised according to the schedule of phases outlined in Table 15.1.

Table 15.1 Cement plant construction: project phases

Phase I	Feasibility study which formulates the project and determines its return on investment.
Phase II	Basic engineering which develops the design and economic analysis and leads to the major decision point beyond which significant investment begins.
Phase III	Project execution covers contract negotiations, detailed engineering, site supervision of civil work and plant erection. Essential functions are time and cost control, guarantees and warranties.
Phase IV	Start-up covers the commissioning of plant and its transfer to plant operations.
Phase V	Management of the plant for commercial operation, maintenance and administration. This is not necessary for existing plants.

The first choice facing an investor building a greenfield cement plant before entering the schedule of project phases is a fundamental strategic decision:

- To bring in an established cement producer as a strategic partner. Advantages include access to technical, management and marketing resources, and the validation of a project which comes from participation by an experienced industry investor. The risk is that the entrepreneur loses control and becomes a passive investor. The plant operator has considerable scope for sequestering profits by such devices as transfer pricing, service charges and procurement rebates.
- To choose an independent consulting engineering firm to perform or supervise the phases described above. This allows the owner to keep control of strategy and operation but implies a greater acceptance of risk and cost. Even where assistance with project funding is not required, it may be prudent to present the feasibility study to a bank or funding institution such as the International Finance Corporation or European Bank for Reconstruction and Development, to obtain confirmation of project viability.

The second decision is between the two principal methods of construction or some intermediate configuration:

- **Turnkey** – This usually involves a single equipment supplier and carries more or less a 25 per cent premium to cover the contractor's risks. The engineer's role here is primarily for the feasibility study, basic design and monitoring of execution.
- **Multi-package** – This route allows an optimum combination of equipment, fabrication and contracting, which should result in a more favourable quality/price ratio. Here, the engineer is responsible for all project phases and it is essential that the owner places a high degree of confidence in the consultant's capability and experience.

Next comes the decision of sourcing equipment, which has both cost and political aspects. In recent years, very attractive equipment and construction pricing has been available from certain low-cost countries. When selecting an equipment supplier, consideration must be given to quality control, equipment durability and reliability as well as the legal system, which must be invoked in case of problems. Usually, the liquidated damages for failing guaranteed performance are quite negligible compared with the potential loss of production and the greatest assurance comes from the long-term reputation of reputable equipment suppliers fixing faults rather than walking away from problems. Fabrication in low-cost countries under the control and guarantee of major equipment suppliers is, of course, quite different and is standard practice.

15.3 Cement plant investment costs

An important consideration for an owner is the breakdown of the investment which will vary widely with the particular project circumstances, including location and ground condition. However, typical guideline figures are offered for reference (see Table 15.2). These are relative cost breakdowns for multi-package, greenfield plants of various sizes using Western equipment. The cost of mechanical equipment and spares can be further reduced to the 25-30 per cent range when sourced from low cost suppliers, particularly China. At present, Asian players are reducing investment costs by combining European and Chinese equipment supply. Land purchase and the captive power plants which are frequently required for remote locations, are both omitted.

Table 15.2 Breakdown of plant investment costs

	Plant capacity		
	1Mta	2Mta	3Mta
Construction cost share in total plant investment cost (%)			
Civil work, including structural steel	26	24	22
Mechanical equipment + spare parts	32	35	38
Electrical equipment	7	8	9
Transport	2	2	2
Erection and local fabrication	13	16	19
Access roads, services, auxiliaries	7	5	3
Engineering	5	4	3
Supervision and commissioning	5	4	3
Others – permitting, government fees	3	2	1
Total	100	100	100
Total construction cost (US$m)	**201**	**318**	**380**
To which should be added:			
Feasibility study	2.25	2.25	2.25
Quarry planning, drilling, analysis	2.1	2.1	2.1
Quarry development and equipment	18.9	21.6	24.3
Land	?	?	?
Offices, workshops, laboratory, etc	6	6	6
Total investment cost – excl finance (US$m)	**230**	**350**	**415**
Cost per installed tonne (US$m)	**230**	**175**	**138**
Other costs (US$m)			
Captive power plant	add US$1m/MW		
Finance expenses	add 5% of construction cost[1]		

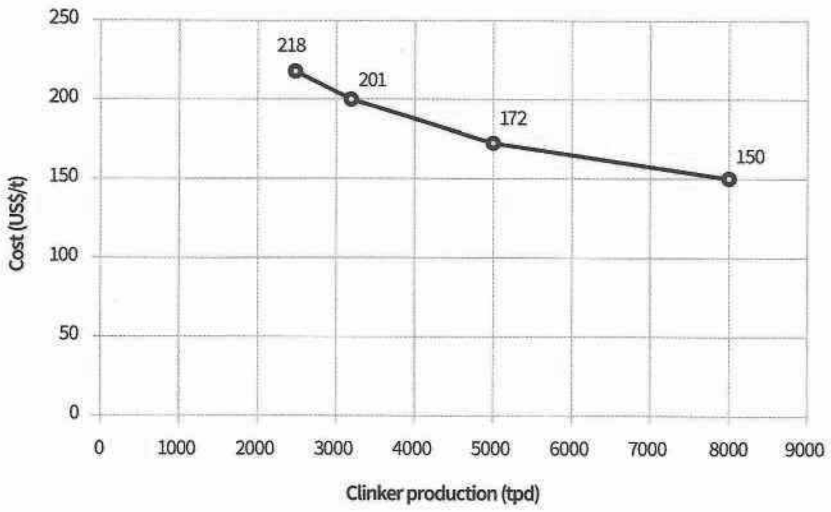
[1] *Note that capitalised interest rates vary all around the world with project capex loan coupons ranging between around three per cent fixed in France to 20 per cent or more in Brazil.*

The following cost breakdown by process was established for a new plant in North America, excluding land, taxes, permits, quarry preparation and trucking (updated from Conroy, 1998). It should be possible to lower these costs further, by between 10-15 per cent (excluding shiping and inland transportation cost) for equipment sourced exclusively from the Far East.

Table 15.3 Cost of a 2500tpd clinker capacity integrated cement plant

Process	Equipment cost (US$m)	Total cost (US$m)	Share of total cost (%)
Quarry equipment	5.1	10.9	5
Raw storage/reclaim	10.2	23.3	11
Raw grinding system	14.5	33.4	15
Blend/kiln feed system	3.6	8.0	4
Coal handling/firing	4.4	9.4	4
Preheater/precalciner/kiln/cooler	23.3	50.9	23
Clinker handling/storage	2.9	11.6	5
Finish mill system	14.5	32.0	15
Cement storage/packing/shipping	4.4	22.5	10
Control room/workshop/office/QC lab	4.4	10.2	5
Infrastructure, roads, water	2.2	5.8	3
Total	**89.4**	**218.0**	**100**

Figure 15.1 The variation of cost with daily clinker capacity – construction including quarry equipment



The variation of cost with daily capacity was also illustrated (see Figure 15.1):

It is important to note that the above curve will not be smooth since, for any given plant, there will be jumps in cost as increasing equipment sizes determine the need to move from single items of equipment (raw mill, preheater strings, cement mills, etc) to multiple units – for example, the move from single- to double-string in a preheater at around 5000-6000tpd. Note that the largest kilns presently available are 13,000tpd (~4.2Mta).

Furthermore, when costs are quoted in US$/t of cement capacity, the applied clinker-to-cement ratio has a very significant impact. The above figures are for an assumed standard 80 per cent clinker content, but this can easily vary between 50-90 per cent clinker, given access to slag, fly ash and other supplementary cementitious materials.

Tables 15.2 and Table 15.3 have been little altered since they appeared in the Fifth Edition of this

Handbook in 2007, and the IEA Greenhouse Gas R&D Programme established a very similar set based on a 1Mta plant in northern Europe in 2008. Total costs have declined, on average, over the past decade as quality-assured steel castings and fabrications have become consistently available in the Far East.

15.4 Project phases

Phase I - Feasibility

The costs above show that a complete feasibility study, including geological investigation of raw materials reserves, will add up to approximately US$4m. Therefore, it is advisable to proceed by an iterative process, starting with a low-cost pre-feasibility study that addresses macro-level demand and price projections, plant location and size, infrastructure, raw material availability from general geological data with limited sampling and analysis, technical plant outline and preliminary economic assessment.

This relatively-inexpensive exercise takes about one month and the project can be abandoned without serious loss should it prove non-viable.

If the pre-feasibility study proves positive, a decision is required to proceed with a complete, detailed, 'bankable' feasibility study that normally includes a quarry drilling programme of perhaps 400 drill samples for chemical analysis. This geological and chemical assessment alone will cost around US$2m.

As the feasibility study proceeds, the project is continuously subject to suspension or termination if major problems appear. It is unlikely, though possible, that insuperable problems will emerge afterwards from market, infrastructure, technical or cost considerations. The principal issues that remain to be settled, and which may produce surprises, are the quantity and quality of limestone reserves, land acquisition, operating permits and environmental impact.

In some countries, obtaining operating permits - particularly, environmental permits - can be fraught and protracted with multiple regulators, agencies and activist groups to be satisfied or placated. The environmental impact study may take two years or more. In such cases, the study and permit applications must be started as early as possible to minimise expensive delays or the risk of proceeding before the outcome is secure.

The major tasks that must be addressed during Phase I are:

Market study

- This includes a review and evaluation of cement market demand, prices, product mix, distribution characteristics, analysis of the competition, trend identification and projections. This may involve consideration of known major projects and contact with individual major consumers.

Environmental study

- Review anticipated environmental factors such as emission regulations, noise, traffic, compensation for neighbours and visual impact.
- Determine applicable regulations and procedures.
- Establish contact with national and local regulators, and estimate possible resistance of neighbours and consequences thereof.

Raw materials study

- Estimate proven and additional probable raw material reserves and availabilities. Normally a minimum of 50 years supply of limestone is a prerequisite.
- Develop mining plan with recommended equipment and personnel requirements.
- Estimate extraction and transport costs.
- Calculate raw mix design and estimate annual consumption of each component.
- Identify and locate required additives and raw materials such as iron ore, silica, gypsum, slag, fly ash, pozzolans, etc.
- Determine if the raw materials will impact plant process or equipment (high moisture, high volatiles, hard grinding, etc).
- Determine qualities of cement that can be produced and suitability for market (low alkali requirement, etc).

Infrastructure study

- Review road access and, if available, rail and water.
- Review electrical power grid proximity, reliability and availability.
- Review water availability.
- Review availability of local accommodation and transport for employees.
- Determine geographical and climatic factors that may impact the investment (eg, seismic, flood, wind/storms, earthquakes, lightning strikes, extreme temperatures, etc).

Plant location

- Evaluate one or more plant locations, considering market and distribution, raw materials, availability of skilled labour and infrastructure, land acquisition, fiscal and legal regulations. (Include a route-to-market and logistics study, with potential depot locations, so that there can be a full network available once production commences.)

Integration to an existing plant for expansion or capacity increase projects

Obviously the same basic considerations apply. In addition:

- Review available excess unit production and storage capacities that can reduce the scope and cost of the project.
- Consider optimum flexibility of the multi-line plant by installing appropriate process cross-over.
- Review potential interference with on-going operations and maintenance by project construction and tie-in.
- Determine necessary downtime of existing operations for construction and tie-in.

Technical study

- Define the process, mechanical and electrical equipment, control concept, basic flow sheet, plant design, equipment specifications, storage capacities, plant utilities, infrastructure, conveying systems, civil work, mechanical equipment erection, electrical installation, plant operation and maintenance, distribution of cement and possible other products, plant lay-out and traffic patterns.

Cost estimates

- Estimate project cost, including cost of finance. Determine duties on imported items, exemptions or subsidies.
- Estimate fixed and variable manufacturing costs.
- Provide a clear indication of estimated precision of data. This should increase as the project progresses from about ±25 per cent for the pre-feasibility study to around five per cent after all contracts have been negotiated and signed. Note that even for a turnkey contract, the almost inevitable change orders will result in some uncertainty.

Financial analysis and return-on-investment

- Based upon estimates of project cost, manufacturing cost, sales volume, sales price, depreciation and taxation, estimate the project pay-back and internal rate of return (IRR).
- Based upon estimates of project cost, project schedule, manufacturing costs, sales projections, distribution costs (if appropriate), depreciation and taxation, prepare accounting statements for the complete project planning period – usually 20 years.

  These statements comprise:

    - balance sheet
    - income statement
    - cash flow.

A sensitivity analysis is also conducted of critical variables.

Project management and organisation protocols

- Establish a preliminary project risk profile by identifying, documenting, and, if possible, quantifying the various risks pertaining to the project (cost or schedule over-run, technical deficiency, etc).
- Outline procedures for project implementation adapted to the method of execution selected (turnkey, multi-package, detailed contracting).
- Outline general time schedule for project execution, management and organisation, parties involved and definition of the division of responsibility.

Preliminary and final report on project feasibility

- Prepare a draft report covering all the above, with a recommendation for further action.
- After discussing draft report with project owner and revising, if necessary, issue Feasibility Study Final Report.

Phase II – Basic engineering and quotation

Assuming a satisfactory outcome from the feasibility study, a decision to proceed to Basic Engineering may be taken. The objective here is to develop the engineering to a level where orders (letters of intent) can be placed with major equipment suppliers. During this stage, project organisation, management procedures and a breakdown of contractual arrangements should be established, together with a definition of procedures for contracting. Further activities include refinement of all engineering designs, detailed definition of process selection and equipment specifications, and compilation of tender documents in the appropriate form and in proportion to the size of the investment.

Raw materials and fuels

- This includes the final assessment of raw materials and final determination of the raw mix, as well as the completion of a quarry plan. These tasks must be completed before the letter(s) of intent for main equipment are signed.

Technical concept

- This includes the continuous refinement of all aspects of engineering, including process-flow, lay-out, general arrangement and section drawings to confirm main equipment and storage capacities and control/automation concepts. Preparation of concise technical specifications including specifications for civil work. Cost effectiveness must always be a primary consideration.
- The basic capacity for each item of equipment can be estimated from its expected running hours and the annual tonnage of material that it will handle. Various margins have to be added to each of these to arrive at design capacities. These matters, which also apply when new facilities are grafted on to an existing production line, are discussed clearly in Deolalkar's 'Handbook for Designing Cement Plants' (2009).

Contracting plan

- Identify contract packages and definition of responsibilities including liability, guarantees and warranties (see definitions below).
- Prepare tender document(s), including one contract proposal in accordance with the contracting plan.
- Pre-select suppliers and contractors.

