University of Nottingham
School of Chemical, Environmental, and Mining Engineering

A Technical Discussion of Mining Operations in the Lime and Cement Industries of Zambia and Malawi

Volume II

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9.1.0 BLAST HOLE DRILLING

9.1.1 History

Together with the development of high explosives, rock drills are the machines that have done most to change the methods of quarrying. Hand chisels came first, these consisted of a team of men with heavy hammers and chisels of various lengths. The chisels were usually of square section, to enable them to be turned easily whilst at the same time being struck by the hammers. These types of shot holes are said to have been in use since 1637. An improvement on men hitting chisels with hammers was developed around 1839 by Isaac Merrit Singer of sewing machine fame, when he produced a mechanical hammer. The first patented rock drill was invented in 1849 by J.J. Couch of Philadelphia. Its drill rod passed through a hollow piston and was thrown like a lance against the rock; caught on the rebound by a gripper, it was again hurled forward by the stroke of the piston.

The first air drill, powered by compressed air, appeared in Germany in 1853. In 1871 Simon Ingersoll patented the first steam powered drill. Just before the turn of the century, Mr J Leyner developed the first of the modern type hammer drills and in 1912 went on to design the hand held "Jackhammer" using hollow drill steels and air to flush the holes. The next significant development came in 1950 when the Halifax Tool Company (Halco) together with others developed the first down the hole hammer. The hydraulic drifter, pioneered by Atlas Copco came onto the market in 1973.

9.2.0 DEFINITIONS

9.2.1 Introduction

It is true to say that; the accuracy of drilling is vitally important for safe and effective blasting, this is especially true if fly rock is to be avoided. Fly rock is usually caused by having too much explosive relative to burden, either at the free face or in the area of stemming, occasionally, fly rock can be caused by geological anomalies such as clay bands or other weaknesses in the face, or cavities which have been overcharged.

In the last ten years the UK legislative bodies have increasingly applied pressure and direct responsibility to the quarry manager his drilling staff and shotfirers to operate under safe blasting practices. This involves producing straight drill holes, accurately locating and logging them both at their collar and at their base and detailing them on a large scale plan. To carry out the above duties requires a certain amount of training and various courses are available. These courses provide certification that the student has reached sufficient understanding of his work to achieve acceptance as a shotfirer and a manager.

Legislation

The legislation found in Zambia is almost completely adapted to underground working in hard rock metallic quarries and is therefore almost totally useless for applying standards to quarries mining, in Malawi the legislation is virtually none existent. In both countries, the authorities are unable to fully understand the principles of quarrying, especially with regard to drilling and blasting. It is likely that the legislation in both countries will be improved and new rules imposed similar to the ones now applied in the United Kingdom. The UK Quarries (Explosives) Regulations 1988 and its Approved Code of Practice on the Quarries (Explosives) Regulations 1988 reflects the general modern approach of most safety legislation in that its philosophy is one of self regulation.

It is the opinion of the writer that the recent change of policy from guidance and mandatory regulations to self regulation is merely a convenience, applied by the legislative bodies to ensure that the producer continually provides detailed information that can be used against him in a court of law. By doing so he relieves the prosecutor of the trouble of gathering evidence against him. The detail that is required to conform to UK legislation would, at this time, not be acceptable in less regulated countries.

The UK Quarries Explosives Regulations 1988 have introduced two requirements new to legislation:-

1. Managers and shotfirers in future must have suitable education, training and experience.

2. Managers must prepare an adequate specification for each shot to be fired.

Regulation 5 (2) (c) requires the manager of the quarry to prepare or approve an adequate specification for each shot to be fired at the quarry to ensure, so far as is reasonably practicable, that when fired, the shot will not give rise to danger.

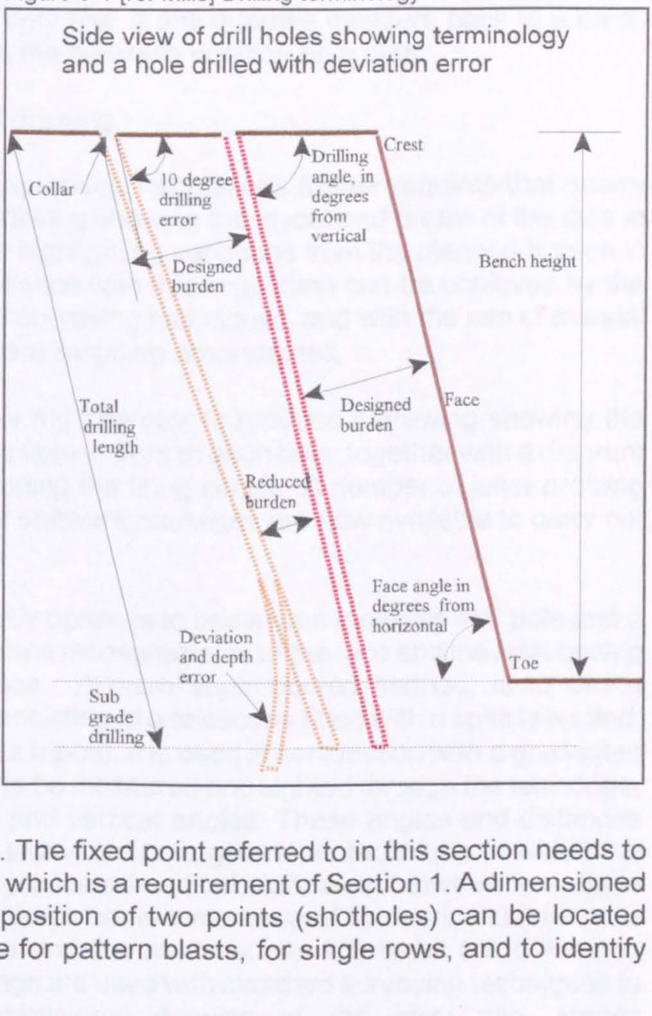
Section 5.15 - 5.19 of the Code of Practice states that an adequate blasting specification for a primary blast will include a detailed survey of the affected area with plans and cross-sections identifying <u>all details of the drilling programme</u>.

Section 1 of the specification suggests that the following information is produced;

- location within the quarry of the blast site
- the surface position of each blast hole relative to a fixed point
- the profile and height of the free face in front of each blast hole,

accurate to 10%
- the planned and actual angle of inclination for each blast hole
- the length, diameter, deviation and azimuth of each blast hole
- the sub drilling both planned and actual of each blast hole
- details of any geological anomalies, ie, bedding planes, clay bands, cavities etc

Figure 9-1 [ref Mills] Drilling terminology



If the working plan also locates the quarry within its own geographical environment, it can also be used to locate sensitive structures beyond the boundaries that could be affected by blasting, this would make environmental surveys easier to carry out. A plan which is more than a sketch ought to be drawn to recognised coordinates, an example is provided in chapter eight, Chilanga Work Study.

Section 2 requires fixed point references. The fixed point referred to in this section needs to be marked on the plan which is a requirement of Section 1. A dimensioned sketch on which the position of two points (shotholes) can be located accurately, will suffice for pattern blasts, for single rows, and to identify back-marks.

Survey and marking out

From fixed survey points "back-mark" lines can be laid out by a surveyor or someone able to comprehend the principles of tape surveys or trigonometry whereby, if one side and two angles of a triangle are known, the other two sides and angle can be readily calculated. From these lines, the proposed position of each hole in the blast can be marked. In most cases these positions will be a specified distance behind the holes in the previous blast at that location. In some cases and in some quarries, it will be obvious that shotholes so placed will have insufficient burden because of back-break, and here the position of each hole will be marked using someone's (perhaps the driller's or the shotfirer's) judgement. The proposed position of each hole should be laid out from the "back-mark" position or checked back to it. In any case the proposed position of each borehole will need to be measured and plotted on a plan or a sketch. Some quarry shotfirers use back-mark techniques, almost always to ensure that the toe burden is controlled by a combination of this technique and accurate drilling. Very few, if any quarries measure back to a back-mark in order to locate the borehole position on a plan.

Profile measuring and drawing

The UK Quarry and Explosives Regulations further requires that quarry operators produce a drawing showing the layout and profile of the face in front of each blast hole highlighting variations from the planned burden in excess of 10%. Compliance with this regulation can be achieved by the application of standard surveying techniques and with the use of manual or computerised software mapping programmes.

The regulations require the operator to produce a drawing showing the profile and height of the face in front of each hole, together with a diagram of the whole blast including the firing circuit. A number of laser profiling systems and computer software packages are now available to carry out this task.

Probably, the least costly option is to use a tape measure and pole and a plumb line to take accurate measurements of the face and the relationship of the holes to the face. A more sophisticated method, is to use a theodolite, (a device consisting of a telescope fitted with a spirit level and, generally, mounted on a tripod). It is used in conjunction with a graduated rod placed at the point to be measured and sighted through the telescope, to measure horizontal and vertical angles. These angles and distances may be measured by tachyometry, a geometric technique in which the vertical distance on a graduated vertical staff, seen between two stadia hairs in the theodolite eyepiece, is a measure of the horizontal distance between the theodolite and the staff usually 100 times the difference between the two readings are used with standard surveying techniques to produce a three dimensional drawing of the blast site. Height determination is achieved by using the theodolite to measure vertical angles and measuring or calculating the distances by triangulation.

Development of the theodolite led to the introduction of the Total Station Concept, this device uses a combination of techniques, these may include Laser, and other optical devices together with a theodolite and electronic distance measuring equipment (EDM). Important technological developments in the 1970s included the use of satellites as reference points (GPS) and inbuilt computers to speed the processing and recording of survey data. The computer records all the observations and calculates the differences obtained by measuring angles. For quarry use, a development of the Total station, is the dedicated quarry survey tool, the unit described is produced by Measurement Devices Limited. These instruments can be operated remotely and feature an integral horizontal and vertical angle measurement capacity which makes it possible to survey dangerous or inaccessible surfaces up to 1,000m away to an accuracy of 1cm.

The auto scanning laser surveying system can also be operated manually using the membrane keypad. However, motorised scanning at user specified increments allows high volume, repetitive surveying such as an entire rock face or building, for example, to be surveyed without operator intervention. The auto scanning laser can observe up to 3,600 points per hour. Hi-low, single and multiple measurement data densities ensure high level definition of openings, cracks and other surface abnormalities.

Measurement Devises Limited produce a borehole deviation survey system, this unit can audit the results of drilling activity quickly and efficiently and is suitable for use to a depth of 100m in dry or flooded holes which measure a minimum of 45mm in diameter. The system uses lightweight alignment rods to prevent the two axis, electrolytic gravity sensor head from rotating, and to lower the sensor head down a hole. This allows depth and time to be logged at regular intervals. Continuous data is time tracked and logged inside the measuring head. The depth data is compiled on the surface electronic notebook to be downloaded into a host PC. An alternative system is produced by the Nitro-Bickford Company, their product incorporates a low-frequency radio transmitter that is positioned close to the toe of the face. The transmissions are received by a probe that is lowered down the drill hole and by calculating the field strength from the device its distance from the transmitter is indicated on a display. An accuracy of better than 95% is suggested.

The information from the mobile storage systems is designed to interface with various software and CAD compatible processing suites such as that produced by C0-ordinated Surveys Systems. These programmes will plot the site and provide a three dimensional display. Used together with instruments to log the path of the drill holes, the programme will provide an image of the whole blast area complete with all the required calculations and data required to complete the blast.

A further device is available to identify cavities and has an integral 360° horizontal and vertical angle measurement capability which enables the operator to survey dangerous or inaccessible underground cavities. The system is suitable for use in dry cavities which measure a minimum of 100mm in diameter and it can reach a maximum two-hundred metres. The device is combined with lightweight carbon fibre alignment rods to prevent the two axis measurement head from rotating within the bore hole. Once deployed via a 100mm borehole, the operator can carry out vertical and horizontal plan section scans at any ARC or CHORD increment.

9.3.0 DRILL TYPES

There are three main types of drills used in quarrying these are the; rotary, down the hole (DTH) and drifter.

9.3.1 Rotary drilling

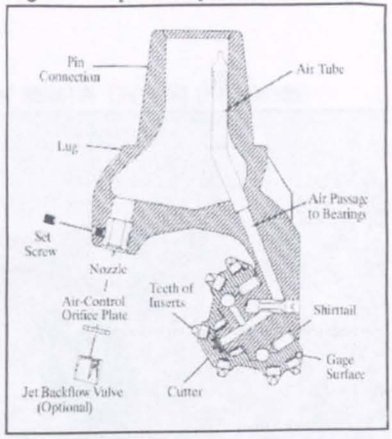
Figure 9-2 [ref Mills] A rotary rig in Zambia.

The first steam powered rotary drill may have been used in England in 1813. The drilling unit consists of a track mounted body fitted with a compressor, mounted on the body is a large tower, on the tower is a drive motor and gearbox. A string of hollow tubes equipped at one end with male threads at the other with female threads, screw onto the gearbox, at the lower end of the drill string is usually fitted a tri-cone drilling bit. The drill bit is threaded to fit to the drill string and is fitted at the lower end with three toothed sprockets or if the rock is hard the teeth are formed from tungsten carbide studs, these studs or teeth break the rock by crushing, chipping, gouging and scraping. Rotary drilling remains the preferred option for deep, large holes having diameters ranging from 150 mm to 450 mm, and where the compressive rock strength is between 1,000 and 5,000 bar. The drilling rigs are usually large, weighing up to 150 tonnes and very expensive. These rigs are often powered by electricity, with either one large alternating current motor driving a hydraulic pump and compressor, or by several direct current

Figure 9-3 [ref Mills] Rotary drills in Zambia

motors each driving a separate operation. With this type of drilling, the energy is transmitted via the drill rod which rotates at the same time as the drill bit is forced down.

Figure 9-4 [ref 118] A tri-cone bit



A pull down force of up to 500 kN and a rotational force of 20,000 Nm of torque of can be achieved by a large rig. With this type of drill, the rock is broken by either compound steel or tungsten-carbide buttons or teeth being forced into the material, the chippings are blown to the surface by compressed air (flushing). Flushing air is carried to the drill bit via the hollow drill string, approximately 10% of the air is directed through the bearings of the bit to clean and cool it, the remainder cleans the bottom of the hole and forces the cuttings to the surface. To be effective, the velocity of the air should be not less than 20 metres per second and not more than 40 metres per second.

Figure 9-5 [ref 118] Required pull down pressures

Pull down required for various drill sizes in a rotary rig	
Hole diameter in mm	Pull down in kg
152 to 200	13,600.00
178 to 230	23,000.00
178 to 270	32,000.00
230 to 311	41,000.00
251 to 450	55,000.00

The relationship between bit pressure and rotation rate can be adjusted to provide optimum penetration, softer rock will require less down pressure with faster revolutions and vice versa, the speed will normally be between 50 and 90 and exceptionally, up to 150 revolutions per minute.

Figure 9-6 [ref 118] Thrust and rpm

Thrust and revolutions required to drill in limestone			
Formation	Type of rock	Revolutions/minute	Thrust / kg/cm^2
soft to medium	soft limestone	75 to 160	100 to 300
medium to hard	hard limestone	60 to 80	200 to 400

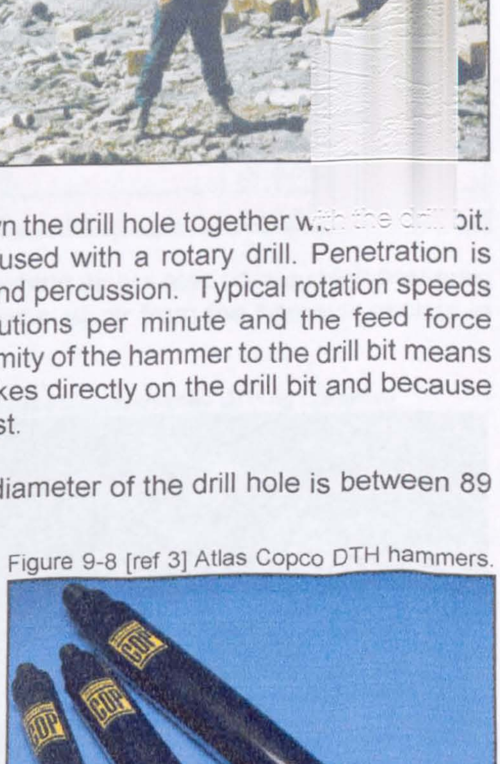
Being more suitable for large scale ore mining operations rotary drills are seldom operated in medium production quarries and because of the relatively small scale of quarrying, rotary rigs are not used in the case studies.