Project organisation and management

- Establish project organisation with clear definitions of responsibilities and procedures for cost, time and quality control.
- Obtain outstanding permits from local and national authorities.
- Outline project cash flow schedule.
- Outline project risk profile.

Contracting for supply of main equipment

- Issue tender documents.
- Evaluate quotations and preparation of purchase recommendations based on total cost/benefit evaluation, including consequential costs such as civil work, operation and maintenance.
- Prepare draft letter of intent.
- Negotiate contracts.
- Sign letter(s) of intent. To preserve capital, some owners postpone the actual signing (commitment of funds) as long as possible.
- Obtain import licences for the equipment.
- Obtain permits for payment in foreign currencies.

Cost estimates

- Prepare detailed cost estimates, leading to the investment project budget, based on binding commitments by main equipment suppliers. This allows completion of general arrangement and major equipment drawings which, in turn, leads to a more accurate civil work cost estimate. During this process, the quality of data is being continuously refined, as indicated on p252.

Economic analysis

- This stage includes the development of final financial projections before the major decision point.
- These final projections represent the project executive summary and form an integral part of the submission for approval by the owner.

The investment decision taken at the end of this phase – likely to be two years or more after the feasibility study commenced – in favour of project realisation is effectively a final decision. Hereafter, substantial funds must be committed and subsequent delay or cancellation of the project will result in additional costs and heavy losses. Final decisions about financing options may also be taken, such as vendor financing, Export Credit Guarantee Agencies, World Bank/IFC/Development Banks and others.

Diligence during the feasibility and basic engineering phases will generally be rewarded by significant project cost savings, a more efficient schedule, fewer contract problems and better ultimate plant performance.

Table 15.4 Definitions for contracts

Bid bond	a financial guarantee that the contractor will enter a contract if selected. A bid bond is usually 5-20 per cent of the bid amount.
Performance bond	a financial guarantee of contract obligations, representing 100 per cent of the contract amount. In the event of default by a contractor, the guarantee covers the cost of completing outstanding work.
Payment bond	a guarantee that labour and material will be paid for. Note that courts can enforce payment to a supplier even when the owner has already paid the contractor or sub-contractor. Protection is sought by pre-qualifying contractors and by requiring the contractor to provide evidence of payments made to suppliers.
Lien	the right by a supplier or contractor to obtain a claim against the real property on which they have worked in the event of non-payment.
Performance guarantee	a scale of liquidated damages for failure of supplied equipment to meet specifications. Note that such compensation is usually trivial against potential loss and this is a strong argument for only using reputable suppliers with a history of fixing problems rather than paying damages and walking away.
Change order	a documented change from the contracted scope, usually originated by the owner, signed by both parties and quantifying any resulting change to cost and schedule.

Phase III – Project execution

Letters of intent and/or purchase orders mark the beginning of the execution phase. Project execution continues until the completion of all engineering, construction and erection work. The objective, of course, is to complete detailed engineering, contracting, construction and erection within budget and within schedule. Therefore, it is essential to establish an effective project monitoring and control system that facilitates rapid identification of problems and deviations, and allows for early corrective actions. There are a number of good project scheduling tools on the market, but the key is regular review and updating of the schedule, which, on a turnkey cement plant project, can cover over 5000 activities.

An adequate organisation and personnel structure to operate and maintain the plant should be established at an early stage, together with the assurance that suitable materials and utilities will be available for start-up.

Construction time and, therefore, overall investment costs can be reduced by fast tracking construction in parallel to detailed engineering. This means that construction has to start as soon as the first civil drawings become available.

Typical detail contracting activities include:

- securing approval of the investment budget
- securing authority to place orders to purchase the main equipment
- preparation of specifications for auxiliary equipment, evaluation and purchase orders
- preparation of detail drawings for construction of civil works, mechanical erection and electrical installation
- preparation of all documents and schedules to coordinate the above activities, including the project manual
- reviewing and approving all suppliers' and/or contractors' final documents, data, specifications and drawings to ensure that they comply with the contract document specification
- completing the risk profile to provide plant operations with a clear analysis of risks and constraints resulting from cost-benefit decisions made during design and construction – where appropriate, contingency plans must be prepared.

Budget and time control

- Continuous monitoring of budget and schedule using adequate control systems.

Quality control

- Performance of adequate and timely workshop inspections to detect and remedy deficiencies before they give rise to construction or erection problems. Note that 'squeezed' contractors sometimes take shortcuts.
- Constant supervision of the contractors work on site.
- Final check of equipment, installations and civil work. A final deficiency list should be established and communicated. Deficiencies should be classed according to priority, to be rectified immediately or prior to commissioning, start up or final acceptance.

Responsibilities of project management and project organisation

- The coordination of all activities of all parties involved.
- Initiation of corrective measures when deviations occur.
- Preparation and communication of regular progress reports. Certification of progress payments.

Plant organisation

- The organisation structure for plant operation must be ready for implementation before commissioning.
- Operation and maintenance procedures should be developed and documented. Equipment supplier materials will form the basis of this compendium. Documentation should be corrected, if necessary, following commissioning.
- Training of operations managers, operators and maintenance staff should be carried out prior to start-up. This should cover the specific plant being

constructed and may also involve more general training depending upon previous experience.

- Ensure availability of all raw materials, additives, fuels, consumables and spare parts as required for start-up.
- Marketing and distribution systems must be established.
- Ensure that all required permits and proof of insurance are available and passed to plant management.

Phase IV – Start-up and commissioning

This is technically and contractually complex, and cannot be covered here in any detail.

The purpose is to ensure that all equipment and processes function correctly and continuously under load, and that contractors and suppliers meet all contractual obligations.

Commissioning involves no-load testing, start-up, optimising and maximising performance, and confirming that all guarantees are met before acceptance by the owner.

Commissioning includes specialist engineers from all major equipment suppliers and contractors but, in a turnkey project, overall responsibility is with the turnkey contractor. However, it is essential that the owner or his representative must confirm that the process and results are satisfactory.

For quality control, it will be necessary to:

- prepare a start-up and commissioning co-ordination programme
- supervise all tests necessary to accept the equipment
- carry out a final review of all equipment, installations and civil work to prepare a deficiency list, monitor correction, and issue acceptance certificates.

Project management and organisation

- Co-ordinate activities of all parties involved.
- Initiate corrective measures when deficiencies or deviations are detected.
- Issue required take-over certificate(s) and determine penalties or remedies for unfulfilled performance guarantees.
- Prepare and communicate periodic project progress reports.
- Perform a final project audit before the project team leaves site, based on continuous monitoring and feedback during all phases of project implementation, and aiming primarily to identify systemic problems and lessons learned.
- Ensure that complete 'as-built' documentation is transferred to plant operations.

Plant operation

It is, of course, necessary to have an operating staff and organisation to take over when construction is complete. This is more easily established by a company that has existing operations. For a new operator, the choice is between hiring an experienced management team to set up and train staff, or to contract an equipment supplier or cement operating company to provide operations management, either with or without equity participation.

A final phase, not always undertaken, is a project audit a year or so after commissioning. The tasks include:

- Audit the concept and economic projections in the light of actual plant performance.
- Review with operations management the risk profile, performance deficiencies and the effectiveness of contingency plans.
- Analyse performance and, where necessary, propose cost-effective modifications to reduce risk and increase reliability or efficiency.
- Review design deficiencies, especially the conscious deferral of certain equipment or storage in the initial project and recommend new investment, if necessary.
- Review operating and maintenance procedures with plant management and modify, if appropriate.

- Analyse plant operating experience against design expectations and record and communicate 'lessons learned' for incorporation in future projects.

Project schedule

A four-year schedule is typical for a greenfield project assuming no delay for land acquisition or permitting. There is certainly scope for acceleration, but frequently this involves additional cost or risk.

Table 15.5 Typical project schedule

Stages	Duration sub-stages (months)	Duration stage (months)
Inception		0-1
Feasibility study		1-3
Tendering and contracting		5-14
Specification	5-7	
Tender document preparation	6-8	
Call for tender and evaluation	9-12	
Negotiation and award	12-14	
Detailed design		15-26
Basic engineering	15-21	
Detail equipment engineering	18-25	
Detail civil design	18-26	
Execution		19-39
Equipment fabrication	19-34	
Steel structure fabrication	21-34	
Civil work (site)	21-36	
Equipment delivery	23-34	
Erection	24-39	
Commissioning		43-48
No-load testing	43-45	
Commissioning	44-46	
Acceptance	47-48	

Disbursement schedule

Typical payment schedule incorporated in the contract(s) is shown in Table 15.6:

Table 15.6 Typical payment schedule

Down-payment	10 per cent of total contract value against performance bond
Import parts and services	90 per cent of total C&F price as irrevocable, transferable and divisible Letter of Credit (LC) to be paid: • 80 per cent pro-rata against presentation of shipping documents and inspection certificate • 10 per cent on taking over
Local and site work	90 per cent of total local price as irrevocable, transferable and divisible LC to be paid pro-rata against certified invoices

Potential for project failure

Many projects are completed on budget and on schedule, though one tends to hear more about those that over-run, particularly grandiose public projects.

There are a number of reasons why projects produce disappointing results:

- unrealistic cost and benefit projections
- setting budget before sufficient engineering has been performed and, when the budget is shown to be inadequate, reducing the scope rather than raising the budget, but without re-assessing benefits
- beating down engineering, contracting and equipment prices to a level where only desperate or unqualified bidders will respond, thereby increasing the risk of poor performance or default
- attempting to fast-track by contracting on a cost-plus basis before engineering has been completed
- lack of diligence in prequalifying of bidders
- inadequate analysis of competing bids to correlate cost with scope
- excessive change orders issued after contract initiation resulting in unplanned cost and schedule increase
- starving the final phases of cash overspent on earlier phases.

In recent years the concept of 'partnering' has been developed to try to replace the traditional adversarial relationships of owner, engineer and contractor, with a strategy of close communication and co-operation supported by incentives. Essential to this strategy is a commitment to preventing disputes or resolving them before escalation.

15.5 Plant valuation

There are three general approaches to cement plant/company valuation: cost, income and market comparison. However, it must also be recognised that existing competitors make many acquisitions, primarily driven by strategic or market considerations that involve different motivations and models of much greater complexity. In the case of cement, all of the approaches below can be affected by available raw material reserves. If, for example, only 10 years of limestone reserves are available and there is no viable extension of life available, the valuation of any plant will obviously be affected.

The **cost or financial approach** uses replacement cost to indicate 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. For typical depreciation values, see Table 15.7

Table 15.7 Cement plant depreciation: useful lives

Mobile equipment	3-10 years
Machinery and equipment	10-25 years
Buildings	20-40 years

For a single plant, a variation of the approach in Table 15.7 is to take estimated present replacement value, depreciated as outlined above, and apply a factor depending upon plant condition, perhaps ±10 per cent.

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 equivalent and accounts receivable)
  FA = fixed assets (original cost less depreciation and depletion)
  OA = other assets (non-producing assets and 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

The 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 (EV) is a refinement which also takes into account long-term debt and net cash in hand:

- EV = market capitalisation + long-term debt - net cash and investments

(A widely-used, rough and ready valuation technique for an initial estimate of the EV of cement plants and companies is to take the EBITDA – either historically or forecast – and multiply by a factor of 7 or 8 and even occasionally 9 to 10.)

Net earnings (after interest, amortisation, 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 reflecting 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 average P/E ratios include:

■ Mature markets (France, Europe, US)	13.8
■ Emerging markets	13.3
of which: India	15.3
China	13.3

Source: Crédit Mutuel CIC Securities

The premium is typically 30-50 per cent.

Net present value (NPV) attempts to estimate revenues during the life of the investment, based upon discounted cash flows (DCF) and residual salvage value (RV) of the asset. Typically, a period of 20 years is considered. The weaknesses of the method are the estimation of discount rate (k), which is the assumed rate for discounting future incomes (see Section 12.2), and the uncertainty of projecting volumes and prices into the future. In addition, it is possible to vary the valuation by using 'As Is' or 'Improved' performance projections for the equipment, the latter case reflecting the capability of the acquirer to bring added value to the assets.

- Valuation = NPV [k (CFyr1, CFyr2¼.CFyrX + RV)]

Return on invested capital (ROIC) is defined as the cash rate of return on capital invested in operations:

- ROIC = net operating profit after taxes (NOPAT) / invested capital
  where: NOPAT = reported net income + amortisation and depreciation + non-recurring costs + interest expense + tax paid - investment income

  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, and the spread (ROIC - WACC) is a prime indicator of performance.

- Valuation = NPV of (ROIC - WACC)

Calculation of ROIC is complex in practice and remains subject to manipulation by management. It is also influenced by accounting changes, and affected by inflation and currency exchange movements. As with NPV, the final ROIC can be modified by including projected improvements to equipment performance.

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 that 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 southeast Asia suffered from this exposure to an extreme degree after 1997.

The 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 US$/t of annual production.

Table 15.8 offers a selection of multiples resulting from major historical transactions to illustrate the wide range seen in the industry and the absence of a clear trend over the time period.

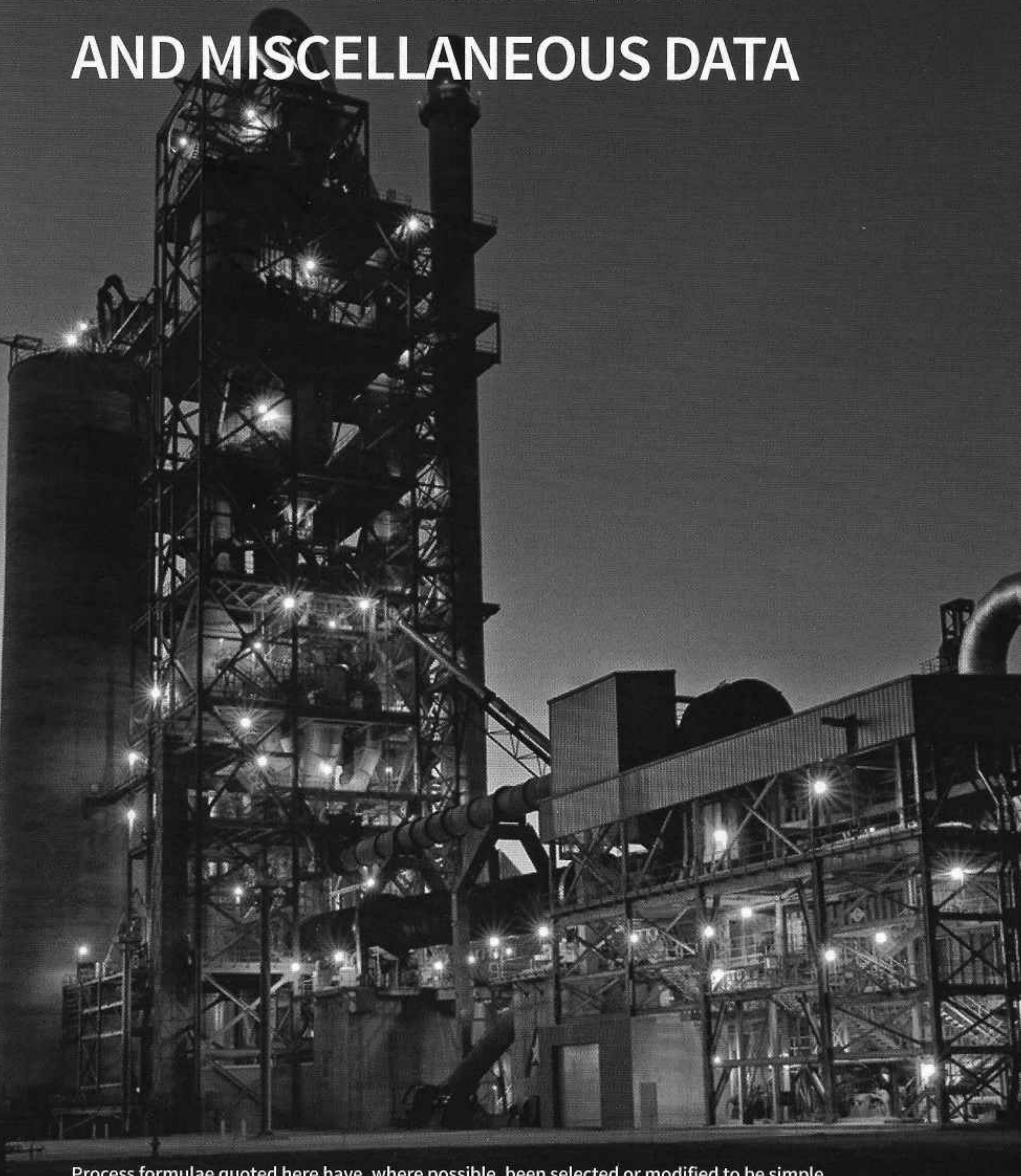
Table 15.8 Selected multiples of historical transactions

Acquirer	Acquired	EV/t (EUR)	EV/t (US$)
Lafarge SA (France), 2000	Blue Circle (UK)	191	176
Cemex (Mexico), 2004	RMC (UK)	172	214
Ciments Français (France), 2005	Suez Cement (Egypt)	212	264
Lafarge SA (France), 2008	Orascom (Egypt)	338	497
Holcim (Switzerland), 2014	Lafarge (France)	115	153
HeidelbergCement (Germany), 2015	Italcementi (Italy)	89	99
Nirma (India), 2016	Lafarge (India)	115	127
HeidelbergCement (Germany), 2017	Cementir (Italy)	48	55
CRH (Ireland), 2017	Ash Grove (USA)	300	339
Semen Indonesia (Indonesia), 2018	LafargeHolcim (Indonesia)	100	118
OYAK (Turkey), 2018	Cimpor (Portugal)	93	110
Vicat (France), 2018	Ciplan (Brazil)	139	165

Source: CM CIC Securities, ICR Research

Note: US$ values use average annual EUR:US$ exchange rate

SECTION B – PROCESS CALCULATIONS AND MISCELLANEOUS DATA



Process formulae quoted here have, where possible, been selected or modified to be simple rather than precise. They are largely intended as 'sanity checks' and, for the purposes of process engineering or design, more precise formulae are usually necessary.

A further selection of useful data and formulae is available in the "Cement Formulae Handbook" (Confederation of Indian Industry, 2010).

Note that metric tonnes are abbreviated as 't' and short tons (2000lb) by 'st'.

B1. Power

B1.1 Specific power consumption

Power consumption rose significantly with the introduction of dry-process kilns and has continued to increase with conversion to solid fuels, increased fineness of cement and with the demands of environmental protection.

Typically, power consumption is 80-100kWh/t cement, which may be broken down as follows:

- 5 per cent for raw material extraction and blending
- 24 per cent for raw material grinding
- 6 per cent for raw material homogenisation
- 22 per cent for clinker production
- 38 per cent for cement production
- 5 per cent for conveying, packing and loading

Source: Cembureau, 2013

B1.2 Power conservation

Conservation of electrical power, allied to achieving satisfactory performance as discussed in Section 13.4, should first address such areas as:

- blending (if turbulent) – convert to CF
- pneumatic conveying – convert to mechanical
- milling (ball) – install pregrinding; optimise media grading and charge; maintain diaphragms and liners
- install expert system and maximise throughput
- ID fans – eliminate air in-leakage, use high-efficiency impellers and variable speed drives (not dampers)
- preheater cyclones – install low pressure drop designs
- 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; install modern efficient lamps
- eliminate avoidable stoppages.

Energy efficiency should obviously be built into original plant design. For existing plants, most remedies involve significant capital cost which must be justified by anticipated power savings (Shenoy and Chacko, 1997)

B1.3 Three-phase power

- $\text{kVA} = \sqrt{3} \times A \times V/1000$
- $\text{kW} = \sqrt{3} \times A \times V \times \sigma/1000$

In AC circuits, the power factor (σ) is the ratio of the real power (watt) that is used to do work and the apparent power (volt-amperes) that is supplied to the circuit (which equates to the cosine of the phase angle between current and voltage).

Actual voltages vary: typically, incoming power may be at 60-70kV transformed to 4160-6300V for large motors (>1000kW), then to 380-460V (three-phase) for intermediate equipment (3-1000kW) and 110-220V (single-phase) for small equipment and lighting.

B1.4 Motor power output

- shaft kW = $\sqrt{3}$ x A x V x σ x E/1000
  where: E = motor efficiency

B1.5 Peak power tariffs

These tariffs allow significant 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 (1996).

B1.6 Power generation

Major fossil-fuelled power generators typically operate at 35 per cent efficiency, employing subcritical boilers (operating below the critical point of water – 647K and 22MPa). Thus 1t of coal with 6000kcal/kg net heat value will produce approximately 2.5MWh of power. Efficiency is 40-45 per cent for supercritical pulverised coal plants.

Note that coal burning produces on average 10 per cent ash, of which 70 per cent is fly ash and 30 per cent bottom ash.

B1.7 Cogeneration

The generation of electric power from kiln waste heat from existing kilns is viable when raw materials are dry but is conserved in new plants using five- or six-stage preheaters and high-efficiency clinker coolers. Also, perhaps, when power tariffs are particularly high or where cogeneration is subsidised or mandatory (Huckauf and Sankol, 2000, and Legmann, 2000). Retrofits may be viable, and recent activity is referenced in Section 8.4.

B2. Fans and air handling

Fans are essential and ubiquitous gas handling equipment in modern cement plants, consuming some 30 per cent 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 (Figure 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 the 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 the 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 useful rule-of-thumb for gas handling is that system pressure drop is proportional to the square of gas volume flow rate.

B2.1 Fan laws

- shaft kW = $Q \times \delta p / 367{,}000 \times \eta$

  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 per cent may be added to base power.

For a fan with fixed inlet and discharge resistance, the effects of changing rpm are:

$$Q_2/Q_1 = (rpm_2/rpm_1) \qquad p_2/p_1 = (rpm_2/rpm_1)^2 \qquad kW_2/kW_1 = (rpm_2/rpm_1)^3$$

For a fan with fixed speed and fixed resistance, the static pressure and kW increase in proportion to gas density (ρ) are:

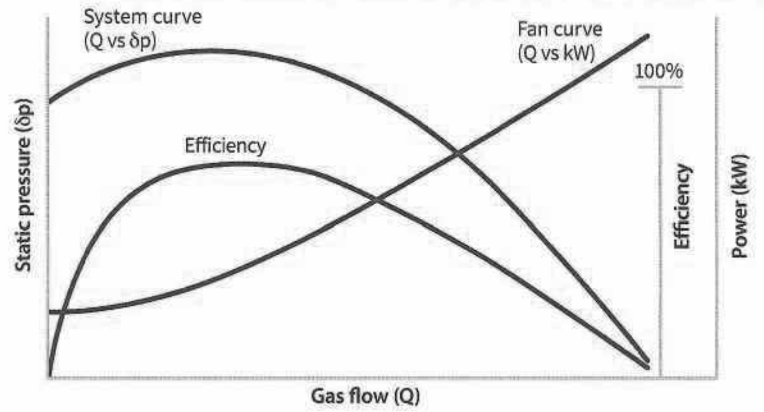
$$\rho_2/\rho_1 = (r_2/r_1) \qquad kW_2/kW_1 = (r_2/r_1)$$

For a fan with fixed speed and fixed resistance, the effects of changing impeller diameter (D) are:

$$Q_2/Q_2 = (D_2/D_1)^3 \qquad \rho_2/\rho_1 = (D_2/D_1)^2 \qquad kW_2/kW_1 = (D_2/D_1)^5$$

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

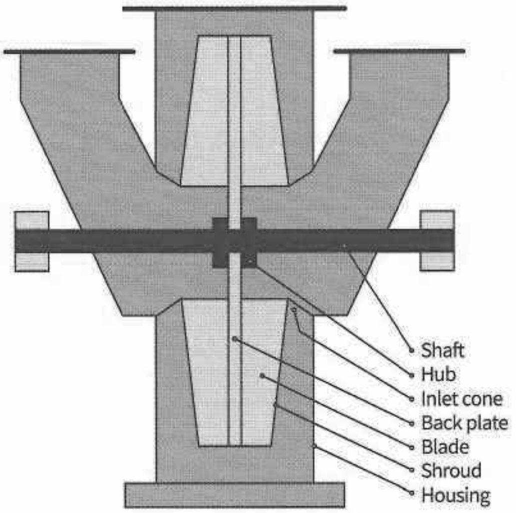


The two impeller geometries most common for large fans in the cement industry are:

- radial blade – which show relatively little variation of static pressure with flow over the operating range. Efficiency is 55-60 per cent, increasing about five per cent if the impeller is closed sided and decreasing by five per cent with dust entrainment.
- backward curved blade – which provide the relatively low static pressure for a given wheel diameter. They are, however, typically 75-80 per cent efficient.

Due to rising power costs, high-efficiency backward curved fan wheels are largely replacing the traditional paddle wheel design (Saxena, 1995). 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. Therefore, the diameter of the shaft at the bearings is often reduced significantly to lower shaft peripheral speed, resulting in less heat generation and easier lubrication. While this may give bearings loaded to only about two per cent of their dynamic rating, it is considered that loading of less than 1.5 per cent 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 200µm 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 Sleeve oil), which are reliable and have high load-carrying capacity. However, such bearings do not like low-speed operation where the lubricant film between the shaft and the bearing may be deficient. Higher-viscosity lubricant may help for low-speed operation when draughting a mill or kiln during maintenance, but baghouse and kiln ID fans should not normally be operated below the bearing supplier's 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 that 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 often be corrected by consideration of impeller material, blade design, or the addition of guide vanes (Krift, 1994, and Gutzwiller & Banyay, 2000). Backward curved impellers are significantly less prone to build-up than radial types. Failing prevention, automatic balancing systems are available which mount to the fan shaft and continuously monitor and correct balance (Winzenz, 2008, and Rizzo, 2006).

Fan vibration that 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 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

Table B2.1 Gas properties

Gas	Molecular weight	Density (g/l)	Specific heat (cal/g/°C)	Latent heat of evaporation (cal/g)
Air		1.293	0.237	
CO_2	44	1.977	0.199	
CO	28	1.250	0.250	
O_2	32	1.429	0.218	
N_2	28	1.250	0.248	
NO	28	1.340	0.232	
NO_2	46	2.054	0.194	
SO_2	64	2.927	0.154	
H_2O (100°C)	18	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 shown in Table B2.2.

Table B2.2 Recommended air pressures

Air slides	0.4-0.6kg/cm^2	6-8psi
Controls	6-8kg/cm^2	80-110psi
Cleaning	8-12kg/cm^2	110-170psi

For controls (instrument air) and for baghouse pulse cleaning, the air should be cooled, dried and de-oiled (Guilman, 1994). Jack-hammers and air-lances should have a separate air supply.

B2.6 Pitots, orifices and venturis

- Gas velocity, V (m/s) = $4.43\,K\sqrt{\delta p/\rho}$
  where: K = coefficient
  δp = differential pressure (mmH$_2$O)
  ρ = gas density (kg/m^3)
- Gas flow, Q (m^3/h) = 3600.A.V
  where: A = orifice area (m^2)
  V = velocity (m/h)

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 0.85 (Perry and Green, 2008). K values should be provided by the manufacturer. Note that sometimes a pressure coefficient is encountered and applied inside the square root – this value is the square of the velocity coefficient.

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*). This may be simplified for approximate process measurements, using fewer points. Measurement should be, if possible, at least eight diameters downstream and two diameters upstream from any bend or irregularity. Note that flow should be calculated from the arithmetic mean of all velocity measurements, not the mean of pitot velocity pressures.

B2.7 False air

Air in-leakage through an aperture of area A (m^2) with pressure differential δp (mmH$_2$O) can be approximately calculated from:

- Volume (m^3/h) = $8900 \times A \times \sqrt{\delta p}$

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

Table B2.3 Dust loading (g/Nm^3)	
Bin vent	5-15
Belt transfer point	15-20
Preheater exhaust	50-75
Long kiln exhaust	<30
Short kiln exhaust	<60
Bulk loading	40-60
Ball mill – gravity discharge – air swept	20-80 300-500
Roller mill internal	650
FK pump vent	15-200

Source: Duda, 1985

B2.9 Stack draught

- Theoretical draught (mmH_2O) = 0.46HP ($1/T_a$-$1/T_s$)
  where: H = stack height (m)
  P = atmospheric pressure (mmHg)
  T_a = ambient temperature (K)
  T_s = average stack temperature (K)

Note: there will be some reduction in draught from friction losses.

B2.10 Dewpoint of moist air at atmospheric pressure

Table B2.4 Moist air dewpoint

Temperature (˚C)	mg H_2O/l	Moisture	
		(% of dry vol)	(% of wet vol)
10	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

- Water (l/h) = $Q.\rho.S.\delta T/[(100-T_w)+539]$
  where: Q = gas flow (m^3/h)
  ρ = gas density (kg/m^3)
  S = specific heat of gas
  δT = gas cooling (°C)
  T_w = water temperature (°C)

More approximately, for kiln exhaust gas with five per cent oxygen, water required to reduce gas temperature by δT(°C) is:

- l/min water = tph clinker x kcal/kg/1000 x $\delta T/66$

B2.12 Abrasion resistance

Air-entrained particulates are liable to abrade fan impellers and housings, particularly in cooler exhaust or recirculation fans. Apart from hardfacing, 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 and Revillot, 1999).