9.3.2 Down The Hole Drill

Figure 9-7 [ref Mills] A DTH rig.(Falklands)

This type of drill usually consists of a track mounted chassis, on which is fitted the hydraulic and pneumatic equipment necessary to operate the drill and a mast. Larger machines usually have their own power plant and, sometimes a compressor is fitted. On the mast is fitted a drive motor and gearbox, similar, if not identical to that found on a rotary rig. In between the drill string and bit is fitted a pneumatic hammer. These rigs are so named because the percussion hammer goes down the drill hole together with the drill bit. Sometimes these hammers are used with a rotary drill. Penetration is achieved by using both rotation and percussion. Typical rotation speeds range between 15 and 40 revolutions per minute and the feed force ranges from 5 to 14 kN. The proximity of the hammer to the drill bit means that the piston of the hammer strikes directly on the drill bit and because of this no percussive energy is lost.

Figure 9-8 [ref 3] Atlas Copco DTH hammers.



In most quarry applications, the diameter of the drill hole is between 89 and 165 mm and a pressure regime of either 10 to 14 bar (low pressure system) or 18 to 25 bar (high pressure system) is employed. When using the high pressure regime, a hammer strike rate of between 1,500 and 1,750 can be expected. In theory, the penetration of these drills is constant regardless of depth, in practice, pressure losses through the joints in the drill string will effect production in deep holes. Because the hammer is in the ground, the noise level is, compared to a drifter, considerably reduced. The drill string is made up of rigid, thin skinned tubes with a choice of diameters, when used with a suitable drill bit size they will have a close fit in the drill hole and holes drilled with a DTH are generally straight with a minimum of deviation. Tungsten/carbide Inserts in the drill tube ream out the hole leaving it smooth and clean and suitable to accept hard packaged explosives.

Figure 9-9 [ref 118 et al] Production guide

Production capacity guide for DTH drilling rigs operating in hard limestone			
Operating pressure bars	Hole diameter mm	Penetration per hour metres	Yield per hour tonnes
6.00	105.00	9.00	360.00
	135.00	10.80	674.00
	165.00	6.00	730.00
10.00	105.00	16.80	672.00
	135.00	19.80	1,240.00
	165.00	16.20	1,984.00
20.00	105.00	39.00	1,560.00
	135.00	40.80	2,550.00
	165.00	42.00	5,144.00

The drill tubes not only carry the flushing air but also convey high pressure air to operate the hammer. The exhaust air from the hammer assists in continuously flushing the drill hole. Due to the small annular space between the drill string and the wall of the drill hole the flushing air retains its high pressure and flow and is therefore effective in clearing the cuttings from the hole. Because the drill string is almost the same size as the drill bit and the hole is reamed clean the risk of losing the hammer and drill string by becoming jammed is reduced when compared to other percussion drills. The penetration rate of a DTH drill will in theory be independent of the hole depth. However, in practice, a drop in penetration rate with increased depth is normal.

Figure 9-10 [ref Mills] Overturned DTH rig (Jamaica)

A DTH drill rig requires a lower volume air of than a drifter but operates better at high pressure. In less developed countries, the use of a high pressure compressor is, for reasons of safety not always considered acceptable, this is because low pressure hoses and accessories are used, thus causing a bursting hazard and perhaps more importantly, (despite strict rules to prohibit it) the use of an air line by the operators to blow dust off their clothing and person. The ability to drill straight deep holes with effective flushing make it possible to achieve excellent results with a DTH drilling rig. Because of this, the two case studies with the highest faces, Ndola Lime Company and Changalume both use these rigs, each fitted with drill tubes of 75 mm and drill bits of 102 mm.

9.3.3 Drifter or top hammer drills

A drifter rig is the cheapest type of rig to purchase and operate but has the least capability. It consists of a light frame, which can be mounted on either a wheeled axle to be towed, or self propelled caterpillar tracks. Designed to pivot on the frame and provide versatility of movement is a mast. A slide is mounted on the mast and on the slide is fitted a motor that provides turning force to the drill, a hammer (either hydraulic or pneumatic) is fitted beneath the motor to percussion impact to the drill bit. Down force is provided by either a motor or a hydraulic ram which is connected to the slide by a link chain. A piston in the hammer hits a shank adapter (an anvil) creating a shock wave which is transmitted at speeds up to 5,000 m/s through the drill string and then to the drill bit. Typically the piston will strike the adapter some 1,500 to 3,500 times per minute with a pressure of between 90 and 150 bar. A high percentage of the percussive energy is lost at the joints in the drill string. This means that the addition of more than a few lengths of drill string will reduce the energy to a point where the drill bit will be unable to penetrate the rock.

Figure 9-11 [ref 118] drifter production

Typical production from a drifter drill in hard limestone			
Hole diameter mm	Drill rod diameter mm	Penetration per hour metres	Yield per hour tonnes
76.00	38.00	20.00	312.00
89.00	38.00	15.00	337.00
102.00	38.00	10.00	400.00

The diameter of an average drifter drill string is usually between 22 and 51 mm with the normal size being 38 mm. These drill strings are fitted to drill bits with diameters ranging from 27 to 127 mm In the case studies the most commonly used size is 76 or 89 mm with occasional use of 102 mm. High pressure flushing air passes down a 14 mm passage in the centre of the drill string to cool the bit and blow rock cuttings to the surface for discharge, a minimum flow velocity of 15 metres per second is required for the drill to operate effectively.

Figure 9-12 [ref 118 et al] Flushing values

Typical values for flushing air			
hole diameter mm	drill rod diameter mm	air velocity m/s	cubic metres required cfm
76.00	38.00	25.00	150.00
76.00	38.00	32.00	230.00
89.00	38.00	17.00	150.00
89.00	38.00	21.00	230.00
102.00	38.00	15.00	230.00

When used with drill bits of over 76 mm the difference in the diameters of the drill string and the drill hole will be excessive and much of the pressure and velocity of the flushing air will be lost. This usually results in either failure of the drill bit by overheating or jamming in the hole due to the accumulation of cuttings. The loss of flushing air places severe limits on the effective depth to which a drifter can operate. Because of the considerable demand for air when flushing the hole, a larger flow compressor is required for a drifter than that required by other types of drills. A further disadvantage of a drifter is a tendency for the drill to deviate in holes deeper than two or three drill string lengths. This is mostly due to the large gap between the drill string and the side of the drill hole together with the flexibility of the drill string, allowing the drill bit to wander. A modern high pressure machine, fitted with a 38 mm drill string and a 76 mm bit will be able to reliably operate to maximum depths of about fifteen to twenty metres and the same drill with an 89 mm bit will be effective to approximately 15 metres.

Figure 9-13 [ref Mills] A drifter rig at Ndola works.

Often, the sides of a drifter drill hole are ripped by the drill string and leave a ragged drill wall, this can cause problems when using cartridged explosives with the packages sticking and becoming jammed in the hole. In soft ground or overburden, the top of the hole will often close in, sometimes resulting in the hole becoming blocked and unuseable. Because of the large difference between the diameter of the drill and the drill string, this can result in the drill being temporarily diverted from its true course by harder rock. Should this happen, the hole will usually become blocked and prevent the drill from being extracted. An advantage of the drifter over a DTH is that, should the drill become jammed, all of the drill string other than the actual drill bit will probably be recovered.

An improvement on the pneumatic top hammer is the hydraulic top hammer. The hydraulic hammer was introduced in the mid seventies and has the following advantages over the pneumatic hammer;

- fuel consumption at 25% less that of a pneumatic drill and 50% less that of a DTH drill
- the ability to vary the characteristics of the drill to suit specific operations
- improved fuel consumption
- smaller rigs and compressors required for a given task
- reduced noise

Possibly because of its complex construction and high cost, the hydraulic top hammer has not proved to be very popular in quarrying operations in Zambia.

9.4.0 GENERIC TERMS AND EXPLANATIONS

Bench drilling

This is a generic term used to describe vertical or angled holes that are drilled in a bench for loading with explosives and blasting towards a free face. The main criteria calls for the consideration of, the diameter of the drill hole, bench height, type and nature of the material, burden, spacing, fragmentation, throw, sub drilling, type of explosive and methods of detonation, and environmental issues. Horizontal holes are generally referred to as snake holes.

Bench height

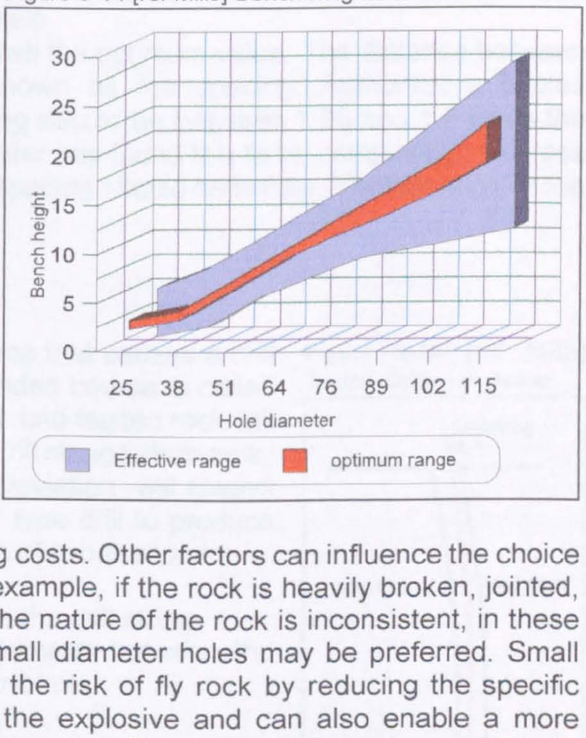
Usually, the bench height will be decided at the quarry planning stage and the dimension retained throughout the complete development of the quarry. Safety will dictate that the height of the benching is a small as possible, however to be cost effective the benches need to be as high as possible, usually a compromise of between five and fifteen metres is acceptable. Legislation in most African countries dictates that for safety reasons the bench height will be governed by the type of machine that is to operate beneath it. For example, the maximum height when using Caterpillar 988B front end loader is about 7 metres, this equates to its maximum bucket height. When using an excavator, a bench height similar to the maximum height of the pivot of the boom and dipper can sometimes be negotiated.

Drill hole diameter

The diameter of the drill hole is the final usable width of the void, in hard rock, this may be very similar to the diameter of the drill bit. In theory, the choice of drill hole diameter is governed by the rate of production from the quarry, in practice the choice is heavily modified by the height of the bench, which is usually governed by legislation.

All commercial explosives and particularly those with relatively low power to mass ratio such as ANFO, will give an increase in power, relative to the diameter of the charge, this increase in the efficiency of the explosive often makes it more cost effective to use larger holes than would otherwise be considered. Assuming a bench heigh of between five and fifteen metres, it is reasonable to assume that the diameter of the drill hole will be between 76 and 102 millimetres. Generally, the greater the diameter of the drill hole, the more cost effective is the blasting, however, bigger holes can require larger drill rigs and stiffer drill strings, with higher associated costs. Conversely, large diameter drill holes can be drilled more accurately than small ones and the drills are less prone to becoming jammed, thus reducing explosive usage and drilling costs. Other factors can influence the choice of drill hole diameter. For example, if the rock is heavily broken, jointed, or in discrete blocks, or if the nature of the rock is inconsistent, in these instances, drilling many small diameter holes may be preferred. Small diameter drill holes lowers the risk of fly rock by reducing the specific charge and the power of the explosive and can also enable a more precise control of the blast direction. Environmentally, assuming that the explosive charges are delayed, it is more acceptable to fire a large number of small holes rather than a small number of large holes.

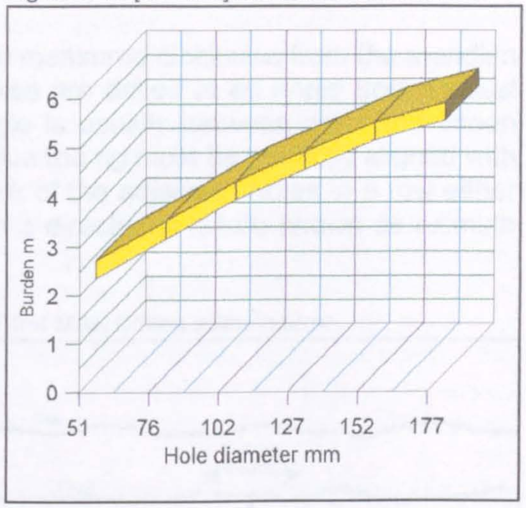
Figure 9-14 [ref Mills] Bench heights to hole diameter



Burden and spacing

Drill holes which are to be used for blasting are drilled at a specific distance from the free face or from each other at right angles to the free face. This distance to the free face dictates the amount of work which the explosive has to carry out, because of this, it is known as the burden. The critical burden has been obtained when the rock is completely fragmented but not displaced. In practice, a burden less than the critical value is used, the amount often depending on the amount of throw that is required and it is normal for the burden to be between 25 and 40 times the diameter of the drill hole. Typically, for a 76 mm drill hole, the burden will be between two and three metres depending on the type of material and the explosive to be used, certainly, field trials must be held to fully establish the optimum value. The distance between the holes in a row is known as the spacing. Authoritative bodies recommend that the spacing should be between 1.25 and 1.5 times the value of the burden, the writer has found this to be completely valueless and recommends that the spacing should be similar, if not identical to the burden.

Figure 9-15 [ref Mills] Burden to hole diameter



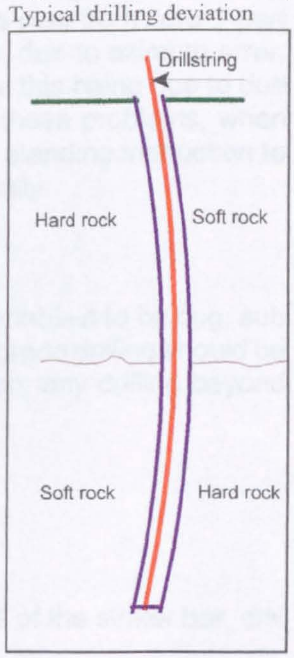
Deviation

The result of any occurrence that causes a drill string to vary from its intended course is called deviation. Broken, fissured and faulted rock will almost certainly cause the drill string to turn away from its intended course. Deviation will always occur when using a drifter type drill to produce deep holes, this is because of two reasons;

- The lack of rigidity in the drill string
- The difference in diameter between the drill string and the drill hole

Poor operation of the rig will, particularly with regard to excessive pull down pressure exasperate the situation. The angle and direction of the deviation can be measured in many ways, ranging from using a torch which is lowered down the hole and a mirror which is used to sight the torch, to sophisticated in hole electronic probes. To reduce the chances of overcharging, it is of utmost importance to relate the position of the hole both to the face and to any other holes.

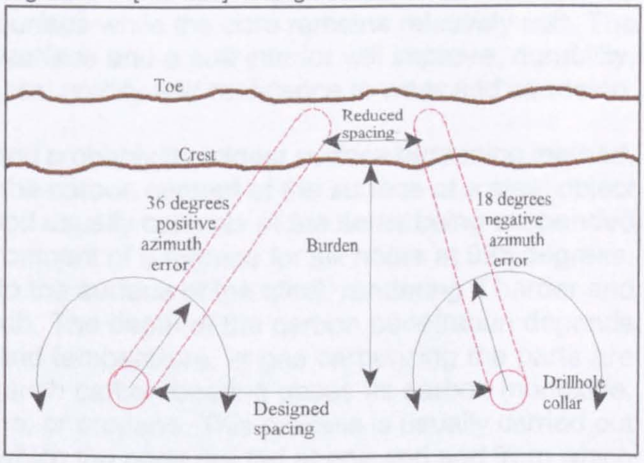
Figure 9-16 [ref Mills] Typical drilling deviation



Azimuth or bearing

Is the horizontal bearing of a line measured clockwise from the meridian through 360 degrees. Often holes are drilled at an angle from vertical towards the free face. The angle is usually between zero and fifteen degrees. When drilling angle holes the rig must be perfectly aligned with the face, this is to reduce the risk of the adjacent holes in a row either converging or diverging through a directional failure known as azimuth error.

Figure 9-17 [ref Mills] Drilling azimuth error



When the drilling rig is perfectly aligned perpendicular to the face, the azimuth value is 0/360. Either the angle can be measured to the full 360 degrees in a clockwise manner such as when using a compass, or the angle can be measured from the meridian as plus or minus values. For example, an angle towards the left when looking over the face would be degrees minus from zero and an angle towards the right being degrees plus from zero. Apart from being less effective, a blast where the holes, due to azimuth error, either converge or diverge can be very dangerous, this being due to due to the possibility of local overcharging. To avoid these problems, when using unskilled operators, the writer often gives a standing instruction to the drillers that all the blast holes are drilled vertically.

Sub grade drilling

To break the rock at the bottom of the bench and enable it to be dug, sub grade drilling and blasting is often carried out. Sub grade drilling should be between 25% and 30% of the value of the burden, any drilling beyond 33% will have little, if any effect.

9.5.0 DRILLING COSTS

9.5.1 Unit description and design life

The drilling string in a top hammer system consists of the striker bar, drill rods, couplings and the drill bit.

The striker bar, or shank adaptor is located in the bottom of the hammer, its purpose is to transmit the shock energy and rotation via a coupling to the drill rods, the bottom of the shank adapter is threaded to accept the coupling. To prevent premature failure, a high quality wear resistant steel is required, usually this is a low carbon-chromium or nickel-chromium alloy. Often, the threads simply wear away and fail to rotate the drill rod.

Couplings and rods fail either by breaking or the threads wearing out. These wear items are rolled from low to medium carbon steels, containing 0,2% to 0.27% carbon, 2 to 3% chromium/nickel and small amounts of manganese and molybdenum. The items are treated to increase the hardness of the outer surface while the core remains relatively soft. The combination of a hard surface and a soft interior will improve, durability, fatigue strength, hardness, rigidity and resistance to wear and corrosion.

One type of treatment and probably the oldest surface hardening method, is carburizing in which the carbon content of the surface of a steel object is increased. The method usually consists of the items being suspended in the carbon rich environment of a furnace for six hours at 925 degrees. The carbon diffuses into the surface of the steel, rendering it harder and causing a volume growth. The depth of the carbon penetration depends on the exposure time and temperature. In gas carburizing the parts are heated in contact with such carbon bearing gases as carbon monoxide, carbon dioxide, methane, or propane. This process is usually carried out in a large furnace into which the parts are fed at one end and from which they emerge at the other end in the hardened state. The same process is used in carbon/nitriding except that ammonia is added to the furnace atmosphere and it takes place at lower temperatures that produce less distortion in the steel. Where there is a risk of bending, carburizing may give rise to premature failure through cracking and fracture.

A second type of treatment is by the use of high frequency induction or flame hardening, in which high heat is applied for a short time, heating the material to 900 degrees followed by immediate quenching. This treatment induces compressive stresses into the surface of the material, leaving the core of the object unchanged. This treatment of steel produces a tough product, more able to withstand bending and impact damage.

The external surfaces may also be treated with a phosphatic compound to protect against corrosion.

The cutting surfaces of the drill bit consist of a combination of tungsten carbide and elementary cobalt. Tungsten carbide is a dense, metallic like substance, light gray with a bluish tinge, that at 2,600 C decomposes, rather than melts. It is prepared by heating powdered tungsten with carbon black in the presence of hydrogen at between 1,400 and 1,600 C . For fabrication, a process developed in the 1920s is employed, whereby, the powdered tungsten carbide is mixed with another powdered metal, usually cobalt, and pressed into the desired shape, then heated to temperatures of between 1,400 and 1,600 C; the other metal, which melts, wets and partially dissolves the grains of tungsten carbide, thus acting as a binder or cement. The greater the tungsten carbide to cobalt ratio in the inserts, the harder is the cutting surface. The grain sizes can range from 1 to 10 micron, fine grain sizes will increase the strength but reduce the hardness and resistance to abrasion. Care should be taken when choosing a type of drill bit, generally, a harder drill insert will reduce wear but may break and ultimately it may be more cost effective to use a softer insert.

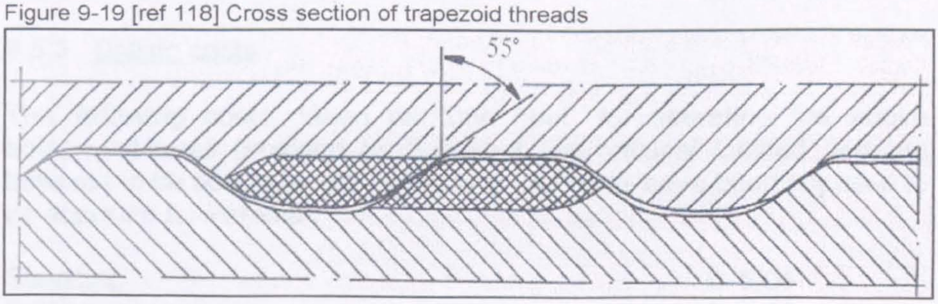
Drill bits are available in a variety of shapes, but they can be described as either cross, (X) or button. Cross bits are available up to 35 mm and the inserts are fitted at 75 and 105 degrees to each other. X-bits are available from 64 to 127 mm. Neither of these bit types are efficient beyond 64 mm, X bits drill faster than cross bits and leave a rounder straighter hole. Button bits are available from 51 to 251 mm, they have moderate drilling rate and good wear characteristics. The high number of cutting surfaces gives a more even cutting rate than a cross bit, resulting in reduced drill string wear. The table below gives average drill bit life, this is often determined by the percentage of quartz in the rock.

Figure 9-18 [ref 118 et al] Drill bit life

Drill bit life in metres drilled				
Bit diameter	limestone	granite	gneiss	quartzite
76 mm	1,500 to 1,800 m	1,100 to 1,400 m	800 to 1,000 m	600 to 900 m
89 mm	2,000 to 2,300 m	1,400 to 1,700 m	1,100 to 1,400 m	700 to 1,000 m
102 mm	2,500 to 2,800 m	1,700 to 2,000 m	1,400 to 1,700 m	900 to 1,200 m

The T38 steels (rods) that all the quarries in the case studies use should have an estimated life of 2,200 percussion minutes. The "T" in the

Figure 9-19 [ref 118] Cross section of trapezoid threads



description indicates trapezoidal thread, this has a larger slope angle than the rope thread and a pitch that increases with the diameter. T thread drills are easier to work with than other threads. The figure, denotes the diameter or the shank. The writer is aware that the actual life of the drill steels in the case studies is much less that suggested above, this is mostly due to operator misuse. For example, often the drillers want to reduce the drilling speed and therefore the amount of heavy manual work handling the drill rods, to achieve this, they reduce the force of the pull down to a minimum, the force of the percussion will then soon cause the rods to work harden, become brittle and snap. Proper supervision of the drillers can overcome these problems, especially when the cost of one drill rod is greater than six months wages for the driller. Other problems commonly found are the lack of lubrication, normally caused by using incorrect lubrication, for example, general purpose grease, and striking the rods, this leads to the formation of stress risers and subsequent failure from fracture.

9.5.2 Equipment costs

The following is a typical price for a small drilling rig and compressor (1999), this type of rig can be used either as a drifter or as a DTH. All of the quarries featured in the case studies use this type of rig. The following prices were provided by Sureshot International Limited ;

Ingersoll Rand CM 345 DTH Drilling Rig
Ingersoll Rand 750 SCAT Compressor

1.	1No Ingersoll Rand CM 345 DTH Drilling Rig	£ 45,300.00
2.	2No QL4 DTH hammer @ £1,400.00 each	£ 02,800.00
3.	6No 4.1/8Q4VB Ballistic Bits @ £ 255.00 each	£ 01,530.00
4.	17No 3" x 3M Drill Tubes @ £ 112.50 each	£ 01,912.50
5.	1No 2" x 100' Bull Hose	£ 01,190.00
6.	1No Lubricant	£ 00,120.00
7.	1no Rod Rack	£ 03,650.00
8.	Dust Collection Unit (fitted)	£ 04,300.00
9.	1No Ingersoll Rand VHP 750 Compressor	£ 32,600.00
	TOTAL	£ 93,402.50

9.5.3 Drilling costs

The following costs, taken per unit hour, for operating the above equipment were provided by Sureshot International Limited and are believed to be accurate, the labour and fuel costs have been adjusted to be accurate for Zambia ;

Servicing	£ 5.00
Drilling accessories	£ 0.50
Hammer costs (two units per year)	£ 1.20
Drill bits (six units per year)	£ 0.80
TOTAL	£ 7.50

Doubled to represent costs in Zambia	£15.00
Fuel at seven gallons per hour	£20.00
Labour two men at £0.50 each doubled to include employment costs	£ 1.00
Total drilling costs	£36.00
At a production rate of 400 tonnes per hour the cost of drilling is	£00.09 per tonne.

Depreciation is a cost that can be calculated at the rate used by the producer, usually this is on a straight line depreciation value.

Chapter Ten

Explosives Systems

10.1.0 EXPLOSIVES

10.1.1 Disclaimer, **please read**!

The information, properties and characteristics given in this chapter and the methods discussed are the product of the writers own research and experience and are believed to be accurate, but readers should make their own tests to determine the suitability of such products and applications of such methods for their own particular purpose, do not experiment with explosives, they can hurt. Explosive materials are dangerous and must be handled with proper procedures. The writer makes no express warranties regarding the use of the information, methods or products described in this document. Training in the use of explosives is recommended.

10.1.2 Low energy explosives

Definition;

A low explosive or deflagrating (burning rapidly) explosive is one that can react in the absence of air and in normal circumstances will deflagrate (burn rapidly) and not detonate. Confining a low explosive will increase the rate of burning to a higher value probably in the order of 300 to 400 metres per second but generally not faster than 1,000 metres per second. Under certain conditions, such as the use of large quantities and a high degree of confinement, some normally deflagrating explosives can be caused to detonate.

History

The first low explosive to be discovered was gunpowder, this being a mixture of 75% potassium nitrate 15% charcoal, and 10% sulphur or brimstone (brimstone, meaning burning stone). Potassium nitrate has a formula of KNO_3 and is commonly known as Bengal saltpetre (saltpetre, from the Latin *sal petae* or salt of stone, because of its encrustation on stone walls), all nitrates decompose when heated and may do so explosively. Sulphur is unstable and ignites at a low temperature. Charcoal is a carbon and source of fuel. For example, when potassium nitrate (KNO_3) is heated, a nitrite (a compound containing NO_2-) is formed and oxygen gas is evolved. When well ground and mixed, these provide all the ingredients required to produce a low energy explosive mixture and because the burning of black powder is a surface phenomenon, a fine granulation burns faster than a coarse one. Gunpowder is believed to have been invented in China and is said to have been used over 3,000 years ago. A version of gunpowder was used in about 668 AD by the Byzantine fleet against the Arabs, it was known as "Greek fire" and probably consisted of naphtha (As early as the 1st century AD, the name originally applied to the more volatile kinds of petroleum issuing from the ground in the Baku district of Azerbaijan and Iran), quicklime (calcium oxide) and sulphur, at that time saltpetre was not generally known. The Arabs learned of saltpetre in 1250 and by 1304 they had produced the first known working gun, this being a bamboo tube reinforced with iron, which used a charge of black powder to fire an arrow. Roger Bacon was an English philosopher and scientist who was born (1214) in Ilchester, Somerset and educated at Oxford and Paris, he died (1292) in Oxford. He seems to be the first man in the England to be familiar with black powder, Bacon read Arabic, and it is possible that he obtained his knowledge from Arabic sources. He described it in a famous cypher, written in Latin as an anagram, thus;

Sed tamen salis petae LURU VOPO CAN UTRIET sulphuris, et sic facies tonitruum et coruscationem : sic facies artificium.

The passage is thought to be interpreted as;

Sed tamen salis petae R VII PART V NOV CORUL V ET sulphuris, et sic facies tonitruum et coruscationem: sic facies artificium.

Which translates as; *"But, however, of saltpetre take six parts, five of young willow, and five of sulphur, and you will make thunder and lightening and so you will turn the trick"*. Guns came into use in Europe shortly after the death of Bacon, evidently his anagram was too easy to decipher. Berthold Schwartz, a Monk living in the 13th century is said to have further developed gunpowder and the gun. In 1544 a Franciscan Friar, Sebastian Munster elaborated on the work of Schwartz and in 1600 The French physicist, Francois Thybourel produced an epitaph for Schwartz (German language for black) naming him "Berthold the Black", hence the term "Black powder". [ref 5, 19 et al]

Gunpowder was used in warfare from the 14th century, usually as a propellant for rockets, but it was not generally adapted to civil purposes until the 17th century, when it began to be used in mining. The Hungarians in 1627 used the explosive at the Royal Mines of Schemnitz at Ober-Biberstollen (now Banská Stiavnica, Czechoslovakia), where they were the first to use black powder to replace the ancient practice of fire setting and quenching which was until that time (still used in some parts of Africa) the standard method used in mining to loosen and break rocks. Because there were no rock drills at that time, a favourite method in use was to dig a passage parallel to the face of the hill, fill it with black powder, close the entrance and fire it. This style of rock breaking is known as heading blasting, in this century it has been used by the Chinese to demolish complete hills in one go. The first application of this method in civil engineering was in the Malpas Tunnel of the Canal du Midi in France in 1679. [ref 5, 19 et al]

Excluding quarrying for block stone, the only place in which gunpowder is still used in mining is in safety fuses and detonators. In 1831, William Bickford, (1774-1834), born in Devonshire, England developed the miner's safety fuse. Prior to this invention, gunpowder was put into a borehole and lit by a fuse that was generally one of a succession of crude means that included straws filled with pulverized black powder, reeds in which the pith was scooped out and replaced with a paste of gunpowder and water (later bound with string and dried), or gunpowder paste spread on wool threads. All of these fuses were ignited either by a piece of wool yarn impregnated with sulphur, called a sulphur mannikin, or some equivalent slow-burning device. A later, and extremely popular type of fuse was formed from goose quills, that were cut so that they could be inserted one into the other and then filled with powder. Quill fuses could be ignited directly, that is, without any delaying element such as the sulphur mannikin. Quill fuses frequently seemed to fail but then rekindled so that the miner who went to inspect the apparently dead fuse sometimes also became extinct or was injured in the blast. Bickford developed a safety fuse that virtually eliminated this danger, it consisted of a continuous core of gunpowder contained within a tube of jute and string. The present-day version is not very different from the original model except that cord is coated with a waterproofing agent, such as asphalt, and is covered with either textile or plastic. The safety fuse provided a dependable means for conveying flame to the charge. Its timing was accurate and consistent, compared to that of its predecessors, and it was much better from the standpoints of resistance to water and abuse. [ref 5, 19 et al]

10.1.3 High energy explosives

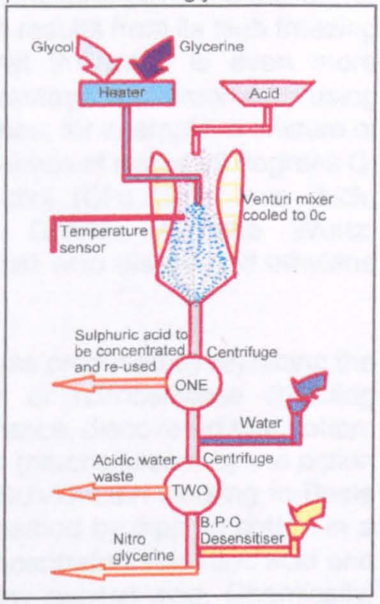
Definition;

A high energy explosive will detonate either confined or not. The progressive acceleration of reaction is accounted for by growth of a flame front area. A transition from laminar to turbulent flow will give rise to a shock wave travelling at a velocity exceeding 1,000 metres per second (MPS). An increase in temperature due to compression in the shock wave results in self-ignition of the mixture and detonation rapidly becomes continuous.

Chemistry

The first high explosive in common use was nitrated glycerine or nitroglycerine, glycerine glycerol or propan-1,2,3-triol $HOCH_2\,CH(OH)CH_2\,OH$ is a colourless, thick, odourless, sweetish liquid that was discovered by the Swedish chemist Carl Wilhelm Scheele. It is obtained from vegetable and animal oils and fats (by treatment with acid, alkali, superheated steam, or an enzyme), or by the fermentation of glucose. In 1855, a British inventor, George Fergusson Wilson, (1822-1902) discovered a process of manufacturing pure glycerine, the purity is of paramount importance when nitrating it into an explosive as impurities will cause it to become unstable. The nitration of glycerine was the result of several pioneering chemists, the final step, taken in 1846 was performed at the University of Torino Italy by the Professor of applied chemistry, Asconio Sobrero (1812-1888) when he nitrated purified glycerine with a mixture of 95% nitric and 4% sulphuric acid to form nitroglycerine ($C_3H_5(ONO_2))_3$ he named the oily liquid piroglicerina. The chemical name for nitroglycerine is glycerine trinitrate. Simple nitration by using nitric acid will not work as it forms water which will convert the trinitrate back to acid and glycerine, the small percentage of sulphuric acid acts as a dehydrating agent and drives off the water. Nitroglycerine is a colourless oily liquid with a strong smell of almonds or marzipan, it is very sensitive to impact and friction. Nobel named the explosive, Glonion Oil, it has a high nitrogen content of 18.5 percent and contains sufficient oxygen atoms to oxidize the carbon and hydrogen atoms while nitrogen is being liberated so that it is one of the most powerful explosives known. Detonation of nitroglycerin generates gases that occupy more than 1,200 times the original volume. For a long time, nitroglycerine was the most used sensitizer of commercial explosives. However, it becomes unstable at 80.0 degrees C and ignites at 180.0 degrees C, when it burns quietly with a green flame. At 218.0 degrees centigrade and when heated rapidly it explodes and produces a temperature of about 5,000 degree C. The overall effect is the instantaneous development of a pressure of 20,000 atmospheres; the resulting detonation wave moves at approximately 7,700 m per second. The manufacturing process was developed to a commercial scale in 1862 by Alfred Nobel. In Zambia the Schmid continuous process was until recently being used, because it only processes small amounts at any one time this process was considered the safest and most effective.