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, 2004).

Pneumatic conveying lines are prone to failure at elbows and sharp bends are best avoided, if possible. 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

Table B3.1 Comparative power consumption for lift

Mode of lift	Power consumption (kWh/t/100m)
Screw pump	1.20
Airlift	1.10
Dense-phase pump	0.59
Bucket elevator	0.41

Source: Fujimoto, 1993

B3.2 Pneumatic conveying

'Fluidised dense-phase conveying' is appropriate for cement and raw meal, using the fluidisation and air retention properties of the materials to provide a reliable and efficient method of conveying materials. Di Matteo (2012) describes adaptations to handle a range of alternative fuels.

Fuller Kinyon (FK) Pump - This is a pneumatic conveyor comprising a decreasing pitch screw that compresses a material seal inside a barrel section, followed by injection of fluidising and conveying air. Power consumption is 0.8-1.5kWh/t/100m combined for screw drive and compressor, screw speed is 750-1500rpm and air pressure is 1-2.5kg/cm^2. The original "H" pump has a screw 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. A capacity range of up to approximately 500tph is possible, but system sizing is complex, depending principally upon line length and configuration (Hunlich and Bogdanski, 1997). Design air volume is typically 0.012Nm^3/kg material. Material flow can be measured using radar technology (Anon, 2004).

Air Lift (Aeropol) - This conveyor comprises a vertical cylindrical pressure vessel, with a bottom aeration pad and conveying air nozzle passing upwards through its centre. A rising pipe (conveying line), open at the bottom, is situated just above the 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 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 and, if the kiln is ID fan limited, will de-rate production. Approximately, the gas flow at the top of a preheater with 850kcal/kg and five per cent 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 per cent of preheater exhaust.
- An Air Lift will deliver 0.048Nm^3 x 1.6 = 0.077Nm^3/kg clinker or five per cent 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 (tph)
  H = height (m)
  k = coefficient 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 size and 250°C temperature but should not exceed 60m in height. **Belt elevators** are normally limited to material sized <3mm and <120°C but can exceed 100m in lift.

Elevator speed up to 0.7m/s results in gravity discharge, while above 1.5m/s the load is ejected centrifugally (Labahn and Kohlhaas, 1995).

B3.4 Belt conveyor power

Calculation for belt conveyors is complicated by the inclination and number of rollers, scrapers, etc (Labahn and Kohlhaas, 1995). Belt speed is typically 2m/s, less for fine material, more for coarse material and wide belts. Construction and operating costs are discussed in Anon, 2005.

B3.5 Screw conveyor power

- kW = 2.25 (L+9)(C/530)
  where: L = length (m)

B3.6 Airslide

Airslides are inclined at 4-10° with aeration compartment lengths of up to 5m. Air pressure is typically 400mmH_2O, air volume $2m^3/m^2/min$, and power consumption approximately 1W/t/m (Labahn and Kohlhaas, 1995). Dust collection is required to maintain the 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, these shape the 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 the ability to climb steeper gradients and turn tighter corners. There are variations of design (www.beumer.com), some of which involve special belting. Installation cost and power consumption are comparable to troughed belts, but maintenance is less. Tube conveyors up to 850mm diameter have been built carrying 2500tph of material up to 400mm in size and suppliers now offer higher capacities.

B3.9 Air-supported belt conveyor

An interesting variation using an air cushion rather than support rollers is offered by Martin Engineering, claimed to offer significant energy, environmental and safety advantages (Saver, 2007).

B3.10 Sandwich conveyor

Troughed belt conveyors are limited in elevating loads to angles of less than 15°. This limitation can be overcome by using a sandwich belt, which allows conveying angles of up to 90° (Dos Santos and Stephan, 2012).

B3.11 Modified belt conveyor

Magaldi Corporation offers "Superbelt" conveyors for original installation or retrofit to belt systems in which steel plates (for horizontal transport) or steel pans (for inclines) are fastened to a steel mesh belt. The belt can run on an existing belt frame and usually employs the same drive. Benefits include high temperature capability and much reduced risk of belt failure (www.magaldi.com).

B3.12 Capsule conveyor

An innovative system is Sumitomo's 3.2km pneumatic "Capsule-Liner" conveyor for limestone, claimed to involve lower capital cost, lower maintenance, and lower noise than a belt conveyor (Anon, 1994). It conveys 2Mta in trains of three 1.6t capsules.

B3.13 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

B4.1 Sieve sizes

Table B4.1 Sieve sizes (ASTM E11)

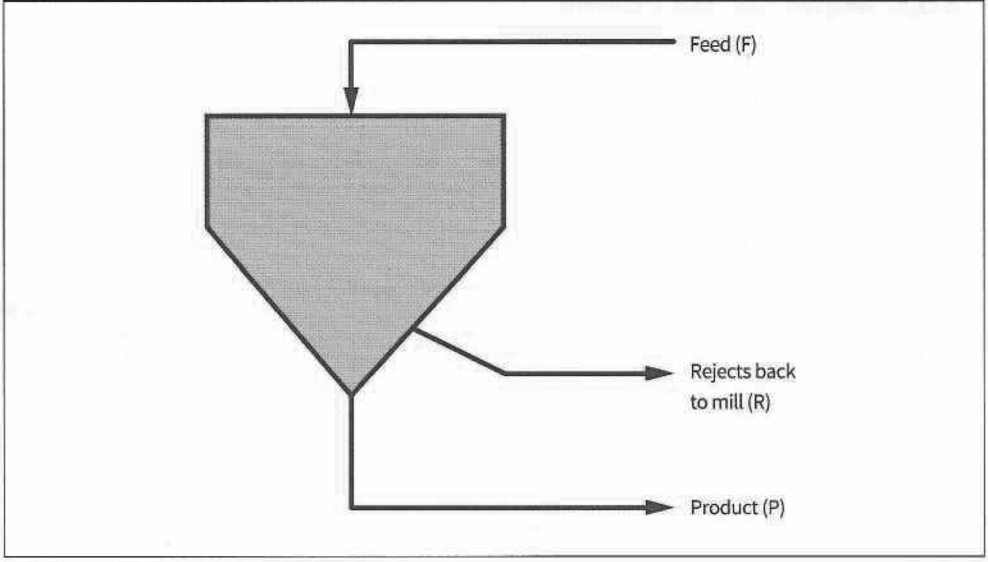
Sieve no.	Sieve size (mm)	Sieve no.	Sieve size (µm)	Sieve no.	Sieve size (µm)
4	4.75	50	300	200	75
8	2.36	100	149	325	45
16	1.18	120	125	400	38
30	0.60	170	90		

B4.2 Circulating load

Circulating load refers only to the classifier and is 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 is actually the "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 over a period of 15-20 minutes. If necessary, the samples should be reduced using a riffle or other valid sample division method.

Figure B4.1 Separator feed and discharge



- Circulating load – CL = R/P = (p-f)/(f-r)
- Circulating factor – CF = F/P = (p-r)/(f-r)
- Separator efficiency – E = p(f-r)/f(p-r)
  where: F = feed to classifier (tph) f = proportion passing (%)
  P = product from classifier (tph) p = proportion passing (%)
  R = rejects back to mill (tph) r = proportion passing (%)
- Recovery (%) = 100 x f(p-r)/p(f-r)

Samples are taken at the separator during stable operation. Each sample is screened at 90µm or 75µm for raw milling and 45µm for cement milling. The specific surface area of cement samples may also be used.

B4.3 Classifier recovery

Classifier recovery is defined as the fraction of fines present in the feed which is recovered 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 Section A6.2:

- % recovery = 100 x f(p-r)/p(f-r)

B4.4 Tromp curve

This is a graphical representation of the probability of a particle in the classifier feed exiting with the rejects. This can also be expressed as the 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 1–100µm for cement, or 1–200µm for raw meal is performed. Figure B4.2 shows typical curves for a mechanical and a high-efficiency separator, respectively.

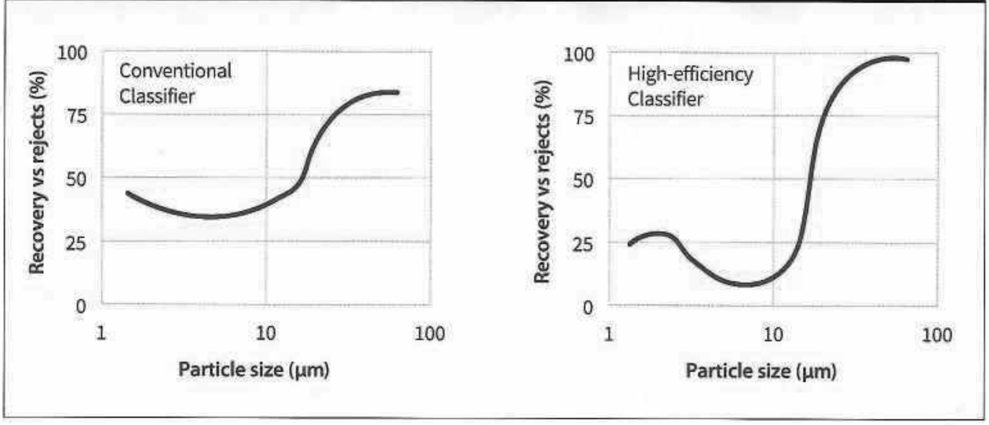


Figure B4.2 Tromp curves

The selectivity for each increment is plotted against the mid-point of each particle size increment. This is conveniently plotted on semi-log paper from which the particle diameters corresponding to 25 per cent (S_{25}), 50 per cent (S_{50}) and 75 per cent (S_{75}) probability of rejection are determined.

- **Apparent bypass** (Bp) is the proportion of fines that are rejected. A perfect separator would have zero per cent bypass and an efficient unit <20 per cent.
- **Classifier cut size** (D_{50}) is the particle size corresponding to a 50 per cent 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 one. 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.

B4.5 Mill critical speed

Critical speed (Cs) is the rotational speed in rpm of a tube-mill at which centrifugal force just counters gravitation and holds the charge against the shell. Raw mills usually operate at 72-74 per cent of critical speed and cement mills at 74-76 per cent.

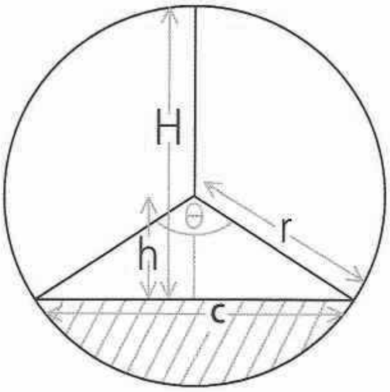
- $Cs = 42.3/\sqrt{D}$
  where: D = effective mill diameter (m)
- $\% Cs = 100n/Cs$
  where: n = actual mill speed (rpm)

B4.6 Charge volume loading

- Loading – $V_L (\%) = 100[(\pi r^2(\theta/3600 - (h\sqrt{r^2 - h^2})]/\pi r^2$
  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
  $\cos 1/2\, \theta = h/r$

Note: only the free height (H) and internal diameter (D) need to be measured directly.

Figure B4.3 Ball mill charge volume



For normal charge levels, charge volume loading calculations are shown in Table B4.2.

Table B4.2 Charge volume loading

h/D	V_L (%)	h/D	VL (%)
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

B4.7 Grace factor and other ball mill parameters

Instantaneous clinker charge – M_I (kg)	$M_I = M_R R(C_L+100)/6000$
Clinker bulk volume – M_V (m^3)	$M_V = M_I/M_D$
Ball charge volume – G_V (m^3)	$G_V = \pi r^2 L V_L/100$
Ball charge weight – G_W (kg)	$G_W = 4550\ G_V$
Steel:clinker ratio – S_c	$S_c = G_W/M_I$
Ball charge void space – G_s (m^3)	$G_s = 0.42 G_V$
Void fill by clinker – V_F (%)	$V_F = 100\ M_V/G_s$
Grace factor – G (kg/min/ m^3)	$G = M_R(C_L+100)/6000G_s$

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 per cent (Welch, 1984).

Retention time is determined using fluorescein (7g/10tph of combined mill feed and circulating load). Samples taken at discharge at 30s intervals are shaken with water, filtered, and the solutions observed under UV light to determine peak concentration. If XRF equipped for zinc analysis is available, 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.

B4.8 Mill power

Bond's Third Theory of Comminution allows an approximate estimation of specific power consumption (kWh/t) from the work index, Wi of the material being ground (see 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 µm size at which 80 per cent of feed passes
  f_2 = the µm size at which 80 per cent 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):

- Bond work index – Wi (kWh/t) = $435/Hg^{0.91}$

Source: Duda, 1985

Approximate mill power consumption can also be calculated from:

- Mill power consumption (kW) = $7.5G_W\sqrt{D}$
  where: G_W = ball charge (t)
  D = effective mill ID (m)

- The energy consumption in kWh/t to produce cement is generally reckoned to be proportional to the surface area of the cement raised to the power of 1.5.

Traditionally, ball mills have been driven by single-girth gear-and-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 and Grothaus, 1994). Ahrens & Gonser (2007) discuss the advantages of gearless drives.

B4.9 Ball weight and surface area

Table B4.3 Ball weight and surface area

Ball diameter (mm)	Individual ball weight (kg)	No. of balls/t	Ball surface area (m^2/t)
20	0.033	30,600	38.46
25	0.064	15,700	30.77
30	0.110	9100	25.64
40	0.261	3830	19.23
50	0.511	1960	15.38
60	0.882	1130	12.82
70	1.40	710	10.99
80	2.09	480	9.61

Steel density is assumed $7.8g/cm^3$. Bulk density of a mixed ball charge may be taken as $4550kg/m^3$.

B4.10 Maximum ball size required

- Ball diameter (mm) = $25.4\sqrt{(f_1W_i/200C_s)\sqrt{(s/3.28.D)}}$
  where: s = specific gravity of feed
  C_s = percentage 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.

B4.11 Measurement of wear

Ball wear – typically reduced to around 20-50g/t with modern high-chrome alloys – 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 keeping a record of material ground and charge added periodically to recover a given level of mill power.

Mill liners should be measured periodically to determine wear 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 localised wear that must be monitored to anticipate replacement. Note that diaphragms normally have about 15 per cent open area (slots plus centre screen) and that the slots are tapered outward on the discharge side to prevent plugging.

The steel consumption for roller mills is determined from weight loss of rolls and table during a campaign, to which must be added the weight of build-up metal used for resurfacing.

B4.12 Effects of gypsum upon setting time

Figure B4.4 Effects of gypsum form upon setting time

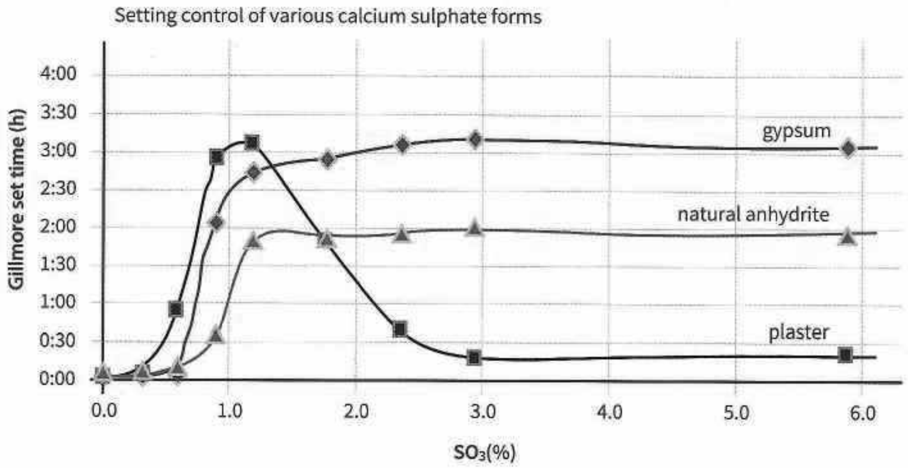
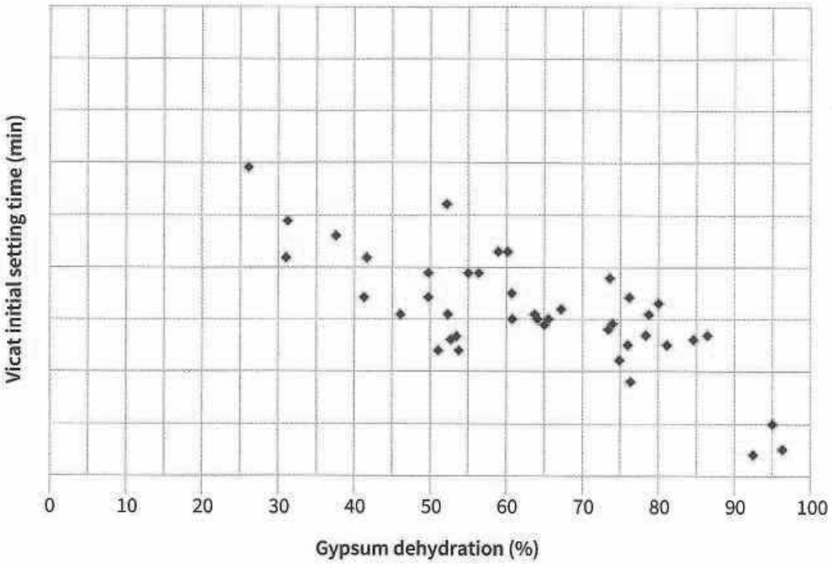


Figure B4.5 Effects of gypsum dehydration upon setting time



B5. Kilns and burning

B5.1 Cement compounds and ratios

The Bogue calculations are based upon phase equilibria in the quaternary system of the pure oxides of calcium, silicon, alumina and iron. They lead to the following equations for the three potential compound compositions:

- $C_3S = 4.071Ca - 7.600Si - 6.718Al - 1.430Fe - 2.852S$
  $C_2S = 2.867Si - 0.754C_3S$
  $C_3A = 2.650Al - 1.692Fe$
  $C_4AF = 3.043Fe$
  $LSF = (Ca + 0.75Mg^*)/(2.85Si + 1.18Al + 0.65Fe)$
  $S/R = Si/(Al + Fe)$
  $A/F = Al/Fe$
  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 two per cent, use constant 1.5 for this term*

B5.2 Coating tendency

- Coating tendency = $C_3A + C_4AF + 0.2C_2S + 2F$

Note: index <28 indicates light coating
>30 indicates heavy unstable coating, rings and snowmen
(Source: 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:

- Free lime_{1400} (%) = $0.31(\%\ LSF-100) + 2.18(S/R-1.8) + 0.73Q + 0.33C + 0.34A$
  where: Q = +45µm residue after acid wash (20 per cent HCl) identified by microscopy as quartz
  C = +125µm residue which is soluble in acid (ie coarse limestone)
  A = +45µm residue after acid wash identified by microscopy as non-quartz acid insoluble

Note: Q, C and A are expressed as share (%) of total raw mix sample
(Source: DuToit, 1997)

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

Table B5.1 Theoretical heat of clinker formation

	Heat required (kcal/kg)
Heat in	
Heat raw materials 20-450°C	170
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
Total	**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
Total	**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 gas velocities

Upper limits	
Through cooler grate	5 Am/s
Hood	6
Under cooler bull-nose	15
Burning zone (1450°C)	9.5
Feed end transition (1000°C)	13
Riser	24
Preheater gas ducts – (#1, top)	8-10
– (#4/5, bottom)	13-16
Lower limits	
Tertiary duct	25
Pulverised coal conveying	20

Note that entrained dust increases with $(\text{gas velocity})^3$.

B5.7 Kiln heat balance (referred to 20 °C)

Table B5.2 Kiln heat balance

	Heat value (kcal/kg)	Four-stage SP (typical)	Five-stage PC (typical)
Heat in			
Kiln feed (__kg x __ °C x 0.22)		15	15
Fuel (__kg x __kcal/kg NCV)		835	735
Total air (__Nm³ x 1.293 x 0.237 x __ °C)		0	0
Total kcal/kg clinker in		850	750
Heat out			
Theoretical heat of clinker burning		420	420
Moisture evaporation (__kgKF x __% x 5.40)		5	5
Kiln exhaust (__Nm³ x D x SH x °C)		210	151
Entrained dust (__kg x 0.23 x °C)		3	7
Bypass gas (__Nm³ x D x SH x °C)		-	-
Bypass dust (__kg x SH x °C)		-	-
Cooler exhaust (Nm³ x 1.293 x 0.240 x __ °C)		115	85
Clinker (1kg x 0.20 x __ °C)		14	14
Radiation, preheater		35	40
kiln		45	25
cooler		3	3
Total kcal/kg clinker out		850	750

Notes:

1. All weights and gas volumes are relative to 1kg clinker.

2. Temperatures (°C) are relative to 20 °C (or ambient).

3. Specific heat values 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

(Hewlett, 2003)

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 per cent moisture x latent heat. If coal is directly 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 B5.13 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.

B5.8 Kiln specific heat loading (SHL)

- Heat loading, kcalx10^6/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 internal 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 included in Table B5.4.

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 per cent.

Table B5.3 Slope degree vs per cent

°	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 (tph)
  A = effective cross-sectional area (m^2)

Table B5.4 Kiln volume loading

Typically	SHL	FR	L/D	Kiln speed (rpm)	Vol loading (%)
Long dry kiln	1.2	1.55	35	1.6	16.8
Preheater kiln	2.0	1.45	16	2.0	9.2
AT precalciner	3.0	1.22	15	3.0	7.3
AS precalciner	4.0	1.11	15	3.5	7.6
New precalciner	5.0	1.05	12	4.0	7.2

Alternatively, the specific kiln capacity is expressed in relation to the effective kiln volume as tonnes of clinker produced/ m^3/day with typical values:

SP without riser firing	2.0t/m^3/d
ATC or SP with riser firing	2.8t/m^3/d
ASC	4.5t/m^3/d

Very large SLC kilns are now being designed with over 4.5t/m^3/d, and several are reported to be operating sustainably at 6.5t/m^3/d.

B5.11 Kiln capacity vs diameter

Assume: Heat loading for SP kiln = 2tpd/m^3
PC kiln = 4.5tpd/m^3 with 50:50 fuel split
20cm brick thickness

Table B5.5 Capacities (tpd)

Kiln Φ x L (m)	Cross-sectional area (m^2)	Volume (m^3)	SHL (kcalx10^6/ h/m^2)	SP capacity (tpd)	PC capacity (tpd)
3.6 x 54	8.04	434.2	3.6	868	1954
4.0 x 60	10.18	610.8	4.0	1222	2749
4.4 x 66	12.57	829.6	4.4	1659	3733
4.8 x 72	15.21	1095.1	4.8	2190	4928
5.2 x 78	18.10	1411.8	5.2	2824	6353
5.6 x 84	21.24	1784.2	5.6	3568	8029
6.0 x 90	24.63	2216.7	6.0	4433	9975

If PC fuel split is 60:40, a 6m kiln can produce 10,300tpd at 6kcalx106/h/m^3.

B5.12 Kiln drive power

- Power (kW) = $\pi L(D/2)^2/4.7$

Kiln power during normal operation should be about half the installed power.

B5.13 Cooler efficiency

- E (%) = $100[C_1 - (V + C_2 + R)] / C_1$
  where: C_1 = heat content of clinker from kiln (kcal/kg)
  C_2 = heat content of clinker out (kcal/kg)
  V = heat content of cooler vent air (kcal/kg)
  R = cooler radiation

Typical efficiencies are 60-70 per cent for a conventional grate cooler, 70-75 per cent for an air-beam, and 75-78 per cent for a cross-bar system.

B5.14 Kiln exhaust gas (coal)

Coal analysis, per cent by weight:

C	75.0 per cent
H	5.0
S	1.0
O	9.0
N	1.0
Ash	8.0
Net (kcal/kg)	7100
Moisture	10.0

with indirect firing, and specific fuel consumption of 800kcal/kg clinker.

Then coal consumption is 0.1127kg/kg clinker and combustion gases produced are:

- C 0.80 x 113 → 84.75g x 22.4/12 = 158l CO_2 = 226g O_2
- H 0.05 x 113 → 5.65g x 22.4/2 = 63l H_2O = 45
- S 0.01 x 113 → 1.13g x 22.4/32 = 0.8l SO_2 = 1
- N 0.01 x 113 → 1.13g x 22.4/14 = 1.8l NO_2 = 3
- O 0.09 x 113 → 10.17g
- *Total O_2* *323g O_2*

Then: added O_2 required for combustion = 275g - 10g = 265g = 186l or 0.186Nm³

Then: equivalent N_2 from air = 185l x 79/21 = 695l or 0.695Nm³

CO_2 from calcination of raw meal to yield 1kg clinker (assuming kiln feed LoI of 35 per cent):
[(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 per cent H_2O):
1kg x 1.65 x 0.005 = 8.25g = 10l or 0.01Nm³

Then: total H_2O in exhaust gas = 10l + 63l = 73l or 0.073Nm³

Then: exhaust gas with no excess air, is:

- CO_2 0.432Nm³ = 36.0 per cent
- H_2O 0.073Nm³ = 6.1 per cent
- SO_2 0.001Nm³ = 0.1 per cent (830ppm)
- N_2 0.695Nm³ = 57.8 per cent
- *Total* *1.201Nm³*

Estimation of net exhaust gas volume (Nm³/kg clinker @ 0 per cent excess O_2):
= (kcal/kg x 0.00115) + 0.284

Estimation of gross exhaust gas volume, (Nm³/kg clinker with n per cent O_2):
= Net Nm³/kg x (1 + (n / (21-n))

Physical data of exhaust gas with various levels of (dry) excess air are shown in Table B5.6.

Table B5.6 Physical data of exhaust gas

O_2 content (%)	Density (g/l)	SH (cal/g/°C)	Dewpoint (°C)
0	1.513	0.245	38
2	1.491	0.244	36
5	1.458	0.243	33
10	1.403	0.241	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 0.00115
  – 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.

B5.15 Circulation of volatile components

K, Na, S and Cl are all subject to partial evaporation at kiln burning zone temperatures (see table B5.7).

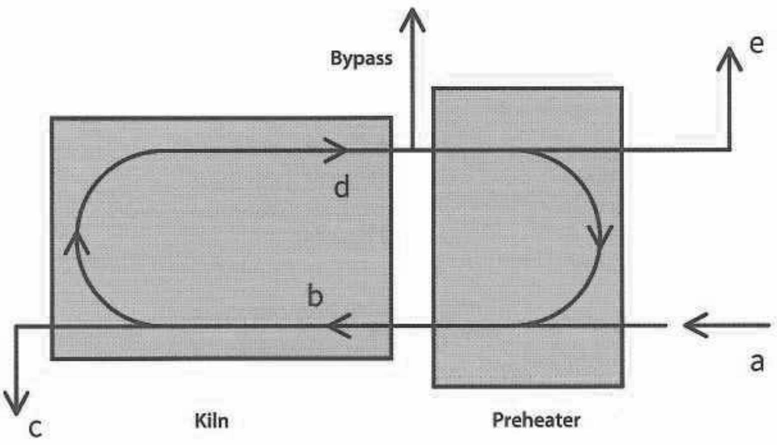
Table B5.7 Melting and boiling points

	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	896D
B Pt (°C)	1500S	1413		1689					

Eutectics have even lower melting points.

Volatilisation in the burning zone and condensation in the preheater may be represented as shown (Norbom, 1973). The external cycles through dust collector are not considered; if dust is not dumped, then virtually all 'e' is returned to the kiln.

Figure B5.1 Kiln volatile cycle



Conventional factors defining circulation and typical values for SP kilns without bypass are shown in Table B5.8.

Table B5.8 Conventional factors defining circulation and typical values for SP kilns (without bypass)

		K_2O	Na_2O	SO_3	Cl
Primary volatility	E1 = d/b	0.67	0.53	0.55	0.99
Secondary volatility	E2 = d/b	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)			

Source: Ritzmann, 1971

(Primary volatility applies to raw feed and secondary volatility to recirculated material. SO_3 volatility increases with stoichiometric excess over alkalis.) But note the more complete discussion given in Section A5.5, considering dissociation of $CaSO_4$ in the presence of CO.