Figure 10-1 [ref 84] Schmid process for nitroglycerine.



In 1867 Alfred Nobel, after much experimentation and the deaths of five men including his younger brother Emil, achieved the stabilisation of nitroglycerine by mixing it at a ratio of 75% nitroglycerin to 25% kieselghur (a chalk like soil) with an absorbent diatomaceous earth known as guhr or Fullers earth to produced an explosive which he named dynamite, from the Greek dynamis, meaning power. Kieselghur will absorb three times its own weight of nitroglycerine. Typical velocities for ghur dynamite are 6650 to 6800 metres per second.

Later, wood pulp was substituted as the absorbent, and sodium nitrate was added as an oxidizing agent to increase the strength of the explosive. A serious problem in the use of nitroglycerin results from its high freezing point of 13 degrees C and the fact that the solid is even more shock-sensitive than the liquid. This disadvantage is overcome by using mixtures of nitroglycerin with other polynitrates; for example, a mixture of nitroglycerin and ethylene glycol-dinitrate freezes at minus 29 degrees C. Glycol or ethylene glycol or ethane-1,2-diol $(CH_2OH)_2$ is a thick, colourless, odourless, sweetish liquid. Charles Adolphe Wurtz, (1817-1884) was the French organic chemist who discovered ethylene glycol (1,2-ethanediol) in 1856.

A more powerful explosive than dynamite was produced by replacing the kieselguhr with nitrated colloidal cotton or nitrocellulose (blasting gelignite). In 1838 T. J. Pelouze in Paris France, discovered that cotton, a cellulose could be made into an explosive (nitrocellulose) by the action of concentrated nitric acid. In 1845, C.F. Schöönbein working in Basle Switzerland improved the manufacturing method by dipping cotton in a mixture of nitric acid in the presence of concentrated sulphuric acid and then washing the substance to remove any excess acid. Chemically, cellulose is a carbohydrate composed of long chains of glucose units, these found as the principal constituent of the cell wall of all plants, it is the most abundant substance found in the plant kingdom. Walter Norman Haworth(1883-1950) an English organic chemist from Chorley, Lancashire studied at Manchester University and with his team, established the linkages in the ring structure of carbon atoms in hexoses, establishing the chain structures of cellulose and starch. Cellulose nitrate (gun cotton) or nitrocellulose are a series of esters of cellulose with up to three nitrate (NO_3) groups per monosaccharide unit. The greater the nitrogen content the more unstable is the cellulose with a practical maximum of 13.4%, at this ratio it is classed as a fully nitrated high explosive. Nitrocellulose is a fluffy white substance that retains some of the fibrous structure of untreated cellulose. It is not stable to heat, and even carefully prepared samples will ignite on brief heating to more than about 150C. When nitrocellulose decomposes, it forms products that catalyse further decomposition; this reaction, if not stopped in time, results in an explosion. When in 1875 Nobel replaced the kieselghur in dynamite with nitro-cellulose he developed the originator to an extensive family of gelignite explosives. This event effectively closed a chapter on the development of commercial explosives, for it is only in recent years that any further developments were made. Modern gelignites (where available) contain from 20 to 60% mixed nitric esters absorbed in wood pulp and nitrocellulose together with a measured amount of sodium and potassium nitrates sufficient to maintain the oxygen balance, the permutations all revolve around the central theme of;

- nitroglycerine
- nitro glycol

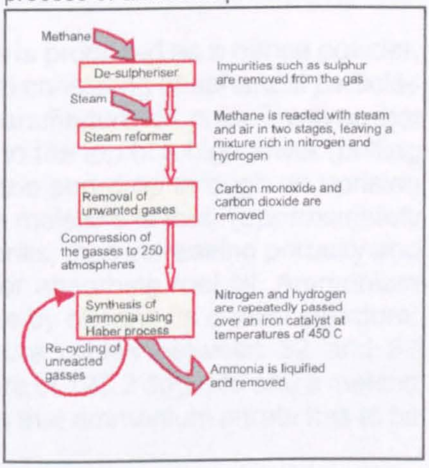
- nitrocellulose
- oxidising salts
- fuels

Oxygen balance

Modern explosives are designed to be safe in use, non toxic, have good shattering characteristics (brisance), high gas production for rock heave and have a good oxygen balance to reduce fumes. An oxygen balance that is too high leads to nitric oxide fumes and nitrogen dioxide, while a balance that is too low leads to excessive amounts of carbon monoxide. In 1863 TNT trinitrotoluene ($C_7H_5N_3O_6$) a pale yellow, solid organic nitrogen compound with a velocity of detonation of 6,900 metres per second was discovered by Wilbrand and is an example of an explosive with a poor oxygen balance.

10.1.4 Ammonium nitrate

Figure 10-2 [ref 84] Haber Bosch process of ammonia production



In 1774, Joseph Priestley was the first to produce pure ammonia and its exact composition was determined by Claude-Louis Berthollet in 1785. Ammonia consists of a mixture of nitrogen and hydrogen, to make the two gases react a temperature of 450 degrees centigrade and a very high pressure of 250 bars is applied, this, over an iron catalyst, the result is condensed as liquid ammonia. The most common method of directly synthesizing ammonia from hydrogen and nitrogen, was developed by the German physical chemist Fritz Haber. He received the Nobel Prize for Chemistry in 1918 for this method, which made the manufacture of ammonia economically feasible. The method was translated into a large-scale process using a catalyst and high-pressure methods by Carl Bosch, an industrial chemist who won a Nobel Prize in 1931 jointly with Friedrich Bergius for high-pressure studies.

Commercially, nitric acid is produced by the Ostwald process, this was developed by Friedrich Wilhelm Ostwald a Nobel Prize winning Russian-German chemist and patented in 1902. This process involves the repeated oxidation of ammonia over a platinum catalyst to make nitrogen dioxide gas NH3. The gas is then reacted with water to form nitric oxide, NO, further oxidation of the NO to nitrogen dioxide, NO_2, and then conversion of the NO_2 to nitric acid (HNO_3). When pure, nitric acid is a colourless liquid that boils at 86 degrees C and freezes at -42 degree C.

Ammonium nitrate ($NH_4 NO_3$) is a salt of the neutralisation of nitric acid with ammonia. In its pure form, ammonium nitrate consists of 60% oxygen by weight, 33% nitrogen and 7% hydrogen. To make ammonium nitrate, the vaporised ammonia and nitric acid are mixed in a neutraliser vessel, where a reaction takes place and the nitrate is formed. It is highly soluble in water; heating of the water solution decomposes the salt to nitrous oxide (laughing gas). Upon being exposed to light or heat, it decomposes to produce oxygen, water, and a mixture of nitrogen oxides these being primarily NO_2 ($4HNO_3$ + light (or heat) $4NO_2 + 2H_2O + O_2$).

Ammonium nitrate (AN) is currently the cheapest source of oxygen in explosives, it has been in use since 1867 when Norrbin and Ohlsson patented its use with nitroglycerin. It was not until after the second world war that the discovery was made that mixing carbon products with ammonium nitrate produced a viable explosive. This was proven in Texas City USA, when in 1944 an explosion took place aboard the French freighter *Grandcamp,* it was loaded with contaminated ammonium nitrate fertilizer, that explosion killed nearly 600 people. A further explosion took place in the docks at Brest, France.

For use as a fertiliser, ammonium nitrate is produced as a dense powder, for use as an explosive a porous material consisting of spherical particles with voids is produced (prills). The manufacture of prills involves hot ammonium nitrate liquor being pumped to the top of a high tower (prilling tower) to be forced through a spray nozzle and drop through an uprising air current. When dropping, most of the moisture is lost, (approximately 4%), this causes voids to remain in the prills, thus increasing porosity and surface area, making it more suitable for absorbing fuel oil. Ammonium nitrate responds to temperature changes by altering its crystal structure. Phase changes occur when temperature cycles between 32 and 83 degrees F, a crystal transition temperature of 125.2 degrees and a melting point of 169.6 degrees centigrade means that ammonium nitrate has to be stored in cool stable temperatures.

10.1.5 ANFO

ANFO is a mixture of ammonium nitrate and fuel oil. Currently ANFO it is the main column charge used at all of the quarries described in this document. By far the most common mixture is where 5.6% of fuel oil is added to the ammonium nitrate to produce the stoichiometric mix where all the available oxygen is

Figure 10-3 [ref Mills] ANFO fuel/strength ratio

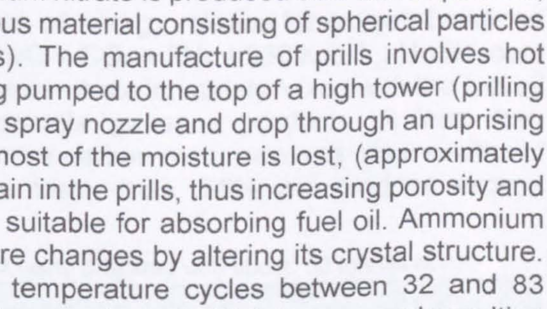
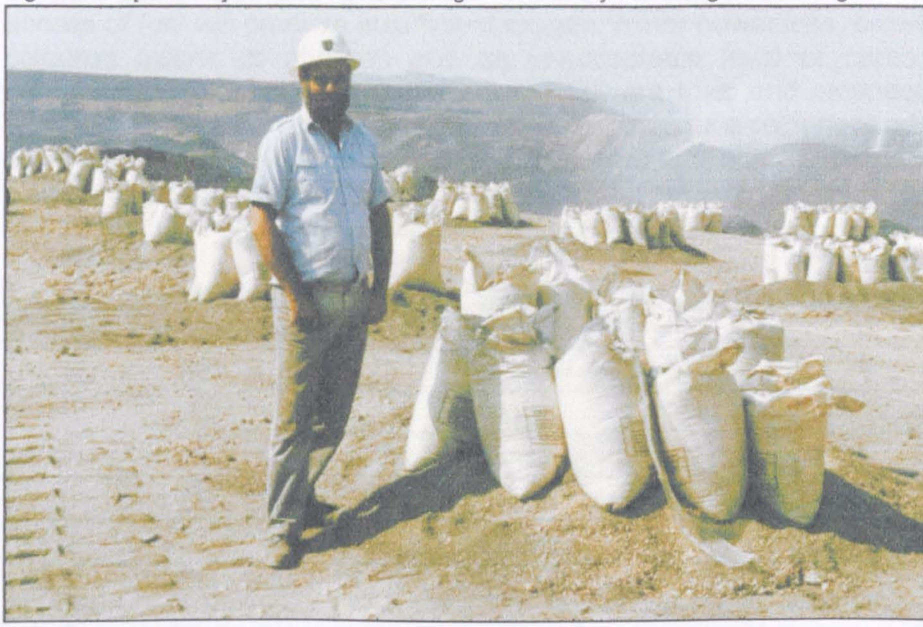
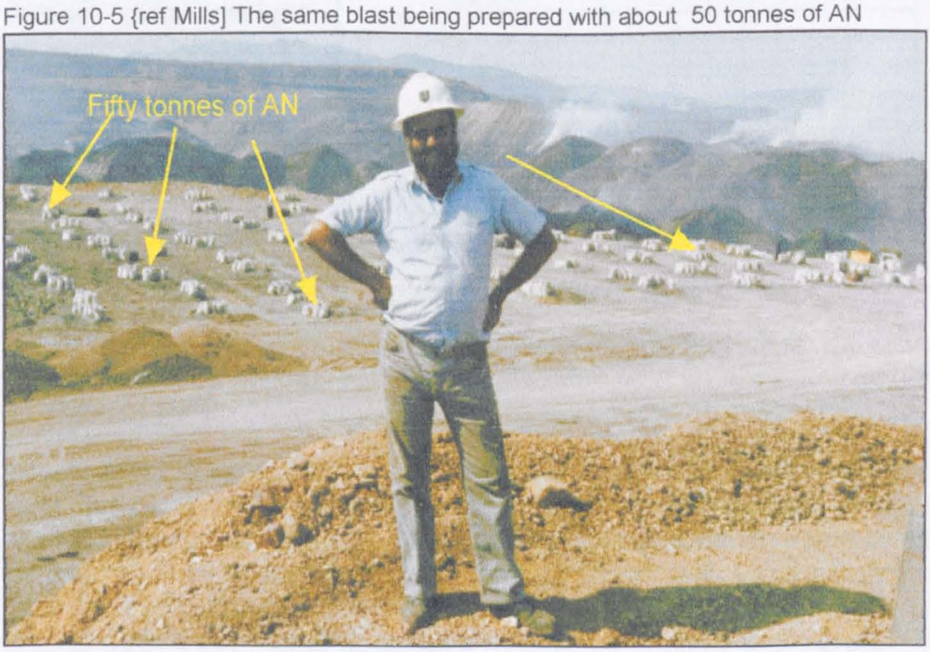


Figure 10-4 [ref Mills] The writer next to bags of AN ready for mixing and loading



burned by the oil. The equation of energy release for oxygen balanced ANFO; $3N_2H_4NO_3 + CH_2 \Rightarrow 7H_2O + CO_2 + 3N_2$ = 930 kcal/kg Under ideal conditions, dry porous ammonium nitrate intimately mixed with the fuel oil will produce the most acceptable gaseous products. A less than homogeneous mix will yield more carbon monoxide and more oxides of nitrogen. Fuel concentrations lower than 5.6% will produce excess

Figure 10-5 {ref Mills] The same blast being prepared with about 50 tonnes of AN



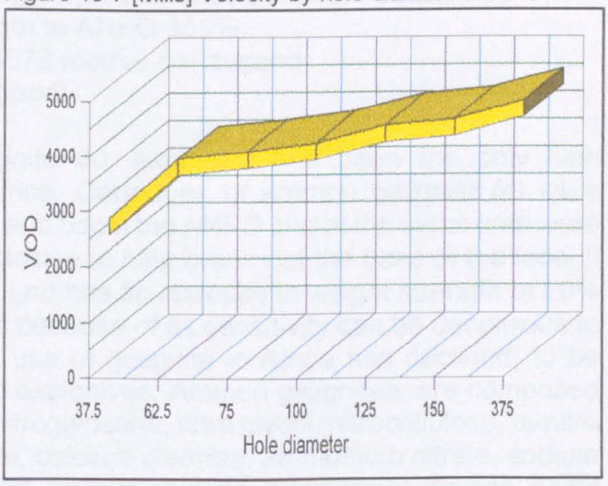
oxygen, power loss and light orange coloured nitrous oxide fumes. An excess of fuel will produce insufficient oxygen, minor power loss, brown coloured oxides of nitrogen and an unacceptable level of carbon monoxide. Oxides of nitrogen (NO and NO_2) are toxic and extended exposure to them may prove fatal. Nitrogen monoxide is a colourless gas which reacts immediately on contact with oxygen to form nitrogen dioxide. Nitrogen dioxide is a brown gas, it has a choking smell and irritates the throat, windpipe and lungs. Inhaling it can result in pneumonia.

Figure 10-6 [ref 5 et al] Velocity by hole diameter of ANFO

Typical confined velocities of detonation in ANFO	
Drill hole diameter mm	Velocity of detonation m/sec
37.50	2,438.00
62.50	3,535.00
75.00	3,657.00
100.00	3,810.00
125.00	4,127.00
150.00	4,236.00

In common with all explosives, the velocity of detonation of ANFO will increase with drill hole diameter until at 375 mm when it will stabilise at 4,572 m/sec. The weight strength of ANFO when made from 5.6% fuel oil and 94.4% ammonium nitrate is 70% of blasting gelatine and the bulk strength is 37% blasting gelatine. Blasting gelatine is the standard explosive from which the power of all other explosives is compared (excluding the USA and South Africa, where ANFO is the standard with an allocated weight strength of 100).

Figure 10-7 [Mills] Velocity by hole diameter ANFO



Loose, ANFO has no resistance to water and rapidly dissolves on contact. Wet ANFO will not detonate and any dampness will cause a deterioration in performance. ANFO is an insensitive explosive and cannot normally be initiated by No 6 or No 8☆ detonators (the number refers to the energy) and requires a booster or high energy detonating cord (more than forty grams loading per metre) to achieve reliable high regime detonation.

The main advantages of ANFO are; its low cost, safety and rock coupling, its main disadvantages are; its lack of strength, lack of water resistance, low VOD and insensitivity to detonator initiation. On-site mixing is usually carried out in the quarry area this has the advantage that, since the two components (fuel oil and ammonium nitrate) are inert up to the point of mixing, they are not usually subject to the safety precautions and regulations normally associated with the transport and storage of explosives.

10.1.6 Ammon Gelignite

Ammon gelignite was patented by C J Ohlsson and J H Norrbin in 1867 and named ammoniakkrut, Alfred Nobel bought the patent from them in 1870, this explosive uses ammonium nitrate to increase the oxygen content and act as an absorbent.