B5.16 Estimation of kiln bypass

Table B5.9 Volatilisation factors

	K_2O	Na_2O	SO_3
Without Cl-	0.5	0.42	0.9
With Cl-	0.7	0.55	0.9
Recycle	0.9	0.8	0.9

Table B5.10 Bypass effect

K	Without Cl^-		With Cl^-	
Bypass (%)	Vol. factor	K reduction (%)	Vol. factor	K reduction (%)
10	0.5	26.3	0.7	36.8
20	0.5	35.7	0.7	50.0
30	0.5	40.5	0.7	56.8
40	0.5	43.5	0.7	60.9
50	0.5	45.5	0.7	63.6
60	0.5	46.9	0.7	65.6
Na	Without Cl^-		With Cl^-	
Bypass (%)	Vol. factor	Na reduction (%)	Vol. factor	Na reduction (%)
10	0.42	15.0	0.55	19.6
20	0.42	23.3	0.55	30.6
30	0.42	28.6	0.55	37.5
40	0.42	32.3	0.55	42.3
40	0.42	35.0	0.55	45.8
60	0.42	37.1	0.55	48.5
(Duda, 1985, p458)				
Cl				
Bypass (%)	Cl reduction (%)			
1	68			
2	81			
3	86			
4	89			
5	91			
10	95			
(Sutoh et al, 1998)				

B5.17 Other kiln types

- Long wet
- Long dry
- Semi-wet SP
- Grate preheater
- Vertical shaft
- Fluidised bed
- Mini-kilns

The earliest kilns were vertical shafts burning mixtures of raw materials and solid fuel in a natural draught of combustion air. Long rotary kilns allowed more uniform heat transfer and controlled clinker burning, initially using slurry feed, as this facilitated raw material grinding and homogenising. While long kilns were excellent devices for heat transfer, nodulisation and materials handling at clinker burning temperatures, they were inefficient for preheating and calcination. The Lepol or grate preheater system was the first alternative approach, with nodulised raw materials conveyed on a travelling grate permeated by hot kiln exhaust – a successful process with appropriate raw materials.

Ultimately, however, it emerged that the most efficient low-temperature heat exchange and calcination are effected in air suspension, leading first to the cyclone preheater and later the addition of separately-fuelled precalcination. These last two systems now predominate and form the substance of this book.

Typical comparative figures, with considerable variation, are as follows:

Table B5.11 Comparison of different kiln types in terms of capacity, fuel consumption and L:D range

	Maximum rating (tpd)	Specific fuel (kcal/kg)	Length:diameter
Shaft kiln	200	900-1000	
Long wet kiln	2000	1200-1500	32-38
Long dry kiln	2000	900-1200	32-38
Lepol kiln	2000	800- 900	14-16
Cyclone preheater kiln	2000	800- 900	14-16
Precalciner kiln	11,000	650- 850	11-16

Clinker grindability differs significantly with kiln type. Relative power consumptions are:

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

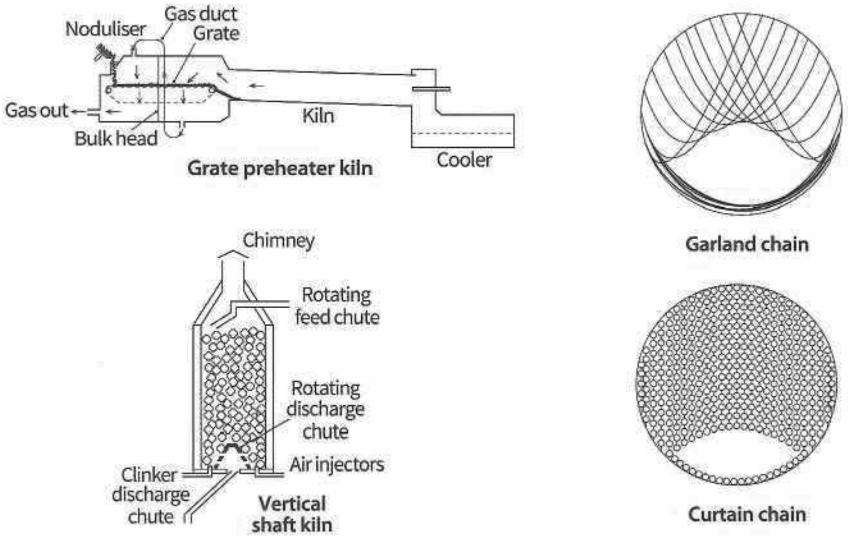
Long wet kiln

Long wet kilns were predominant until cyclone preheaters appeared in the 1950s. Few remain, usually justifying operation where they are fully depreciated, where only a small output is required and fuel is cheap. A few burn hazardous wastes with significant disposal fees and actually benefit from their high fuel consumption. Wet kilns also avoid the need for drying naturally wet raw materials and slurry homogenisation is very effective.

Raw materials are milled with water to form a slurry (30-35 per cent moisture by weight), which is stored and blended with continuous agitation before feeding to the kiln. Water content is adjusted, perhaps using water-reducing additives, to produce the thickest consistency that allows pumping. One per cent less water is equivalent to about 15kcal/kg clinker.

Chains are hung inside the kiln to enhance water evaporation by increasing surface area for heat exchange, to facilitate the transport of pasty drying feed as it transitions through a plastic phase, to clean the inside of the shell, and to detrain dust from kiln exhaust gas by dropping its temperature and hence its velocity (Figure B5.2).

Figure B5.2 Kiln types and chain systems



Duda (1985) recommends chain design parameters of 12 per cent of daily clinker production for total chain loading and surface area of 6-8.5m^2/m^3 of chain section volume. For larger kilns, de Beus (1997) suggests 15 per cent and 6-10m^2/m^3, respectively. Chain consumption is about 100g/t clinker.

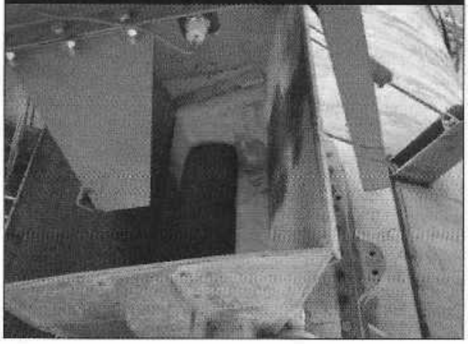
The feed leaves the chain section dry and proceeds to the refractory-lined preheating, calcining and burning zones. Total material retention time is approximately 3h and gas is discharged at 150-200 °C.

Dust loss should ideally be 8-10 per cent but is often much higher as kiln production increases, with a resulting rise in gas velocity and dust entrainment. Return of dust to the slurry system is inadvisable, usually causing stiffening. Up to five per cent relative to clinker weight can be returned by insufflation at the flame. Beyond this quantity, flame cooling becomes unacceptable. Alternatives are separate slurrying and parallel injection with the main feed, or using shell scoops near the hot end of the chain section, or slightly downhill. However, the basic causes of high dust loss are chain design and condition (especially at the hot end) and failure to adequately cool gas near the back end to reduce its velocity and drop out dust.

Long dry kiln

In certain areas, such as the USA, water shortage led to use of long dry kilns, requiring, and resulting in, improved pneumatic blending systems. These differ from wet kilns primarily in raw meal grinding and handling, and in their lower specific fuel consumption. Within the kiln itself, there are only curtain chains as the requirement is for heat exchange and dust detrainment rather than for conveying, and extra chains in hot (higher velocity) gas generate more dust. Usually, 6-7 diameters of curtain chain are employed below about two diameters of bare shell at the feed end. Chain loading is some 10 per cent of daily production and other internal heat exchangers may be added. Gas discharge temperature is typically over 300 °C and, if available, water is sprayed into the feed end to reduce gas temperature before dust collection, or a waste heat boiler may be fitted.

Figure B5.3 Mid-tyre kiln port



Long kilns (wet or dry) can accept fuel such as whole tyres fed through special ports in the mid-kiln area (Figure B5.3) with potential to reduce NO_x emissions, as pioneered by Cadence Inc, who has since added mixing fans to take reactions further towards completion (Reese et al, 2009).

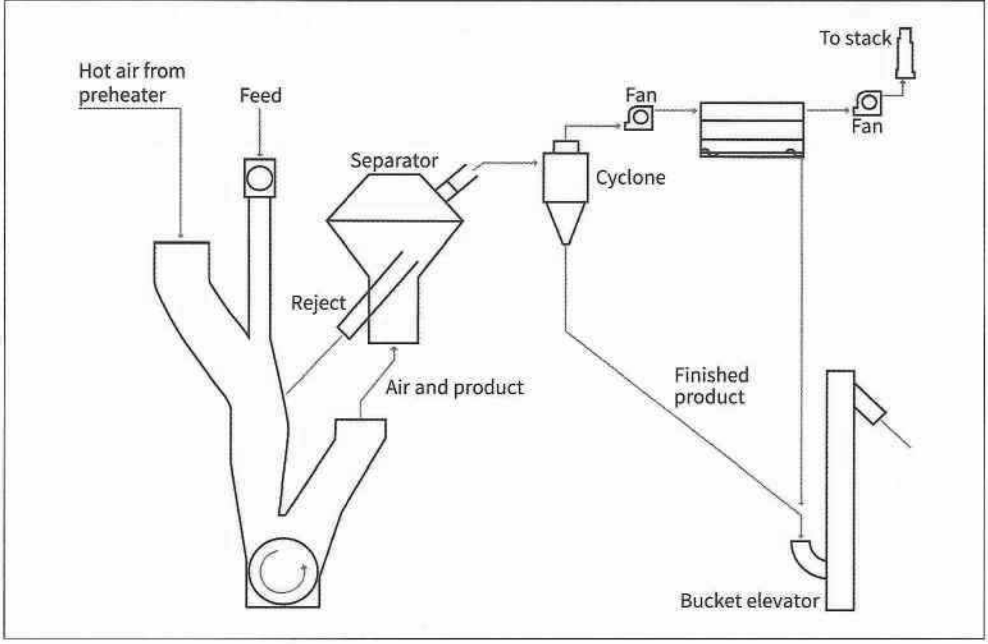
Semi-wet SP kiln

Where raw materials are very moist it may be preferable to proportion and blend them as a slurry and feed this to a preheater via a drier-crusher device, usually after filter pressing to remove some moisture. This is known as the semi-wet SP system (see Figure B5.4). It is important that the nodules which are formed are quite weak, because they must be cracked apart by thermal shock or abrasion before arriving in the preheater. Often only two or three preheater stages are used and, in the best cases, fuel consumption rivals that of a grate preheater.

Lepol (grate preheater) kiln

Polysius introduced this system during the 1930s, achieving a dramatic reduction in fuel consumption compared to the wet process. Dry-milled feed is nodulised with water and moves on a travelling grate traversed by hot kiln exhaust – originally once but, in a later development, twice. The material reaches approximately 900 °C before entering the kiln, while the exhaust gas cools below 150 °C, humidified for dust collection, and is filtered by the material bed to a low dust concentration. Nodulisation uses an inclined rotating pan with a water spray to achieve a moisture content of 11-15 per cent. In some cases feed may be slurried and filter-pressed to yield a cake of about 20 per cent moisture, which may allow soluble salts such as chlorides to leave with filtrate. The cake or nodules must possess good mechanical and thermal stability – usually associated with clay ingredients – to avoid excessive disintegration on the grate and loss of efficiency. The second (low-temperature) exhaust gases pass through the grate to dry and preheat material. The first pass involves an initial gas temperature of about 1000 °C and a final temperature below 500 °C, serving to condense volatiles. A cyclone between the first and second pass can collect dust with a high volatile concentration, providing an effective bypass. The material enters the short kiln at incipient calcination and kilning is similar to that of a preheater kiln (Figure B5.2).

Figure B5.4 Dryer crusher with seprator cyclone and filter



Vertical shaft kilns

Shaft kilns were the only available technology for lime burning in ancient times as far back as the 5th century BC in Greece. They have been largely superseded by rotary kilns, except where very small production units are appropriate and high cement quality is not required. China has closed down many shaft kilns, but examples remain in India, the Himalayas and some other developing countries (Reiter, 1997).

Traditionally, unground feed was roughly mixed with solid fuel and burned in batches with natural draught. Poor feed homogeneity and non-uniform ventilation yielded widely varying temperatures and oxidising conditions, hence poor and erratic quality. Preferably, raw materials and solid fuel are ground together and nodulised ('black meal process'). Alternatively, raw mix and fuel are ground separately and then blended and nodulised ('white meal process'). Rajbhandari (1995) describes Spohn's black meal process as one of the most advanced technologies available (Figure B5.2). Practical capacities are 20-200tpd. As with the Lepol kiln, stable nodules are important and usually require both a clay component and solid fuel with less than 16 per cent volatile content. The shaft is filled with the prepared mix and air is blown in at and near the base. Feed is heated, calcined and burned at progressively higher temperatures, moving down the shaft counter-current to combustion air. Near the base the clinker, with fuel consumed, is rapidly cooled by injected air and discharged through a gate. Production is continuous with new feed added at the top to balance discharge.

The principal difference to a rotary kiln is uniformity: the rotary kiln ensures that material is constantly agitated and is all subjected to the same retention time and heat transfer. The shaft kiln has a definite thermal gradient with core material reaching a maximum temperature of around 1450°C, some 200° higher than material at the walls. Differential melting tends to increase air flow at the walls, increasing heat loss and exacerbating the temperature differential. More sophisticated systems compensate for this with increased peripheral fuel concentration and reduced wall heat loss. Retention time above 1250°C is typically 30 minutes. Additional air flow increases both production rate and clinker quality. The necessary air injection pressures (1000-2500mmWG) require an efficient air lock on the clinker discharge, preferably a triple gate. In recent years there has been a trend towards increased diameter and reduced height at an aspect ratio of 2.5-3.0, but diameter is usually limited to about 2.4m as any increase makes uniform air distribution. Modern designs can be fully instrumented and PLC controlled.

Fluidised bed kiln

Kawasaki developed a fluidised bed kiln and has reported on the operation of a 200tpd unit (Hashimoto, 1999). This comprises a cyclone preheater, calciner, 1.2m diameter granulation and clinkering fluidised beds and a packed bed cooler, all in the same tower. The granulation bed is controlled at 1300°C and the clinkering bed at 1400°C. Clinker is produced in a fluidised bed under very closely-controlled conditions of temperature and atmosphere appears much more reactive than conventionally produced clinker, with mortar strengths 20 per cent higher. This is probably due to the absence of overburning, as evidenced by much finer alite and belite crystals in the 10-20µm range.

Projections from the pilot plant were reported by Watanabi (2006), proposing:

- 10-15 per cent less heat consumption due to high cooler efficiency and low radiation loss
- over 40 per cent lower NO_x emission
- 25 per cent less CO_2 emission due to reduced fuel consumption
- low capital cost and <30 per cent of the space requirement of a conventional kiln and cooler
- fuel flexibility
- 20-25 per cent lower maintenance and operating costs for the clinker stage
- flexibility for production of different clinker types.

A raw meal blow-in device and special distributor allow self-granulation, with bottom-discharge clinker classified for return of fines to the granulating bed. A 1000tpd unit was constructed at Huling, China, under the auspices of NEDO of Japan. Some data for properties of clinker from this larger plant were reported by Ando et al (2011).

Mini-kilns

Though the scale of kilns is generally increasing, small capacity plants can be justified in some situations. Efficient preheater kilns of 100-300tpd have been designed and constructed by SIME Ingenieros of Colombia (2005).

Table B5.12 Summary of critical data information on different kiln systems

Kiln systems	Kiln speed (rpm)	Loading (tpd/m³)	L/D	SFC (kcal/kg)	SPC (kWh/t)	Residence time (min)	Kiln system exit temp (°C)	ΔP (mm H_2O)	Exit gas (Nm³/kg clinker)
Long wet	1	0.45-0.8	30-35	1300-1650	17-25	180-240	150-180	150-180	3.4
Long dry	1	0.5-0.8	30-35	1100-1300	20-30	180-240	380-400	150-200	1.8
Lepol	1.5	1.5-2.2	12-15	950-1200	20-25	30	100-120	250-400	2.0
Cyclone preheater	2.0	1.5-2.2	14-16	750-900	25	30-40	350	500-700	1.5
AT precalciner	3.6	3.5-5.0	10-14	720-850	25	20-30	300-360	500-700	1.4

Note: Gas volumes are 'actual', not corrected to the reference level used for regulatory emission figures (dry gas at 10 per cent oxygen content). SFC = specific fuel consumption. SPC = specific power consumption.
Source: Manias in Bhatty, 2004

B5.18 Other cooler types

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

Planetary coolers comprise a ring of tubes attached to the kiln shell and turning with the kiln, serving as multiple rotary coolers. These can cause mechanical problems, and tend to complicate flame aerodynamics. In addition, it is difficult to balance clinker flow to the satellites. Internal fittings can wear appreciably and supplementary external cooling is often needed, especially when production rates are pushed. Such coolers are now preferably limited to special applications and their removal and replacement is not uncommon.

Both the above are less effective than grates for cooling clinker, using only the amount of cooling air needed for combustion. However, with efficient internals, clinker can be cooled to about 180°C (Steinbiss, 1992) and they do not require a separate vent or dust collector.

Claudius Peters G-Coolers (supplied by Babcock Materials Handling Division) 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 (where 'G' stands for 'gravity') 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 50mm/minute with heat exchange to air cooled tubes (Harder et al, 1994). They 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, and 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.

B6. Fuels

Data for solid, liquid and gaseous fuels are from various sources including Jenkins, 'The Changing Market of Fuel for Cement Manufacture' (1999). A summary of world fuel reserves is given on the BP website: www.bp.com

B6.1 Typical data for solid fuels

Table B6.1 Typical data for solid fuels (as received)

	Lignite	Sub-bituminous	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 Index	>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 petcoke 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 = H_2O content (%)
  Gross - net = 51.5H*
  where: H* = 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

Table 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	3.3	862	1000-3200	600	15-50
	@38°C	@38°C	@38°C	@100°C	@50°C	@38°C
GCV (kcal/kg)	11,100	10,250	10,250	10,200	7260	
NCV (kcal/kg)	10,390	9670	9670	9610	6740	5-10,000
Solids						100%-200μm
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

Table B6.3 Typical data for gaseous fuels

	North Sea NG	W Aust NG	LPG	Blast furnace 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
CnHm	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 (m/s)	34.0	33.7	39.7	10.5	59.8	23.9	22.3-24.6

**Air required is theoretical volume ratio*

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

B7. Materials

B7.1 Bulk densities of materials for silo storage

Table B7.1 Bulk densities of materials for silo storage

Material	Bulk density (kg/m^3)	Material	Bulk density (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, pulverised	450	Shale/clay	1000
Coke, petroleum	880		

- Angle of repose:

Clinker and dry rock	30-35°
Cement	20°
Gypsum	30°
Bauxite	30°

B7.2 Specific gravities and grindabilities

Table 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	

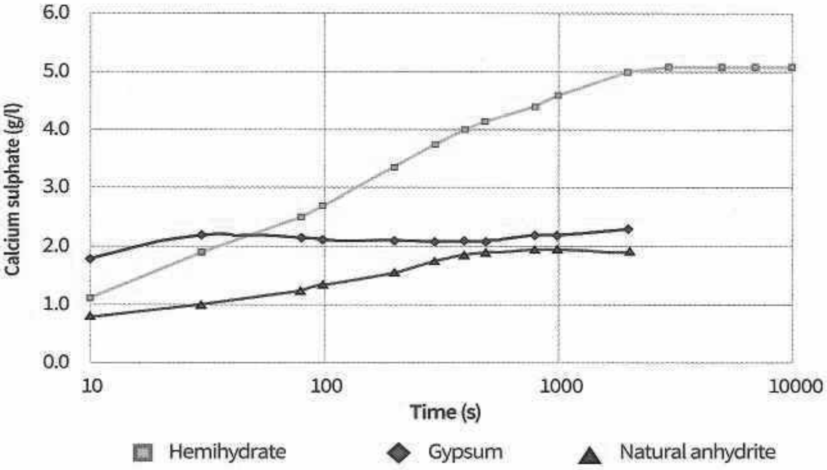
Source: Perry and Green, 1997, p8-13

B7.3 Solubilities of sulphates

Gypsum	$CaSO_4.2H_2O$	24g/l
Hemihydrate	$CaSO_4.0.5H_2O$	ca 6
Soluble anhydrite	$CaSO_4.0.001\text{-}0.5H_2O$	ca 6
Natural anhydrite	$CaSO_4$	21
Syngenite	$K_2Ca(SO_4).2H_2O$	25
Potassium sulphate	K_2SO_4	136
Source: Hansen et al, 1988, p54.		

B7.4 Rates of dissolution of different forms of calcium sulphate

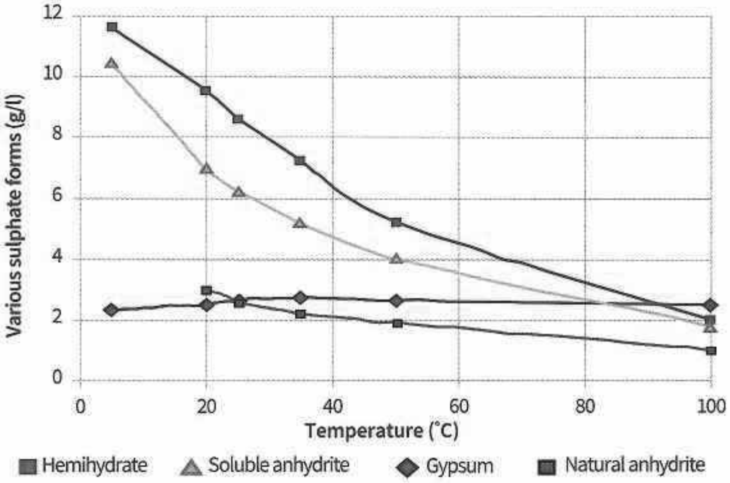
Figure B7.1 Rates of dissolution of different forms of calcium sulphate at 20 ° C



Source: Hansen et al, 1988, p37

B7.5 Influence of temperature on of various sulphate forms

Figure B7.2 Influence of temperature on the solubilities of various sulphate forms



Source: Hansen et al, 1988, p37

B7.6 Chemical formula weights

Table B7.4 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	SO_3	80	SiO_2	60	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	$CaSO_4.1/2H_2O$	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 = 1.175 x mass ratio				SO_3/Na_2O molar ratio = 0.775 x mass ratio			
SO_3 in $CaSO_4.2H_2O$ =	46.5%			SO_3 in $CaSO_4.1/2H_2O$ =			55.2%
SO_3 in $CaSO_4$ =	58.8%			H_2O in $CaSO_4.2H_2O$ =			20.9%
H_2O in $CaSO_4.1/2H_2O$ =	6.2%			H_2O in $Ca(OH)_2$ =			24.0%

B7.7 Coefficients of linear expansion

Aluminium	22.7µm/m/°C
Brick	6.4µm/m/°C
Concrete	14.6µm/m/°C
Copper	16.5µm/m/°C
Steel	11.6µm/m/°C

B8. 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 Quality and 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: $\overline{x} = (x_1 + x_2 + x_3 \ldots + x_n) / n$

Median is the middle value of a distribution.

Mode is the value of a variable that 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:

- $s^2 = \Sigma (x_i - \overline{x})^2 / (n-1)$

  where: x_i = individual test value

  $\overline{x}$ = arithmetic mean of n determinations

Standard deviation, s, is a measure of the dispersion of a set of n observed values expressed as the positive square root of the variance:

- $s = \sqrt{(x_i - \overline{x})^2 / (n-1)}$

Variances may be added. Thus the variance of test results is the sum of the actual sample variance and the test method variance:

- $s^2_{\text{observed}} = s^2_{\text{actual}} + s^2_{\text{test}}$

or, in terms of standard deviation:

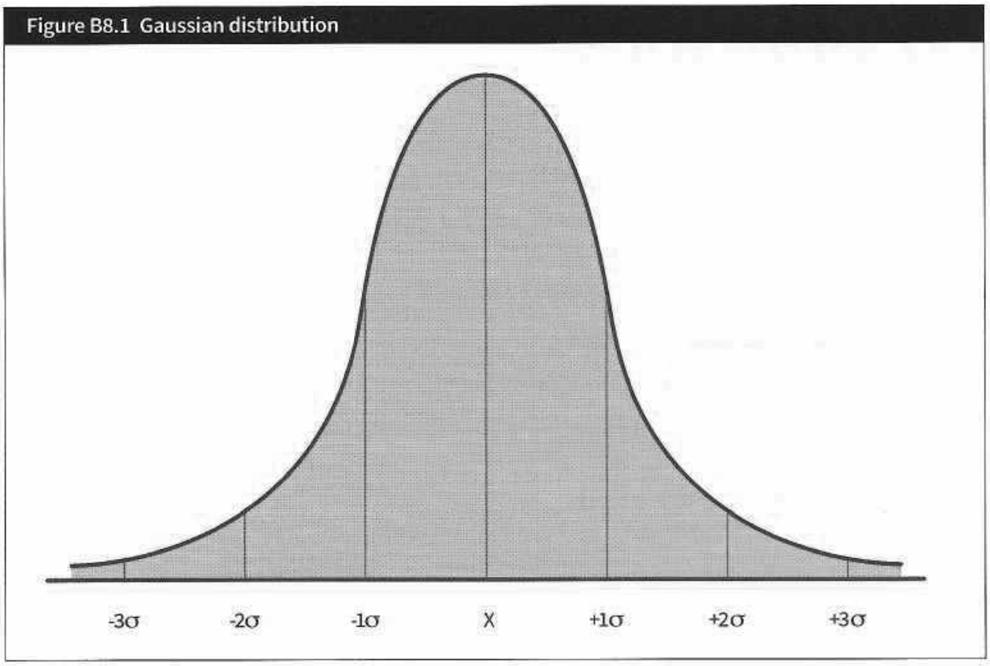
- $s_{\text{actual}} = \sqrt{s^2_{\text{observed}} - s^2_{\text{test}}}$

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 input data are from 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*, $\varphi = n-1$ since two values are required to define a line and only values in excess of two provide statistical information. Where n is finite, only an approximation 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, $\overline{x}$, becomes closer to the population mean, μ.

Random errors give rise to Gaussian (normal) distribution, defined by two parameters, the population mean μ and the standard deviation σ and represented by the familiar bell curve (see Figure B8.1). 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 B8.1 Gaussian distribution



See Table B8.4 for other values.

To convert a sample estimated standard deviation, s, to per cent confidence limits, it is multiplied by an appropriate 't' factor (see Table B8.3). Thus, from a sample n of 11, there is a 99 per cent probability that a single test result will lie within 3.169s of the population mean.

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 the group 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 (see Table B8.1)

Table B8.1 n values and corresponding d values

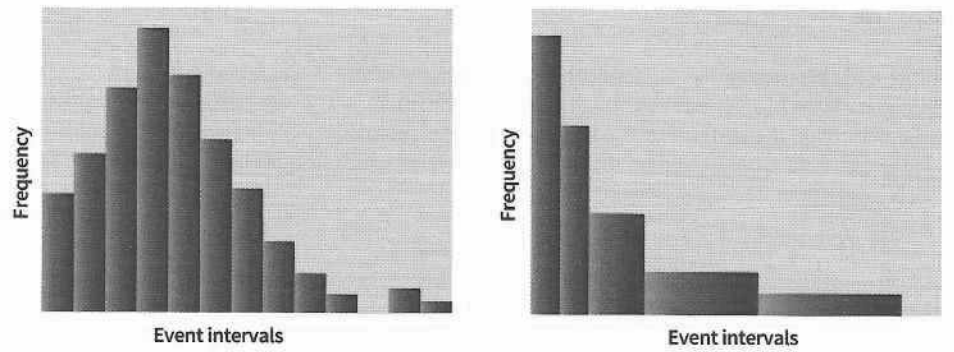
n	2	3	4	5	6	7	8	9	10	16	25	50	100	250
d	1.13	1.69	2.06	2.33	2.53	2.70	2.85	2.97	3.08	3.5	4.0	5.0	5.5	6.0

A **bar chart** is a graphical representation of frequency vs interval using constant intervals.

A **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 B8.2 Bar chart and histogram



Outliers are exceptionally-high or -low values in 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 five-day (n=5) moving average for seven-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.

F test allows estimation of whether two values of s are from the same population. If calculated $F = s_A^2/s_B^2$ (where $s_A > s_B$) is equal to, or less than, the corresponding F value given in Table B8.5 for a specified probability, then identity can be deduced, ie, the groups come from the same population.

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) – see Table B8.2.

Table B8.2 Comparison of observed sub-group values with theoretical probabilities

Sub-group range	Observed value (O)	Theoretical probability (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

Then $X^2 = \Sigma\,(O-E)^2/E$ is compared to tables such as Table B8.6 to determine probable fit for the given degrees of freedom (for six groups, φ =6-1=5). Then X^2 should normally be less than 11 and greater than 1. Greater than 11 indicates poorer than five per cent probability of fit while less than 1 suggests too good a fit for natural data. Other numbers of sub-groups, other probability levels, and distribution other than Gaussian can be used. Comprehensive X^2 tables are then required.

Linear regression is the best fit of xy data to a line:

- $y = mx + c$

  where: $m = s_{xy}/s_x^2$ $c = y - s_{xy}.\overline{x}/s_x^2$

  $s_{xy} = \Sigma(x-\overline{x})(y-\overline{y})/n$

Covariance is a measure of the inter-dependence of two variables; independence gives a correlation (r) of 0, and perfect dependence gives 1 or -1.