Typical characteristics are;

- density 1.25 kilos per litre
- detonation pressure 67 kbar
- absolute bulk strength 1,175 calories per cubic centimetre
- relative bulk strength to ANFO 159%
- confined velocity 4572 metres per second
- water resistance (good)

Historically ammon gelignite 60, explosive has been the only high explosive available in Africa. Cartridges of ammon gelignite 60 were commonly used to initiate and boost the ANFO and at the same time were used as a high power explosive to help break out the base of the face. It is reasonably waterproof, and has an increase in weight strength of 20% over ANFO. It is toxic and because of its sensitivity can be dangerous to use. In recent years the use of gelignite in Africa has declined, to be replaced by more modern explosives. Ammon gelignites, are composed of some of the following; nitroglycerine, nitro glycol, nitrocellulose, di-nitro toluene, potassium oxalate, calcium stearate, ammonium nitrate, sodium nitrate, cellulose, aluminium, sodium chloride, manganese dioxide, wood meal, flour and guar gum. Generally, when produced, African gelignites contained 61% ammonium nitrate, 32% nitroglycerine/glycol, 6% nitrocellulose and other fillers and stabilisers.

10.1.7 <u>Water gel explosives</u>

A gel is a coherent mass consisting of a liquid in which particles are either dispersed or arranged in a fine network throughout the mass. Gels are colloids (aggregates of fine particles, as described above, dispersed in a continuous medium) in which the liquid medium has become viscous enough to behave more or less as a solid. Slurry or water gels were developed in an attempt to produce an ANFO type explosive which was waterproof, one method of achieving this is to shield them chemically from the water. Work began by many engineers and chemists succeeded in 1957 when Cook and Farnam developed a TNT sensitised slurry. They found that by using a variety of water saturated oxidisers, together with a sensitizer and fuel, a satisfactory explosive could be produced.

The manufacture of a water gel explosive is a simple process whereby, the nitrate salts are mixed with a hot liquid solution consisting of water and guar gum (a long chain polymer). This produces a thickened liquid. A chromate "cross linker" usually borax is then added to link the polymer chains and form a firm jelly. The jelly prevents water from entering and escaping from the explosive. Slurries are packaged in a plastic bag which also gives some protection from water. Two families of water gels are available, depending on the supplier, the main difference is the sensitiser, whereby, one contains the isopropyl, monomethylamine nitrate (Du Pont) and the other contains fine platelets of aluminium (ICI), these are coated in a water repellent such as butyl or calcium stearate. Methylamine is an inflammable explosive gas, $CH_3 NH_2$, that has a strong smell of ammonia and is used in the synthesis of other chemical compounds. Aluminium is often used to raise the strength of explosives. To be used as a sensitizer, the fine aluminium is produced to encourage air bubbles to form on the surface. The air being compressed by the shock wave of detonation causes the aluminium to react violently and sustain a detonation front. Water gels produced with little or no aluminium contain hollow spheres of plastic or glass to contain an air pocket. Typical constituents of a water gel explosive are;

13% amine nitrate
15% water
05% sodium nitrate
03% ammonium perchlorate
63% ammonium nitrate
01% guar gum and borax

The explosive that is available and manufactured in southern Africa is believed to be an aqueous gel based on, ammonium nitrate, and will contain some of the following; monomethylamine nitrate, water, guar gum, laurylamine acetate, two grades of aluminium, sodium chlorate, sodium perchlorate, calcium nitrate, borax, microspheres and cenospheres. Although available in developed countries for more than 35 years (1958), water gels or slurry type explosives have only been available in South Africa since 1968. The current range of slurries being produced in Africa (since 1983) are of Du Pont style and use nitrate monomethylamine as a sensitizer, the trade name is Energex. An advantage of a water gel is that it can be mixed on site from ingredients that are not explosive, in addition, bulk deliveries are available which provide different strength mixtures.

This is particularly useful if strong base charge is to be used with a less powerful column charge. Bulk water gels have excellent wall contact, this helps in the transfer of a strong tensile shock wave. A further advantage of using water gels is that they are more powerful than the local gelignites. A failing of water gels is that they do not travel well and only have a short shelf life, this is due to the air bubbles migrating to the surface. Water gels will decay to a safe state were they are no longer able to detonate. Eventually the gel structure will break down and the suspended salts will dissolve and escape.

Figure 10-8 [ref 46] Characteristics of water gels

product name	Typical characteristics for water gels (Exchem)				
	strength % ANFO		density l/kg	velocity m/sec	gas volume l/kg
	weight	bulk			
Delta plus	140.00	240.00	1.35	5,100 to 5,300	850.00
Delta 100	136.00	205.00	1.20	4,600 to 4,800	850.00
Delta 200	120.00	180.00	1.20	4,600 to 4,800	900.00
Delta 300	95.00	148.00	1.25	3,800 to 4,000	990.00

10.1.8 Emulsion explosives

In physical chemistry, an emulsion is a mixture of two or more liquids in which one is present as droplets, of microscopic or ultramicroscopic size, distributed throughout the other. Emulsions are formed from the component liquids either spontaneously or, more often, by mechanical means, such as agitation, provided that the liquids that are mixed have no (or a very limited) mutual solubility. Experiments in developing a cheaper and less complex water gel led in the 1960's to the development of emulsion explosives, when it was discovered that with entrapped air, the basic ingredients of ammonium nitrate, water and fuel in the form of water in oil emulsion could be detonated. This discovery of a high performance explosive was unexpected.

Emulsion type explosives are two phase products prepared in the form of water in oil emulsions, whereby the dispersed phase is disseminated throughout a continuous phase. The internal phase is composed of a solution of oxidizer salts suspended as microscopically fine droplets that are surrounded by a continuous fuel phase, the ensuing emulsion is stabilized against liquid separation by an emulsifying agent. Advanced engineering has ensured that the available surface area is coated with a thin layer of wax at a specific thickness to ensure a good fuel to oxygen ratio. The oxidant to fuel ratio is approximately 10:1 and the structure of the emulsion, when inspected by an electron microscope, exhibits a polyhedral shape with each droplet covered by a film of the fuel phase, the wax acts as a fuel and binder. Initiation is provided by the compression of air bubbles, these are either natural , or are tiny, hollow glass spheres.

The detonation reaction occurs at the .boundaries between the two phases. Emulsions provide increased explosive efficiency because both phases are liquid and the dispersed nitrate solution droplets are several orders of magnitude smaller than other conventional explosives; 0.001 mm compared to 0.2 mm. They are tightly packed within the fuel phase and provide increased surface contact enhancing the reaction efficiency. The strength of the reaction can thus be altered by changing the degree of mixing of the fuel and oxidant.

The increase in efficiency is reflected in the velocity of detonation, 5000-6000 m/s for emulsions compared with around 3200 m/s for ANFO and 3300 m/s for slurry. This high velocity of detonation is one of the major advantages of emulsions as it provides high shock energy, a significant factor with hard rocks. Unlike other explosives, emulsions do not use chemical sensitisers. Instead, the presence of voids in the emulsion fulfils this requirement; the number of voids determines the density of the mixture. The viscosity and density of any emulsion is determined largely by the physical characteristics of the organic fuel phase, which can vary from liquid fuel oil to viscous waxes. Unlike slurries, emulsions cannot be gelled or cross-linked, their structure being characteristic of the nitrate and continuous fuel phase. Emulsions are structurally distinct from slurries, containing no thickeners or gelting compounds, and require mixing at around 8O degrees C. The lack of structural additives means the phases have to cool by 3O to 5O degrees C before the fuel phase becomes semi-rigid.

Emulsions are water resistant, give high energy, are resistant to shock and friction, produce high velocity of detonation and degrade to safety. Bulk emulsion explosives are especially suited for quarrying by virtue of their water resistant properties and chemical stability. The consistency of the explosive can be altered by changing the proportions of oil and wax to produce either a cartridged product or a pumpable slurry. This versatility of explosive force, sensitivity, and consistency means that the same basic product can be accurately tailored to meet each application. By heating the wax, emulsions can be mixed in a truck and pumped into the hole where it will harden. The strength can be altered by adjusting the AN to fuel ratio and by adding other fuels. When being supplied in bulk and loaded directly into the drill hole it is possible to dispense with microspheres and as the explosive is being pumped into the hole include a gas. Solid fuels such as aluminium can be included to increase the energy and if desired, the emulsion can be used in the same drill hole as ANFO, the result is a reasonably high density explosive with a relatively high bulk strength. If more than 50% emulsion is used, the mixture will for a short time be waterproof. Blends of emulsion/Anfo are becoming very popular with each ingredient bringing its own benefits to the blast.

Figure 10-9 [ref 5] Characteristics of water emulsions

Typical characteristics for emulsions (Atlas)					
strength		density l/kg	velocity m/sec	Detonating pressure k/bar	product name
absolute cal/cc	relative Anfo=100%				
820.00	111.00	1.19	5,791.00	100.00	Powermax 420
1,140.00	154.00	1.19	5,639.00	100.00	Powermax 440
1,310.00	177.00	1.21	5,334.00	90.00	Powermax 460
1,275.00	172.00	1.35	5,791.00	N/A	Powermax 840

Typical ingredients for a water in oil emulsion are;

06% wax/oil
02% emulsifier
17% water
58% ammonium nitrate
15% sodium nitrate
02% glass microspheres

With some encouragement emulsion explosives can be destroyed by burning, otherwise they degrade to safety, because of this inherent safety, and cheap cost, this type of explosive will lead the industry.

10.2.0 INITIATION SYSTEMS

10.2.1 Introduction

An initiation system consists of the devices and equipment that are used to cause the main charge to detonate and time the blast. There are three major initiation systems and these can be identified as;

- chemical
- electrical
- non-electrical

Each of the above systems is covered in detail below.

10.2.2 Chemical

This system usually consists of, igniter cord, beanhole connectors, safety fuse, plain detonators, detonating cord and detonating relays. Igniter Cord is a plastic protected cord of pyrotechnic composition developed for lighting a number of safety fuses in series via beanhole connectors. Two types are available (fast and slow). When ignited, an intense flame passes at a uniform rate along the cord to ignite the black powder core in the safety fuse. The burning speed of the brown coloured, fast igniter cord is approximately 5 seconds per metre (SPM) and the burning speed of the green coloured, slow igniter cord is approximately 40 seconds per metre. It can be used to ignite trunk lines of Fast Cord acting as a delaying agent. Both cords have excellent water resistance and their burning speeds are reliable and consistent even under adverse conditions. Beanhole connectors consists of aluminium tubes closed at one end and containing a plug of pyrotechnic composition. An oval aperture is cut through the tubes and the incendiary composition, through which a loop of plastic igniter cord can be inserted. Into the open end of the tube safety fuse is inserted. The safety fuse consists of a core of black powder wrapped with layers of textile yarn or tape and waterproof coatings to provide protection against mechanical damage and water penetration. The final coating is of white alkathene. The normal burning speed of safety fuse is 100 seconds per metre, this can be altered by using powder with a different grain size, whereby smaller grains will react faster than large grains.

Figure 10-10 [ref AECI] Igniter cord

Plain detonators are small aluminium tubes closed at one end with a base charge of pentaerythitol-tetranitrate PETN. and a priming charge of lead azide they are usually number "six" strength, but number "eight" strength may be specified. The safety fuse and plain detonators used to be brought to the quarry as individual items, the fuse would be cut at a predetermined length, this to ensure sufficient time for the shotfirer to escape the area. The plain detonator would be crimped onto the fuse and a bean hole connector would be crimped onto the other end. Plastic ignitor cord would be inserted into the bean holes connector for easier ignition and if required, further delay. Nowadays, to eliminate the main cause of misfires, capped fuses are used, these are identical to the above, however, they are factory made with specially designed crimps and efficient waterproof seals.

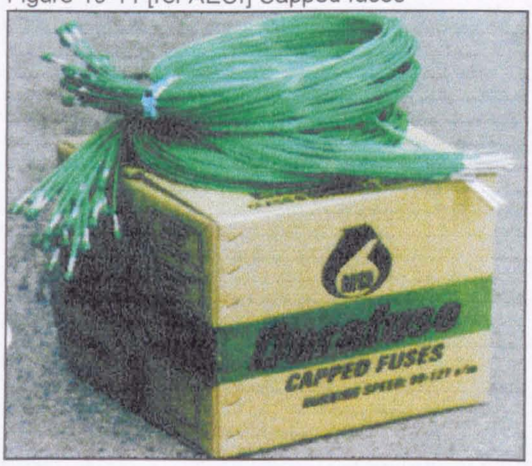


Figure 10-11 [ref AECI] Capped fuses

Detonating cord resembles safety fuse but contains a high explosive instead of black powder. The first successful one, patented in France in 1908, consisted of a lead tube, about the same diameter as safety fuse, filled with a core of TNT. It was made by filling a large tube with molten TNT that was allowed to solidify. The tube was then passed through successively smaller rollers until it reached the specified diameter. In France the product was called cordeau détonant, elsewhere shortened to cordeau. Its velocity was about 4,900 MPS.

Modern detonating cord consists of a hollow core which is filled with the very powerful explosive, PETN, and enclosed in a tape which is wrapped with polypropylene countering yarns. To ensure water resistance, the cord is then totally enclosed in a tubular cover of plastic material. This explosive is reasonably resistant to accidental initiation, it is less sensitive than nitroglycerin but is easily detonated. PETN is a highly explosive organic compound belonging to the same chemical family as nitroglycerin, that is, the nitric acid esters of poly-alcohols, and is a nitrate of the organic esters acetaldehyde and formaldehyde. PETN is a colourless, crystalline material that is generally stored and shipped as a mixture with water, even so, PETN will reliably detonate even when wet.

Figure 10-12 [ref 46] Detonating cords

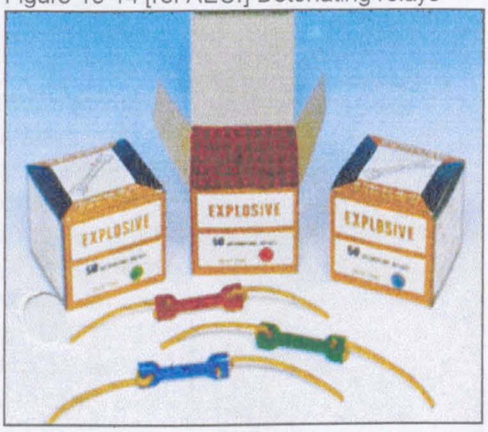
Product name (Exchem)	Details of the range of detonating cords sold by Exchem				
	Core loading grams/metre	Diameter m/m	VOD metres/second	Tensile strength kg	Colour of cover in UK
Pentaflex 6	6.00	4.20	6,500.00	50.00	green
Pentaflex 10	10.00	4.80	6,500.00	60.00	white
Pentaflex 12	12.00	5.00	6,500.00	60.00	white
Pentaflex 15	15.00	5.20	6,500.00	75.00	white
Pentaflex 20	20.00	6.40	7,000.00	80.00	red
Pentaflex 40	40.00	7.90	7,000.00	125.00	brown
Pentaflex 80	80.00	10.30	7,000.00	150.00	yellow
Pentaflex 100	100.00	11.80	7,000.00	150.00	bronze

Detonating cord has many applications in blasting, any number of holes can be connected with it in just about any desired pattern. Attached to the blasting charge and knotted to a trunk line, it is fired by means of either a fuse-type or electric blasting cap. Sequential timing may be obtained by cutting the trunk lines and inserting delay connectors, which have delay periods ranging from about 12 to 47 milliseconds. By changing the amount of charge per metre from about 3 grams to 60 grams per metre, detonating cord can be supplied in many different strengths and as a consequence, is used for many different purposes, including use as a primary explosive. Detonating cord does not readily deteriorate and will remain in good condition over several years of storage.

Figure 10-13 [ref 46] Detonating cords

The velocity of detonation of cords is between 6000 and 7000 metres per second and it is one of the safest products to handle. Unless the cord is crushed, and, in some way the PETN. core is accidentally initiated by a second blow, accidental detonation of det cord by impact is virtually impossible. During his career, the writer has probably see several hundreds of metres of cord pass through crushing plants.

Figure 10-14 [ref AECI] Detonating relays



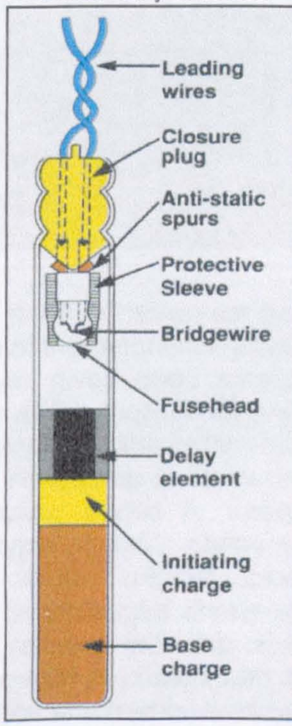
Detonating relays are devices for introducing a delay in a detonating cord, trunk or surface line. The relay comprises two small detonators with an integral delaying element placed between them. To ensure that the detonation is reliable, relays are only recommended for use with detonating cord of ten gram per metre loading. In the UK, three delay periods are available, these being, 12 m/s (blue), 25 m/s (green) and 40 m/s (red). Relays are bi-directional and in the event of the trunk line being cut they can be used with "back up" lines to ensure reliability. The colours indicated above are only valid for Southern Africa.