- $r = s_{xy}/s_x.s_y$

Multiple-variable regression is applicable to situations where a characteristic (such as cement strength) is related to several controllable parameters (such as C_3S, SO_3, alkali, free lime, fineness, etc). To determine the correlation of each independent variable (x) with the variable of interest (y), a regression analysis is performed on multiple data sets from which weighting constants (a) are derived for each independent variable. Thus:

- $y = a_0 + a_1x_1 + a_2x_2 + \ldots\ldots\ldots + a_nx_n$

Standard computer programs are available (see eg, www.spss.com and www.analyse-it.com) to solve the set of n simultaneous equations for which the minimum number of input data sets should be at least (3n-1). A sanity check should be applied to the overall goodness-of-fit and to individual apparent correlations.

Quality or process control charts are plots against time of any of the following:

- individual measurements, x
- moving averages of subgroups, $\overline{x}$
- moving ranges of consecutive pairs, R
- moving ranges of subgroup averages, $\overline{R}$
- moving standard deviation of subgroups, s

The control chart then displays limit lines at +/-1s, +/-2s, +/-3s about the mean line. The +/-3s lines are known as the upper and lower control levels (UCL, LCL). The purpose of the control chart is to distinguish non-random from random variation and to ensure that corrective action is taken against trending and not against random variation. The significance of various patterns is described in The American Society for Quality Control's 'Tests for Out of Control Patterns on Control Charts' (1984), and by Hoyer & Ellis, 1996. For example, one point outside the upper or lower control limits has a 0.135 per cent probability; two consecutive points differing by more than 4s have a 0.234 per cent probability; six consecutive points on one side of the mean have a 1.563 per cent probability.

Commercial software is also available both to construct control charts and to identify non-random variation patterns.

Table B8.3 Table of selected values of student's 't' distribution

-	80%	90%	95%	98%	99%	99.9%
1	3.078	6.314	12.706	31.821	63.657	636.62
2	1.886	2.920	4.303	6.965	9.925	31.598
4	1.533	2.132	2.776	3.747	4.604	8.610
6	1.440	1.943	2.447	3.143	3.707	5.959
8	1.397	1.860	2.306	2.896	3.355	5.041
10	1.372	1.812	2.228	2.764	3.169	4.587
20	1.325	1.725	2.086	2.528	2.845	3.850
30	1.310	1.697	2.042	2.457	2.750	3.646
120	1.289	1.658	1.980	2.358	2.617	3.373
∞	1.282	1.645	1.960	2.326	2.576	3.291

Most statistical textbooks list t distribution tables. Nowadays, the better way to a fully precise critical t value or a cumulative probability is the statistical function – usually 'TDIST' – implemented in spreadsheets (Office Excel, OpenOffice Calc, etc), or an interactive calculating web page.

Table B8.4 Table of selected cumulative values of the normal distribution – percentage of values within txσ of mean

t	% (one tail)	% (two tails)
0.1	3.98	7.96
0.2	7.93	15.86
0.3	11.79	23.58
0.4	15.54	31.08
0.5	19.15	38.30
0.6	22.58	45.16
0.8	28.81	57.62
1.0	34.13	68.27
1.5	43.32	86.64
2.0	47.73	95.45
4.0	50.00	100.00

Table B8.5 Table of selected values of the 'F' distribution

90% probability

sA/sB	1	2	3	4	6	8	10	24	∞
2	18.5	19.0	19.2	19.2	19.3	19.4	19.4	19.5	19.5
3	10.13	9.55	9.28	9.12	8.94	8.85	8.79	8.64	8.53
4	7.71	6.94	6.59	6.39	6.16	6.04	5.96	5.77	5.63
6	5.99	5.14	4.76	4.53	4.28	4.15	4.06	3.84	3.67
10	4.96	4.10	3.71	3.48	3.22	3.07	2.98	2.74	2.54
40	4.08	3.23	2.84	2.61	2.34	2.18	2.08	1.79	1.51
∞	3.84	3.00	2.60	2.37	2.10	1.94	1.83	1.52	1.00

98% probability

sA/sB	1	2	3	4	6	8	10	24	∞
2	98.5	99.0	99.2	99.2	99.3	99.4	99.4	99.5	99.5
3	34.1	30.8	29.5	28.7	27.9	27.5	27.2	26.6	26.1
4	21.2	18.0	16.7	16.0	15.2	14.8	14.5	13.9	13.5
6	13.74	10.92	9.78	9.14	8.47	8.10	7.87	7.31	6.88
10	10.04	7.56	6.55	5.99	5.39	5.06	4.85	4.33	3.91
40	7.31	5.18	4.31	3.83	3.29	2.99	2.80	2.29	1.80
∞	6.63	4.61	3.78	3.32	2.80	2.51	2.32	1.79	1.00

Table B8.6 Table of selected values of the X^2 distribution

φ	99%	95%	10%	5%	1%
1			2.71	3.84	6.63
2	0.02	0.10	4.61	5.99	9.21
3	0.11	0.35	6.25	7.81	11.34
4	0.30	0.71	7.78	9.49	13.28
6	0.87	1.64	10.64	12.59	16.81
8	1.65	2.73	13.36	15.51	20.09
10	2.56	3.94	15.99	18.31	23.21
20	8.26	10.85	28.41	31.41	37.57
40	22.16	25.51	51.81	55.76	63.69
60	37.48	43.19	74.40	79.08	88.38
100	70.06	77.93	118.50	124.34	135.81

B9. Miscellaneous

B9.1 Atmospheric pressure and density vs altitude (0°C)

Table B9.1 Atmospheric pressure and density vs altitude (0°C)

Alt (m)	p (mmHg)	density (kg/m^3)	Alt (m)	p (mmHg)	density (kg/m^3)	Alt (m)	p (mmHg)	density (kg/m^3)
0	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)

B9.2 pH & normality

Pure water dissociates slightly into hydrogen ions and hydroxyl ions:

$$H_2O = H^+ + OH^-$$

The concentration of each is 10^{-7}/l.

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 per cent.

B9.3 Laboratory reagents (aqueous solutions)

Table B9.2 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

B9.4 Seawater composition

Cl	2.05%
SO_4	0.23%
Na	1.10%
Mg	0.14%
K	0.05%
Ca	0.03%
Total	3.60%

B9.5 Abundance of elements in earth's crust

Table B9.3 Abundance of elements in earth's crust

Element	Atomic no	Wt	Element	Atomic no	Wt
O	8	46.60%	Sr	38	300ppm
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	520 ppm	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

Source: CRC Handbook of Chemistry & Physics

B9.6 Hardness of materials

Mohs scale

Talc	1	Feldspar	6
Gypsum	2	Quartz	7
Calcite	3	Topaz	8
Fluorite	4	Corundum	9
Apatite	5	Diamond	10

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.

B9.7 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

B9.8 Beaufort wind scale

Table B9.4 Beaufort wind scale

Beaufort number	Wind type	Speed (mph)
0	Calm	<1
1	Light air	1-3
2	Light breeze	4-7
3	Gentle breeze	8-12
4	Moderate breeze	13-18
5	Fresh breeze	19-24
6	Strong breeze	25-31
7	Moderate gale	32-38
8	Fresh gale	39-46
9	Strong gale	47-54
10	Whole gale	55-63
11	Storm	64-75
12	Hurricane	>75

B9.9 World cement production

Table B9.5 World cement production and consumption, 2016

Major producers	Annual production (Mt)	Share in world production (%)	Per capita consumption (kg)
1. China	2395	58.0	1378
2. India	288	7	208
3. USA	95	2.3	287
4. Turkey	67	1.6	831
5. Indonesia	62	1.5	242
6. Egypt	59	1.4	602
7. Brazil	57	1.4	319
8. Vietnam	56	1.4	607
9. South Korea	56	1.4	1001
10. Russia	56	1.4	436
11. Saudi Arabia	55	1.3	1922
12. Iran	50	1.2	619
13. Japan	42	1.0	342
14. Mexico	41	1.0	307
15. Pakistan	40	1.0	161
World total	**4129**		
Selected companies	Annual production (Mt)		
Anhui Conch	*277.0*		
LafargeHolcim	*233.2*		
HeidelbergCement	*103.8*		
Cemex	*66.7*		

Source: The Global Cement Report, 12th Edition, 2017

B9.10 Regional cement consumption

Table B9.6 Regional cement consumption, 2016

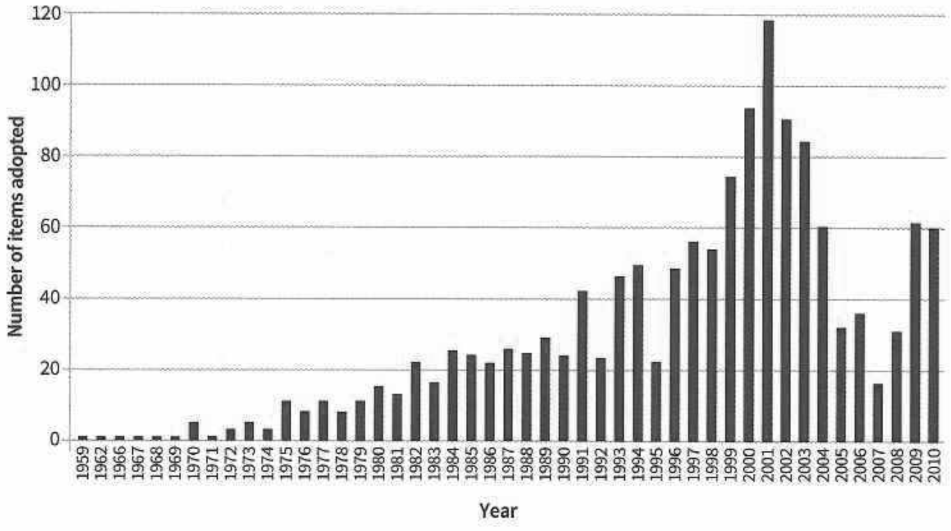
Region	Consumption (Mt)	Share of world consumption (%)
China	2395.07	58
Asia – excl China	330.65	8
Australasia	11.43	0.3
Indian subcontinent	372.04	9
Middle East	179.61	4
Africa	226.11	5
North America	103.00	2
Latin America	175.47	4
Western Europe	126.14	3
Central and Eastern Europe	209.75	5
Total	**4129.27**	**100**

Source: The Global Cement Report, 12th Edition, 2017

B9.11 EU environmental regulation

The EU adopted around 1000 items of environmental law between 1972 and 2001. Thereafter, there was a marked decrease in the annual adoption rate year upon year until 2007, after which there was an increase for a few years until the rate again declined, with concern expressed over the low number during 2012. (Wurzel et al, 2013).

Figure B9.1 Adoption of EU environmental laws, 1972-2009



B9.12 Ship and truck capacities

Table B9.7 Ship capacities

	Deadweight (t)	Length (m)	Beam (m)	Depth (m)
River barge	2000	50	14	4
River barge	5000	85	20	6
Minibulk	<15,000	150	21	9
Small handy	15-25,000	170	24	10
Handy	25-40,000	200	28	11
Handymax	40-55,000	220	32	12
Panamax	55-80,000	250	38	13.5

Dimensions are given as approximations only

Source: Ligthart, 2002

Table B9.8 Truck capacities

Truck type (US)	Capacity (t)	Capacity (st)
Road bulk truck	23	25
Rail bulk car – small	68	75
– large	91	100

B9.13 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 can and have varied with jurisdiction but are increasingly converging. The GATT-approved term is for 20 years from the date of filing. It confers 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-practised 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 practise 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.

Key parts of the patent expenditure in the case of a solely national application are:

- preliminary search (optional)
- patent specification and claim drafting (patent draftman's fee)
- patent drawing preparation fee (patent draftman's fee)
- national application filing (attorney + official fee)
- search (and examination) fee (official fee)
- publication fee (official; sometimes – attorney)
- annual fees starting third year after filing date (attorney + official).

The basic fee is small for a 'do it yourself' application, but businesses use qualified attorneys or agents. In this case, in the USA the cost of the above would be from US$5000-10,000 for average inventions. In UK the price starts at around GBP4000 and goes up to GBP8000 or more. Maintenance fees through the 20-year life can add up to considerable further amounts in each country in which cover is maintained.

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 www.ipo.gov.uk

B10. Conversion tables

Table B10.1 Length

1mm	= 40 thou
25.4mm	= 1 inch
1m	= 3.28ft
1.61km	= 1 mile

Table B10.2 Volume

1m^3	= 35.31ft^3
28.32l	= 1ft^3
3.785l	= 1 gal (US)
1ft^3	= 7.48 gal (US)
4.546l	= 1gal (Imp)

Table B10.3 Pressure

1 atmos	= 1.034 kg/cm^2
	= 14.7psi
	= 1.013bar
	= 101.8kPa
	= 76.0cm Hg
	= 1033cm H_2O (WG)
	= 10.13 N/cm^2
1kPA	= 101.4mm WG
1MPa	= 144.4psi
1 inch WG	= 2.49mbar
	= 250Pa
1N/mm^2	= 145psi

Table B10.4 Weight

1g	= 15.43 grains
31.1g	= 1oz (Troy)
28.35g	= 1oz (Avdp)
1kg	= 2.205lb
1 tonne (t)	= 1.102 short tons (st)
	= 0.984 long tons (lt)

Table B10.5 Area

1m^2	= 10.76ft^2
1 acre	= 4840yd^2
1ha	= 2.47 acres
259ha	= 1 square mile

Table B10.6 Density

1g/cm^3	= 62.4lb/ft^3
1t/m^3	= 262lb/ft^3
1gal (US) water	= 8.345lb
1gmole gas	= 22.4l
1bmole gas	= 359scf
1cm^3 mercury	= 13.6g

Table B10.7 Energy

1kWh	= 860kcal
	= 3413BTU
	= 1.34hp
	= 3.60MJ
1kcal	= 4.187kJ
1kcal	= 3.698BTU
1kcal/kg	= 1.80BTU/lb
	= 3600BTU/st

Table B10.8 Force

1kNewton	= 224.8lbf

Table B10.9 Miscellaneous

2.29mg/Nm^3	= 0.001 grain/scf
1bbl (oil)	= 42.0gal (US)
1t (oil)	= ca 6.5bbl
1bbl (cement)	= 376lb
1m^3/h	= 0.589ft^3/min
1m^3/min	= 35.32ft^3/min

References

Journal abbreviations

CI	*Cement International*	*ICR*	*International Cement Review*	*ZKG*	*Zement-Kalk-Gips*
GCL	*Global Cement & Lime*	*WC*	*World Cement*		

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