10.2.3 Electrical

The principal advantage of electric firing is exact control of the time when the blast is initiated. Attempts to make electric blasting caps date back to the 1700s, but nothing of a really practical nature was developed until late in the 19th century Nobel's original fuse-type blasting cap remained virtually unchanged for many years, except for the substitution of 90-10 and 80-20 mixtures of mercury fulminate and potassium chlorate for the pure fulminate. The modern commercial electric detonator as used in quarrying, consists of a thin walled tube of 8 mm diameter, made from aluminium, closed at one end and containing a high explosive base charge, a priming charge and a fusehead. The fusehead consists of two metal foils separated by a layer of insulation. The leading wires are soldered to the base of the foils and a very fine wire connects their tips. Around this wire is an igniting composition, this is usually constructed in several layers, the innermost being readily ignited by heat. When an electric current of sufficient power is passed down the lead wires, the fine wire in the fusehead rapidly heats up to the point where the igniting composition flares and initiates the priming charge which, in turn, initiates the base charge. The tube is sealed with a Neoprene plug through which the leading wires of the fusehead assembly pass.

Many systems of electrical detonation are available, from the instant plain detonator which is most commonly used to fire the trunk line in a chemical system, to the advanced, but little used Magnadet system In quarrying the most commonly used system is the millisecond delay detonator or short period delay (SPD). The construction of a delayed action detonator is like that of a standard electric detonator, except that a special delay element has been introduced between the fusehead and the base charge. This delay element consists of a column, of slow burning composition contained in a thick walled metal tube. The length of this column determines the amount of delay time introduced into the detonation train. This system provides a number of delays depending on the country of use with a range in the UK of 31 detonators in a series numbered 0 to 30 with a delay interval from numbers 1 to 12 inclusive of 25 milliseconds and from numbers 13 to 30 with intervals of 30 milliseconds. In South Africa, the range is numbered from 0 to 20 and the intervals vary from 25 to 100 milliseconds. The main restriction on these detonators is the limited number of delays that are available and their accuracy.

Figure 10-15 [ref AECI] Electrical delay detonator



The use of electrical detonators is prohibited in many quarries, especially those extracting metallic ores. Using these detonators puts pressure on the operator to take special care of extraneous electrical currents, such as radio waves, static, lightening and currents differential inherent in metallic ore bodies, any of these can fire an electrical detonator and frequently have. The use of this system is reducing and it is possible that within a few years they will no longer be commonly available.

10.2.4 Non electrical

NONEL is a trade name registered by the Nitro Nobel Company of Sweden, for a non electric initiation system which was introduced to the mining industry in 1973. In Zambia the system is known as both NONEL and "shock tube".

Figure 10-16 [ref 46] The Nonel range of products

Shock tubes are a non-electric system of initiation based around a hollow plastic tube coupled to a No 8 star detonator. Recently, the original tube has been replaced by the introduction of a three layer composite tube. Each of the layers has different characteristics, with the innermost layer having good adhesion for the explosive and an ability to withstand the passage of the detonation front.

Figure 10-17 [ref 46] A NONEL tube

The middle layer gives good tensile strength (25 kg at 20 degrees C) and chemical resistance and the outermost layer has good resistance to abrasion and can be colour coded to easily recognise the type of delay element. The problem of the original tube "remembering" its packaged shape of coils has been reduced with the new tube. Inside, the tube is coated with a reactive substance which when fired by an external device (high voltage spark or some form of detonation) is mixed with air in the tube and forms a cloud of explosive material, this detonates and causes a shock wave to travel down the tube. The velocity of the shock wave will reach 2,100 metres per second. The composition of the reactive substance remains unavailable to the general public, the writer believes it to be a military style aluminised explosive of the RDX (Research Development Explosive), cyclotrimethylene-trinitramine $C_3H_6N_6O_6$ or HMX, cyclotetramethylene-tetranitramine, family of explosives, with a formula of $C_4H_8N_8O_8$. This explosive has a confined velocity of detonation of 9,100 metres per second.

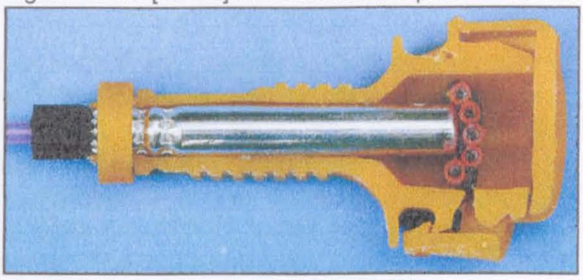
The shock wave carried through the tube contains sufficient energy to fire the detonator which is securely clamped onto the tube and sealed with neoprene. The most recently developed detonators by Nitro Nobel are named Non Primary Explosive Detonators (NPED). The detonator appears the same as any other aluminium shell detonator and the sealing plug and delay element are the same as before. The main difference is that they do not contain any sensitive primary explosives such as, diazodinitrophenol ($C_6H_2N_4O_5$), fulminate of mercury ($C_2N_2O_2Hg$) or lead azide ($Pb(N_3)_2$), this substantially reduces the risk of accidental detonation through flash over and impact, also lead and mercury fumes are eliminated from the gases.

Figure 10-18 [ref 46] Snapline connectors with colour coded delay times.

The base charge in the NPED detonator consists of various grades of hexogen which is also known as; thylen-trinitramin with a formula of $C_3H_6N_6O_6$, the different grades of hexogen allow the propagation of the material in the delay element to rapidly transition to detonation. The initiation element is PETN which is also known as, penitrit or pentaerythitol-tetranitrate with a formula of $C_5H_8N_40_{12}$. Apart from increased safety, the main advantage of using a shock tube system is that, since the shock wave is contained within the tube, the main explosive column is not affected by the passing shock wave, thus removing the problems encountered when using detonating cord with the dead pressing and consequent de-sensitisation of the explosive.

The usual method of timing a blast using the NONEL UNIDET system is to use long period delay detonators in the hole, and join the lines of holes with zero delay or instantaneous detonators. The timing of the blast is achieved by using millisecond delays to connect the various lines. These surface delays are incorporated into the connector and are known as snapline connectors and can be specified to incorporate a delay element of 0 (green), 17 (yellow), 25 (red), 42 (White), 67 (Blue), 109 (black), or 176 (orange) milliseconds. The detonators which are used in the hole can be supplied with 400, 425, 450, 475 or 500 millisecond delay elements. The purpose of these delays is to ensure that the detonation front has passed to the down the hole detonators before any of the column charges fire. This considerably reduces the risk of fly rock cutting the cords or shock tubes and otherwise disrupting the charged holes, it is known as advanced initiation.

Figure 10-19 [ref 46] A sectioned snapline connector



Timing with the NONEL system is almost without limit, however, if it is intended that the whole of the surface detonators have fired before the first of the sub surface charges, then timing will be limited to the number of 17 millisecond delays that can be accommodated in 500 milliseconds, ie 29 delays, should this be a real problem, a further 500 millisecond delay can be installed prior to the sub surface one, this will double the time of the advanced initiation and therefore, the possible number of delays.

10.2.5 Future developments

Since the late nineteen-eighties engineers have been working on a means of producing a detonator that is totally accurate and flexible. This work has led to the development of a new type of electric detonator with an accuracy of scatter less than 1m/s. The device is know as 'An Electronic Delayed Action Detonator' and consists of a standard electric detonator, that has a very small micro chip and one or two capacitors added to it. The whole of which is encased in a standard aluminium detonator tube. With this system, each detonator has its own identity and memory. To operate the system, first the holes are primed and charged as normal, then each detonator is linked in turn to a computer which will give the detonator a unique identification, and time to fire. The need for this complexity has yet to be proven. Although seemingly of much benefit to the community, it is probably of more value to the developers of the computer software and purpose manufactured hardware. It is likely that pressure from the manufacturers will persuade the legislative authorities that scatter characteristics of less than 1 m/s is good for the industry and that their use should be a legal requirement, the writer believes that this level of sophistication is more suitable for military rather than civilian use.

10.2.6 Summary

Blasting explosives are an essential rock-breaking tool in the mining, quarrying and civil engineering industries. There has been a long, fascinating development path from the rather crude and hazardous liquid nitroglycerine of Sobrero to the efficient and user friendly emulsions of today. The purchase of NG explosives is now virtually impossible, the choices that are currently available are ANFO, water gels and emulsions or any combination of the same. These modern explosives match their predecessors in most applications and are considerably safer to use.

Thirty years ago, the quarrying industry was happy to use nitro glycerine based explosives and chemical initiation. This combination worked in small diameter holes was very effective at shattering the rock and was generally accident free. Then explosive manufacturers persuaded the operators that ammonium nitrate based explosives were the way ahead, mainly because they were cheaper and safer, unfortunately these did not always work well with chemical initiation, particularly ANFO which was often destroyed by the detonating cord. To overcome the problems with ANFO, methods were developed to restore the popularity of electric initiation systems and the Magnadet system was developed, unfortunately this system also had many problems, not least of which was the small number of detonators that could be fired from one exploder. Then came NONEL, this system is a totally flexible chemical system that does not destroy the ANFO and works very well, however, the quarrying industry is now being asked to consider going back to electric detonators, and the reason being given is that they are more accurate.

Under supervision of the writer, Chilanga Works Zambia, Changalume (Portland Cement Company Malawi) and Ndola Works Zambia all changed the basis of their blasting to use emulsion or water gel for the base charge, ANFO as the column charge and NONEL as the initiation system. Ndola Lime Company continues to use either NG explosives, water gels, emulsions together with ANFO and either electric detonators, or the chemical system of detonation. Needless to say the blasting results at the first group mentioned is far superior to those of the second.

The development of the use of explosives in the UK has now reached a stage whereby the manager of a quarry will ask an outside consultant, to design the blast, the drilling will be carried out by a sub contractor. The initiation circuit will be designed and possibly applied by the manufacturers and the explosive will be ordered much as it is now with a call being placed to the supplier who will mix it on site to order with the added dimension of ordering emulsion/ANFO mixtures with the choice of VOD, and varying degrees of water resistance strength and density.

It is tempting to state that the development of commercial explosives has now reached a peak, but it is likely that they will continue to run in the same way as any other product with a new and updated version being introduced on a regular basis. For quarrying purposes its probably true to say that, explosives are now as good as they will get, and any "improvements" will be marginal and mainly for the use of the suppliers marketing department.

Chapter Eleven

Blasting

11.1.0 ROCK BLASTING

11.1.1 Introduction

The major difference when quarrying rock for cement or lime production as opposed to aggregate is the need to produce a consistently homogenous material. This chapter demonstrates one of the methods of blasting that the writer has developed into practice to improve the blending of rock and it represents the central theme of this document. The writer believes that blasting remains as much skill or art as science, where the sum of the blast is more than the addition of its individual parts. The results of a blast depends largely on the skill and experience of the shotfirer who will consider many more parameters than those presented to him by a computer, however sophisticated the programme. The modern trend is to map benches accurately and perhaps using a computer programme, calculate the best drilling and charging programme. A good combination of common sense and accuracy will often give better results, especially in a third world environment, where laser measuring devices and computers are rarely available.

Blasting represents a significant cost in any quarrying project and it should be borne in mind, that explosives are simply another source of energy, albeit with a very rapid reaction time, they constitute an almost instantaneous and enormous release of energy with pressures of up to 4,000,000 psi, temperatures of 4000 degrees centigrade and detonating speeds reaching 8000 metres per second. The efficient use of this energy is paramount to controlling raw material processing costs and limiting damage to the environment. An analysis of the true cost of using explosives requires that the results of the blasts are placed into a proper perspective with regard to the cost effectiveness of the whole undertaking, whereby the results of the blast will effect the smooth running of the remainder of the operation. The main factors to be considered are;

- Required fragmentation
- Dig or loading cycle
- Bucket fill factor
- Wait times
- Truck capacity and capability
- Haul distance and speed
- Crusher hopper capacity
- Grizzly size
- Crushing characteristics and use of energy
- Production rates
- Secondary breaking
- Unit costs
- Plant utilisation
- Operating costs

In the UK, the imposition of environmental thresholds with regard to noise and vibration that are sometimes uneconomic to attain, have thrust very large costs upon the producer, in the coming years, it is likely that third world countries will follow this trend. Should this happen, the type of effective blasting that is illustrated in this chapter will be impossible to carry out. The method and style of blasting described below gives results that are outstanding in terms of cost effectiveness, fragmentation, heave, and blending however they will not necessarily conform to the latest UK environmental recommendations of Minerals Planning Guidance note 2 which recommends a PPV of less than 10 at site boundary).

11.1.2 <u>Mechanics of Rock Blasting</u>

The mechanism of blasting and rock breakage is extremely complex and scientists have not yet come up with an entirely consistent conclusion. The later research seems to be pointing towards the nuclei stress flow theory, this was developed at the University of Maryland USA and took into consideration the joints, fractures and bedding planes etc. [ref 5]

Figure 11-1 [ref 5] Blasting theories

Blasting Theories						
Date	Researcher	Breakage mechanisms				
		Tensile reflected waves	Compressional stress waves	Gas pressure	Flexural rupture	Nuclei stress flow
1949	Obert & Duvall	1				
1956	HINO	1				
1957	Duvall & Atchison	1				
1958	Rinehart	1				
1963	Langfors & Kihistrom		2	1		
1966	Starfield	1				
1970	Porter & Fairhurst		2	1		
1970	Persson et al		1			
1971	Kutter & Fairhurst		1	1		
1971	Field & Ladegarrd-Pederson		1	1		
1972	Johansson & Persson	2		1		
1972	Lang & Favreau	4	2	1		3
1973	Ash			1	1	
1974	Hagen & Just		1			
1978	Barker et al					1
1983	Winzer et al					1
1983	Margolin & Adams et al					1
1983	Mc Hugh					1

Figure 11-2 [ref Murmanski] An assortment of blasting formulas.

Equation	Author	Explanation	Equation	Explanation
$B_{max} = \frac{K}{50}\left(d^{8/3} h_c H\right)^{0.33}$	Fraenkel (1944)	K = blastability factor (0.72-1.1) h_c = charge height, d hole diameter, mm H=depth of the blast hole	U = 10*d	H, Bench height (m) d, Borehole diameter (m)
$B = K d\sqrt{P_s / \sigma_t}$	Pearse (1955)	K = 0.7-1.0 P_s = Borehole pressure, MPa	U=0.2-0.5*B	U, Subdrill Length (m)
$B = 10^{-3}\frac{d}{4}\left(P_d/\sigma_t\right)^{1/n}$	Hino (1958)	n = 1.5 (average), σ_t = Tensile strength, kg/cm² P_d = Detonation Pressure, MPa	$L_b=(H+U)\cdot k_{inc}$	L_b, Borehole Length (m)
$B_{max} = 38d\sqrt{\rho_e/\rho_r}$	Ash & Konya (1968)	ρ_e =Explosive density, kg/m³ ρ_r =Rock density, kg/m³	F=d/10³+0.03*H	F, Total Error (m)
$B_{max} = 0.958d\left(\rho_e S_e / c_0 f (S/B)\right)^{0.5}$	Langefors & Kihlström (1976)	c Blastability factor (Dynamex M kg/m³) c_0 = c+0.05 (B=1.4-15 m), c_0 = c+0.07/B (B<1.4 m) d=30-89 mm, S_e= Weight strength f = Hole limitation number	L_s=0.7-1.3*B	L_s, Stemming (m)
$B = \frac{K d}{12}\left(\rho_e V_d^2/\sigma_t\right)^{0.5}$	Borquez (1981)	K = 1.96 - 0.27 ln(ERQD) ERQD= RQD ϕ_a, ϕ_a = 1, 0.9, 0.8, 0.7 (hard, medium, weak, very weak rock)	$q=Q_{tot}/S\cdot B\cdot H$	q, Specific Charge (kg/m³)
$B = K d\left(P_s/\sigma_t\right)^{0.5}$	Nova & Zaninetti (1990)	$\sigma_t = \frac{\sigma_{min}\sigma_{max}}{\sigma_{min}\sin^2\alpha_{ek} + \sigma_{max}\cos^2\alpha_{ek}}$ α_{ek} = Angle between tensional force and joint plane direction	L_{bot}=1.3*B	L_{bot}, Bottom Charge Length (m)
$B=0.67d\left(S_{ANFO}/\rho_r\right)^{0.33}$	Konya & Walter (1990)	S_{anfo}=Relative strength (base ANFO)	L_{col} = H-L_{dip}-S_t	L_{col}, Column Charge Length (m)
$B = 1.25d^2\left(\frac{R_d^2 \Pi \rho_e E_d}{\sigma_c^2 h \tan\theta/2}\right)^{0.5}$	Kou & Rustan (1992)	E_d =Dynamic Elasticity Modulus, MPa θ=Angle of breakage, ° R_d =Decoupling ratio σ_c=Compressive Strength, kg/cm² h=energy of breakage / max deformational energy	S=1-1.8*B	S, Spacing (m)
$B_c = 69.5 - 1{,}795\cdot Z_r$ (rock block tests, $R^2 = 0.85$)	Rustan & Nie (1992)	B_c Critical burden (mm) Z_r Rock impedance, kg/m²s	$Q_{bot}= L_{bot}\cdot A\cdot \rho_{bot}$	Q_{bot}, Bottom Charge Weight(kg)
$B_c= 78.42 - 7.6 \ln(K)$ (rock block tests, $R^2 = 0.945$)	Mamurekli (1996)	$K = \frac{(1-v)}{(1-2v)(1+v)}E$ (GPa)	$Q_{col}= L_{col}\cdot A\cdot \rho_{col}$	Q_{col}, Column Charge Weight(kg)
Emprical Equations of Fragmentation Size				
$K_{50} = 28 K_1 f(L_t/L_b)(B\,S)^{0.29}[B_{max}/(B\,S)^{0.5}](c/q)^{1.18}$ Swe De Fo - Nitro Nobel	Hjelmberg (1983)	K_{50}, Average fragment size, (cm), K_1, Blasting factor B_{max}, Maximum Burden, m		
$K_{50} = A q^{-0.8} Q^{1/6} (E/115)^{-19/30}$ Kuz - Ram	Cunningham (1987)	A, Rock quality factor (7, 10, 13 for medium, hard but highly fissured, very hard, weakly fissured rocks subsequently), E Relative weight strength of the explosive (ANFO 100, TNT=115)	A=0.06(RMD+JF+RDI+HF) JF = JPS+JPO RDI = 25 ρ_r -50 HF = E/3 E<50 Gpa HF = σ_c/3 E>50 GPa	RMD, Rock Mass Designation HF Hardness Factor RDI, Rock Density Index JPS, Joint Plane Spacing index JPO, Joint Plane Orientation Index
$R = 1 - e^{-0.693\left(X / K_{50}\right)^n}$ Rossin-Rammler(RR), Uniformity index	Cunningham (1987)	X, the diameter of rock fragment, cm, n, RR frag. gradient	$n=(2.2-14\,d/B)(1-F/B)((1+S/B)/2)^{0.5}((abs((BCL-CCL)/L_t))+0.1)^{0.1}(L_b-U)/H$	BCL, bottom charge length, m , CCL, column charge length, m
$K_{50} = 1.24 k(\sigma_t \rho_r / V_P)^{0.38}$ $(1/(0.72 q)^\varepsilon)$ Unifrag	Nie (1988)	k, Scaling factor between full-scale tests and the model tests, ε Exponent for specific charge (1.29-2.89) depends on the rock type tested, σ_t, Tensile strength, MPa V_p, P-wave velocity, m/s		

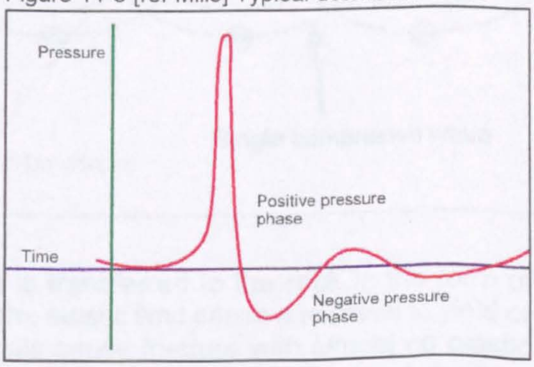
The researches concluded that the flaws acted as a nuclei for crack formation. It is likely that this theory coincides with the results described by the writer in his work carried out over the past thirty years to reach a practical method to maximise fragmentation. If nothing else, the above chart and list of theories indicates that there is a wide variety of thought on how explosives break rock, for practical purposes the writer only considers the reflective wave theory. Viewing high speed photography of a rock face when subject to the effects of blasting and examining countless papers and reports has convinced the writer that it is an interaction of the compressive and tensile waves that causes the rock to break and many of the propositions offered in this chapter are based on that belief. A discussion of all the variables of rock breakage will ensure that the permutations will be endless, to limit the variables, the explanations given below deal only with the use of high explosives when used to blast hard monolithic rock, where Young's modulus, elastic strain and homogeneous and isotropic stress and strain can be regarded as a constant. In reality, on a microscopic scale there are grains and pores in sediments and a fabric of crystals in igneous and metamorphic rocks. On a large scale, rock bodies exhibit physical and chemical variations and structural features. Furthermore, conditions such as extended length of time, confining pressure, and subsurface fluids affect the rates of change of deformation. When considering the effects of explosives on rock, the following two constants should be recognised;

- explosives break rock by stress and strain produced by a tensile shock wave
- explosives move and further break the rock by the application of gas pressure

Shock wave

When a high explosive is detonated, the progressive acceleration of reaction accounted for by growth of the flame front area and by transition from laminar to turbulent flow gives rise to a shock wave. The increase in temperature due to compression in the shock wave results in self-ignition of the mixture and detonation sets in, this travels through the explosive from the point of initiation. The shock wave combustion zone complex forms the detonation wave and approximately 15% of an explosives total energy is manifested in the formation of the shock wave. The speed of the shock front will soon stabilise to the natural 'velocity of detonation' (VOD) of the particular explosive.

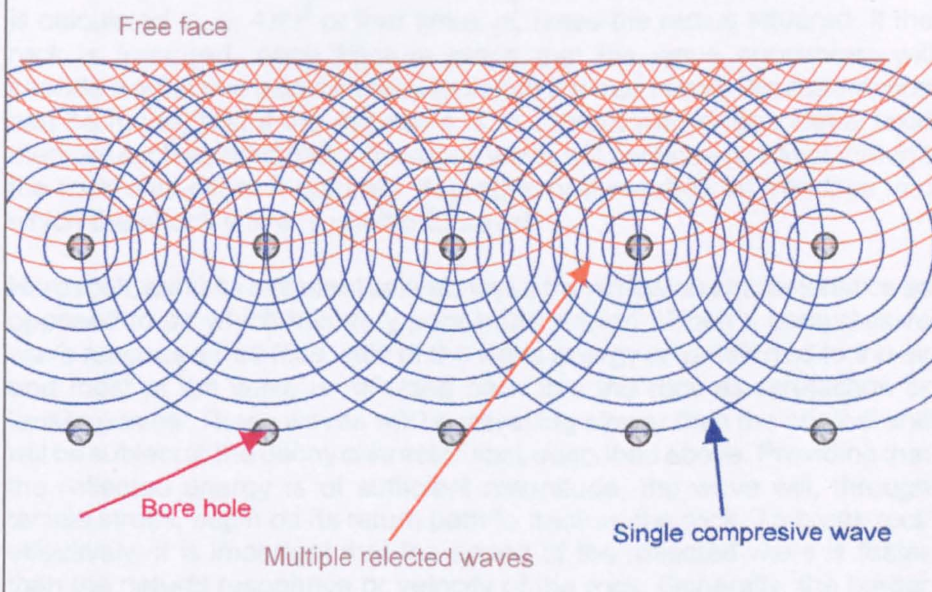
Figure 11-3 [ref Mills] Typical detonation curve



This could be somewhere between 1000, and 8,500 metres per second (for comparison, an average high velocity rifle bullet ((NATO 7.62mm)) travels at 1,000 metres per second), an average VOD for quarrying explosives may be 5,000 MPS or 5 metres per millisecond (m/s). When using bottom initiation, the time taken for the wave to reach the top of a seven metre explosive column is 1.4 m/s. The wave progresses spherically from the source, into the surrounding medium in the form of a single or multiple discontinuous compressive phase with a steep forward shock front that is immediately followed by a negative phase. During the initial compression stage, materials behind the shock front experience pressures so high that their dynamic compressive strengths are greatly exceeded. In the negative phase the pressure declines to zero, but the particle velocity does not. This soon sets up rarefactions (pressure differentials) that effectively place the rock medium under tension, resulting in a general disruption of the component units.

Figure 11-4 [ref Mills] Shock wave progression

This image shows the progression of shock waves in a typical blast, it assumes that the rock is intact and entirely flawless, because of this, the return waves are perfectly symetrical. In an actual rock, there would be bedding planes, jointing, fractures and crystal faces, these would produce millions of reflections and refractions. Also, the drawing assumes that all the holes fire simulateously, in practice, there can be up to eight or more milliseconds difference in the reaction times of supposedly identical detonators. Together these flaws would make a mathmatical model virtually useless, however, it is these reflected surfaces and different reaction times that the writer believes produce an 'augmentation by interaction' in the behavior of the waves.



The energy of the shock wave is transferred to the rock in the form of equal elastic (stresses beyond the elastic limit cause a material to yield or flow and for most brittle materials cause fracture with almost no plastic deformation), and kinetic (the description of motion in terms of position, velocity, and acceleration, concerning the effect of forces and torques on the motion of bodies having mass), components, these can cause the rock to display fluid properties, whereby subject to its elasticity, a plastic flow or fracture takes place. The wave can be measured by both its amplitude and its frequency. The amplitude is proportional to the square root of the energy per unit area, ie, it is inversely proportional to the distance that the wave has travelled. In all mediums, higher frequency waves undergo greater attenuation that those at a lower frequency. As it progresses from the source, the wave will degrade and the following is applicable;

The energy of a wave in a given medium is proportional to the square of its amplitude. As a spherical wave moves away from its source, the energy will be distributed over the area of the sphere which increases as the square of the spheres radius. Therefore, the energy per unit area varies inversely as the square of the distance from the source, the amplitude, which is proportional to the square root of the energy per unit area, should be inversely proportional to the distance the wave has travelled. [ref Geophysical Prospecting]

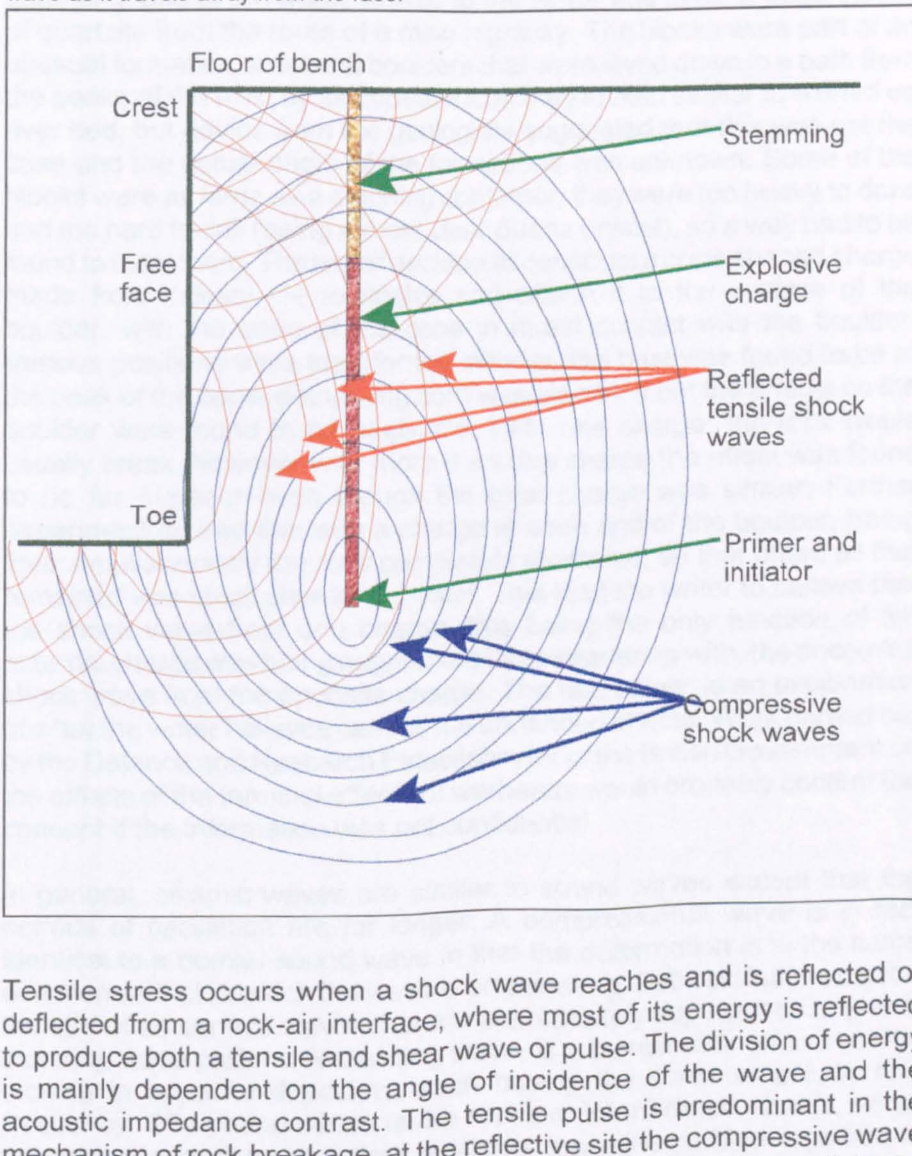
Or more simply, the energy per unit area varies inversely as to the square of the distance from the source. However, in accord with the law of conservation of energy, the total energy of the wave remains the same even though the wave is spread over a greater area. The area of a sphere is calculated thus; $4\pi r^2$ or four times *pi*, times the radius squared. If the rock is fractured, each fracture plane that the wave encounters will provide a reflective surface that will disrupt the main wave. An assumption can be made, that if the explosive is in contact with a reasonably solid medium such as hard rock, the shock wave will continue to travel through the rock until either it naturally degrades or it reaches an interface into which the shock wave is unable to travel.

Hard rock, exhibits excellent seismic wave transmission characteristics as opposed to air which has very poor transmission. When a compressive wave reaches a free face, little of the wave energy is transferred to the air and most of the wave is reflected back into the rock as rarefaction or tensile waves. These waves will be travelling slower than the original and will be subject to the decay characteristics described above. Providing that the reflected energy is of sufficient magnitude, the wave will, through tensile stress, begin on its return path to fracture the rock. To break rock effectively, it is important that the speed of the reflected wave is faster than the natural resonance or velocity of the rock. Generally, the harder the rock, the faster is its natural resonance or sonic velocity, most shotfirers will appreciate that ANFO initiated at low detonation regimes will not satisfactorily fracture rocks such as granite or diorite, even though the powder factor or blasting ratio is high, instead, the gas pressure will dislodge boulders from their natural bedding planes. Intact, hard dense limestone should have a natural seismic velocity of around 3,800 m/sec.

As rock possesses great compressive strength it is very difficult to break with a compressive force (the shock wave). However, rock can be broken relatively easily by the application of tensile stress, (probably requiring between 5% and 10% of the compressive energy).

Figure 11-5 [ref Mills] The advance of shock waves from a blast hole

An approximation of compressive and tensile shock waves generated from a typical quarry blast using bottom initiation. With a column charge of ANFO in a 75mm diameter hole and a burden of 2.5 metres, the time for the reflected wave to reach the bore hole would be a little over one millisecond from initiation. The compressive wave energy that is not directed towards a free face will eventually diminish into undesirable ground vibration. Extensive cracking around the blast hole will disrupt the travel of the shock wave as it travels away from the face.



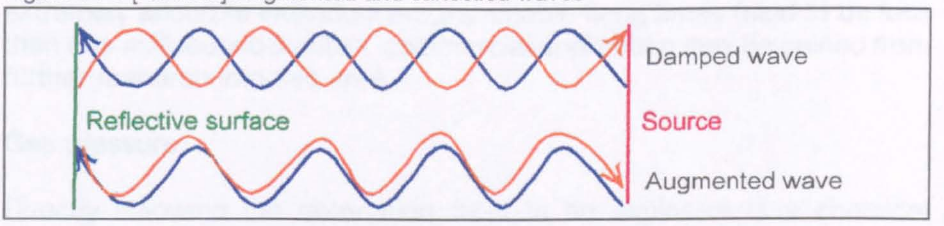
Tensile stress occurs when a shock wave reaches and is reflected or deflected from a rock-air interface, where most of its energy is reflected to produce both a tensile and shear wave or pulse. The division of energy is mainly dependent on the angle of incidence of the wave and the acoustic impedance contrast. The tensile pulse is predominant in the mechanism of rock breakage, at the reflective site the compressive wave and the rarefaction waves compensate each other, whereas, at a distance from the reflective source and at a suitable instant of time, the rarefaction waves can be augmented by reflections and the negative phase of the compressive wave, this can result in a net increase to the stress inflicted upon the rock.

Shock wave augmentation (standing wave)

The writer first noticed the effects of augmenting the shock wave when carrying out massive blasts in a strip coal mine in southern Zambia, determined to discover more about the effects, the opportunity was taken when clearing material for a road construction in the Falkland Islands to experiment with shaped charges and directional blasting. Other than basic quarrying, one of the works tasked to the writer was to clear huge blocks of quartzite from the route of a main highway. The blocks were part of an unusual formation of cubical boulders that were layed down in a path from the peaks of the hills, by all appearances they looked similar to a dried up river bed, but advice from the geologists suggested that this was not the case and the actual origin of the formations was unknown. Some of the blocks were as large as a shipping container, they were too heavy to doze and too hard to drill (being almost clear quartz crystal), so a way had to be found to blast them. The writer decided to construct a cone shaped charge made from military C4 explosive and attach it to the surface of the boulder, with the base of the cone in direct contact with the boulder. Various positions were tried for the initiater, the best was found to be at the peak of the cone, detonating cord was also tried but the effects on the boulder were found to be negligible. With one charge, the rock would usually break, however with more than one charge the effect was found to be far superior even though the total charge was similar. Further experiment proved that with a charge at each end of the boulder, being fired simultaneously the rock completely shattered, so that often, all that remained was small shards of crystal. This lead the writer to believe that the shock wave from one charge, this being the only function of the external charge was being augmented by or interfering with, the oncoming shock wave from the opposite charge. The text below, is an explanation of what the writer believes causes the enhanced effects. Work carried out by the Defence and Research Establishment of the British Government on the effects of the terminal effects of warheads would probably confirm the concept if the information was not confidential.

In general, seismic waves are similar to sound waves except that the periods of oscillation are far longer. A compressional wave is in fact identical to a normal sound wave in that the deformation is in the same direction as that in which the wave is propagating, free oscillations can be thought of as surface waves that are appropriately superposed to give a standing wave pattern. A standing wave, is a combination of two waves moving in opposite directions, each having the same amplitude and frequency. The phenomenon is the result of interference, that is, when waves are superimposed, their energies are either added together or cancelled out. In the case of waves moving in the same direction, interference produces a travelling wave; for oppositely moving waves, interference produces an oscillating wave fixed in space. This concept is explained here in an analogy with water in a container, for example, if a tank containing water is struck on one side, a preliminary ripple will travel across the surface, (this represents the compressive wave from a detonation) when it reaches the opposite side of the tank, the ripple will bounce back, if these returning ripples (or tensile waves) are in opposite phase to the original ripples, they will cancel each other out, (a concept used for reducing sound nuisance, whereby a source of noise will have the same noise projected back to it, thus cancelling it out). Should the

Figure 11-6 [ref Mills] Augmented and cancelled waves



returning ripples be in phase with the preliminary ripple, augmentation can take place and the resultant amplitude of the two waves will be the sum of their individual amplitudes. When designing a blast, interesting results can be obtained by experimenting with timing of the detonations. With the correct choice of delays, spacings and burdens, it is possible to focus or augment the effect of the shock waves by using the principle described above, whereby, the direction and the intensity of the wave can be altered by calculating the position of the shock wave at a particular time and firing the next charge to cause a surge or amplification.

In practice, by considering the stemming burden and timing of the blast, the operator is attempting to provide the optimum environment for the wave to reach the appropriate free face and make use of its reflection. Perhaps, because many shotfirers are not aware of the significance of the shock wave, improved results can often be obtained, mainly by causing interference in the pulses and by bottom initiation. With bottom initiation the detonation of the explosive column takes place at the base of the explosive column and the shock wave travels upwards to be reflected and changed to a tensile wave by the interfaces of the bench floor and the quarry face. The interrelationship of the resulting reflected waves together with those from adjacent holes can be made to produce enhanced effects, resulting in improved fragmentation. Initiation made at the top of the explosive column will cause the a large proportion of the shock wave to travel downwards into the earth thereby losing useful energy and promoting undesirable ground vibrations.

For military purposes and scientific interest this process has been effectively researched and extremely complex computer models developed, For military applications, the British, Defence Establishment and Research Agency use a powerful range of hydrocodes, however, the writer believes, that when blasting a rock containing thousands of fractures and other planes of weakness, differing characteristics such as with changes in structure and density, it is not possible to achieve a consistently accurate result. The accuracy of civilian detonators is also inconsistent and current detonators can have a scatter time of up to ten milliseconds (the difference in time of operation between two identical devices) and because of this, they are not accurate enough to consider using for the modification of shock waves as these may only exist for a useful period of less than five milliseconds. It may be, that with the recent introduction of computed controlled detonators that are said to have extremely accurate individual programmable firing times (said to be less than one millisecond scatter), commercial application may be gained from further research into this area.

Gas pressure

Directly following the detonation front in an explosive is a chemical conversion of explosive products to high temperatures and high pressure gases. If the stemming is poor, or if top initiation is employed, then a significant percentage of the gases will vent to the atmosphere, thus reducing the potential work energy by lowering the confinement pressure, and additionally, causing environmental nuisance. With good confinement a resilient rock may have sufficient cohesion to retain the gases for up to 50 milliseconds. Providing that the gases are suitably contained, a second stage of breakage will occur when the high pressure gasses migrate into the fractures joints and faults, widening them and causing further cracks and breakage. Once the gases have reached sufficient pressure to equal the mass of the rock, movement will take place. Mass movement will usually begin some ten milliseconds after detonation (using a 2.5 metre burden the shock wave will have shattered the rock within one to three milliseconds). The rock mass will be heaved towards the area of least resistance with speeds of up to seventy millimetres per millisecond [ref 27]. The rocks with a greater mass will be projected farther and because of this, are more likely to be involved in mid air collisions, both increasing fragmentation and removing the kinetic energy from the projectiles. Badly applied stemming can be projected from the collar of the hole at velocities reaching 400 metres per second (half the velocity of a bullet fired from a pistol). The third stage of breakage is caused by mid air collisions, these are particularly effective when an offset deep chevron pattern is applied.

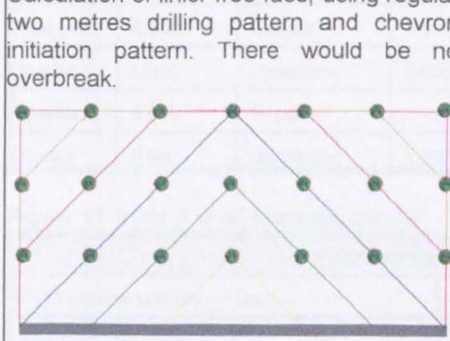
When considering the timing of the rows of a blast it is important to take into account the time required for the gas to move the rock and provide a free face for the subsequent row. It should be remembered that when the rock leaves the bench it will acquire a "swell factor" in simpler terms, the rock will be loose and less compact, generally it will now occupy a space 1.5 times its original, therefore, the space required for the blasted rock will need to be at least one and a half times larger than the original bench dimension. As stated earlier, the breaking mechanism will be completed in less than three milliseconds and the rock will begin to move after some 10 milliseconds, leaving the site at say 70 millimetres per millisecond, thus the new free face will begin to be formed, when using a 2.5 metre burden approximately 13 milliseconds after initiation. Using an inter-row delay of 17 milliseconds the rock may have moved 280mm before the next initiation and after 25 milliseconds it may have moved 840mm and after 47 milliseconds it may have moved 2,380 mm. Without the use of either high speed photography or sophisticated electronic sensors it is not possible to determine the actual speeds of effect in a blast, however as there are currently only three delay periods available, some experimentation will soon prove which is the most effective period, further modifications can be made by adjusting the burden. As long as the rock has cleared the new free face the next row can be successfully initiated.

Aspect ratio

This is a term used by the writer to describe the ratio of free face, to depth of the blast (depth meaning the distance from the free face to the last row of blast holes, not to be confused with bench height). A blast that has been designed to have 10 holes across the length of the free face and is ten rows deep will have an aspect ratio of 1:1. It is convenient to work with the face always being the first number, for example, a similar blast of ten holes across the free face and five rows will have an aspect ratio of 2:1 and a blast that has only one row will have an aspect ratio of 10:1. This figure is import with regard to making enough room for the rock pile and ensuring that a clear free face is available for the subsequent row of blast holes. As previously explained, the blasted rock will have a bulking or swell factor of 50%, it is therefore, important to examine the detail of the blast and ensure that there is sufficient space available in front of the blast to for the rock mass to expand without being choked.

Figure 11-7 [ref Mills] Various aspect ratios

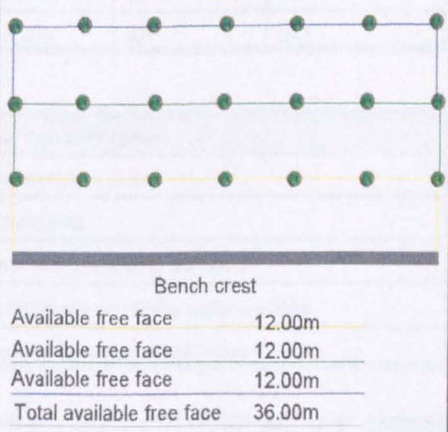
Calculation of linier free face, using regular two metres drilling pattern and chevron initiation pattern. There would be no overbreak.



Bench crest

Available free face	2.00m
Available free face	9.66m
Available free face	15.32m
Available free face	16.58m
Available free face	11.32m
Available free face	5.60m
Total available free face	60.48m

Calculation of linier free face, using regular two metres drilling pattern and straight line initiation pattern. Overbreak on the sides of the blast would possibly make another two metres per line available



Bench crest

Available free face	12.00m
Available free face	12.00m
Available free face	12.00m
Total available free face	36.00m

An option preferred by the writer is to arrange that the charge is powerful enough to throw or heave the rock upwards and by doing so, take advantage of both the free face in front of the blast and the free face of the previous surface. Due to environmental reasons, this option is unlikely to be available within the UK. The initiation pattern must be considered when calculating the aspect ratio, this is because a bench drilled in a regular rectangular format will have a different aspect ratio when fired with a chevron pattern, than when it is fired if straight lines. One way to compare the aspect ratios of various patterns is to calculate the total length of the various advancing free faces, for example see the drawings on the previous page; The drawing on the left shows an available free face of 60.48 metres, this blast will be effective and it will leave the corners and sides of the blast clean with no overbreak. The drawing on the right shows an available free face of 36.00 metres, this blast will not be effective, the corners will be tight, the ends of each line, particularly the front will be ragged and there will be overbreak to the rear.

Peak particle velocity.

Assuming that the rock is consistent, a high velocity explosive will cause more fragmentation than a low velocity one and providing the cracking is the same, an explosive that produces a high volume of gas will create more secondary and tertiary breaking. A rock with a seismic velocity of 6,000 m/s or more is considered to be high and 2,000 m/s or less is regarded as too low for effective blasting. Rock that has a seismic velocity of less than 1,850 m/s can usually be ripped. Bauer and Calker identify a velocity of 2,500 m/s as being the threshold for rock breakage.

Figure 11-8 [5 et al] Seismic velocities

Typical seismic velocities (m/s) of various materials					
Material	Velocity	Material	Velocity	Material	Velocity
Gabbro	6,500	Granite	4,000 to 6,000	Water	1,500
Diorite	5,800	Limestone	3,000 to 6,100	Clay	1,100 to 2,500
Marble	5,800	Gypsum	2,100 to 3,600	loose sand	1,400
Basalt	5,600	Sandstone	1,200 to 4,300	Air	341

Figure 11-9 [ref 5 et al] Damage criteria

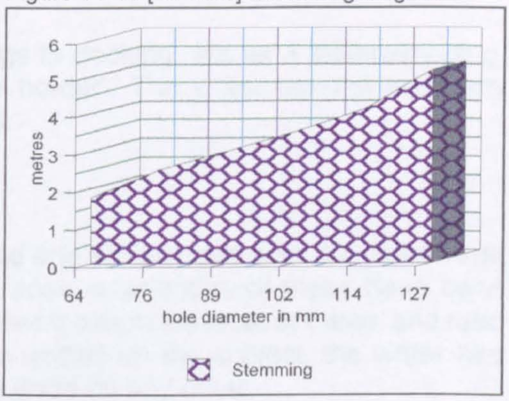
Damage criteria for in situ rock mass	
Peak particle velocity mm/sec	effects on rock mass
less than 250	no fracturing
250 to 625	minor tensile slabbing will occur
625 to 2,500	strong tensile and some radial cracking
more than 2,500	complete break up of rock mass

Most commercial high explosives have a velocity of detonation in excess of 3,000 m/s. The general term used for describing velocity in terms of blasting is 'Peak Particle Velocity (PPV) this is a calculated from V = a (D √W)b where V = PPV in mm/sec, D + distance of measuring point from blast, W = instantaneous charge, a and b are variable site conditions.

11.1.3 Stemming

Stemming can be defined as the inert material used as filler in blast holes. Although seemingly unimportant, by retaining the expanding gases, stemming has a significant effect on the efficiency of a blast. The early release of gases can lead to the production of excessive toxic fumes, noise and dust. Typically, the shotfirer will use the drill cuttings from the hole as stemming, these are usually composed of fine powder and have little ability to bind and lock within the hole. A more suitable material is a mixed 20 millimetre aggregate, this will interlock and form an effective seal. A typical ratio for the calculation of stemming is that the length should be equal to between 30 and 33 times the diameter of the drill hole. Experimentation will allow the shotfirer to refine the calculation, to include external factors which may require that the ratio is modified slightly. The chart shows typical stemming lengths, these have been modified from the 30 or 33 times rule and are thought to be more realistic to actual operating conditions. When considering stemming, the following points may be considered;

Figure 11-10 [ref Mills] Stemming lengths



- increasing the stemming beyond the calculated figure has little effect
- stemming is independent of the charge length or bench height
- 10 gramme or more, detonating cord will destroy the stemming
- top initiation blows out stemming
- explosives with faster VOD require less stemming and vise versa

The most common stemming material is cuttings from the drill, these are virtually useless and often blow out of the hole, as in the case shown in the picture. A more effective material is 20mm crushed rock, this can lock in the hole and form an effective seal. The writer has found that water contained in plastic bags makes excellent stemming, however this can only be considered when using a waterproof explosive. Recent innovations in the use of air blocks and inverted cones are being put forward as improvements on crushed stone stemming, the inverted cones are said by their manufacturers to "allow better control of flyrock, and air blasts, improve spread patterns to give same fragmentation with less drilling thereby reducing drilling costs, improve profitability by increasing safety and reducing insurance claims, replace air bags in decking, act as a stemming plug, temporary hole plug and charge holder". The writer has not seen any evidence to support these claims.

Figure 11-11 [ref Mills] Poor stemming.

11.1.4 Blasting ratio

Various methods both complicated and simple have over the years been suggested to calculate blasting ratios, a selection of these have been included earlier in this chapter. Having examined most of these, and read almost every book that has been written on the subject, the writer has found the following table to be as good as any other.

Figure 11-12 [ref Mills et al] Typical blasting ratio's

Scale of rock hardness with average blasting ratio's				
Degree of hardness	Rock type	Protodiakonov number	ANFO per BCM	Gelignite per BCM
Exceptionally hard	Dense and tough quartz and basalts other exceptionally hard rocks	20	1.3	1.0
Very hard	Granites, silicified schists, quartzites hardest sandstone and limestones	15	1.05	0.85
Hard	Granitic rocks, very hard sandstones & limestones, quartz ore veins, softer basalts	10	0.80	0.69
Less hard	Hard limestones & sandstones, softer granites, hard marbles, dolomite, gneiss	8	0.60	0.55
Fairly hard	Limestone, sandstones & hard dolomites	6	0.40	0.40
Hard	Sandy schists, slaty sandstones	5	0.31	0.31
Medium	Hard shale, softer sandstone & limestone	4	0.24	0.24
Less than medium	Soft shales, dense marl	3	0.15	0.15
Fairly soft	Shale, anthracite, rock salt, gypsum	2	0.10	
Soft	Coal, shale, hard clay	1		

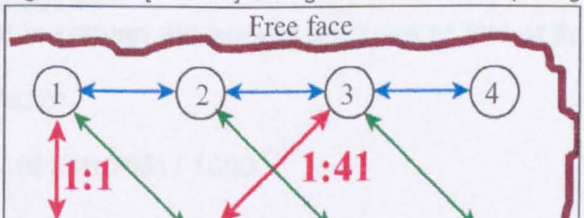
Because each site is different and every blast is changed from the last one, the only sure method is experimentation and even that only works when the rock is consistent. Generally though, it can be expected that three blasts will provide the blaster with sufficient information to arrive at the best blasting ratio. The first blast may for safety reasons be undercharged, after that the ratio may be increased until a suitable relationship between cost and fragmentation is arrived at. The table above was produced by Protodiakonov and modified by the writer.

11.1.5 Calculation of Burden, Spacing and Bench Height

One of the main criteria to be considered in blasting is the cost factor whereby it costs a lot of money to produce drill holes, it should, therefore, follow that this exercise is cost effective. Leaving gaps in the explosive column to attain the correct blasting ratio just means that the specifications have been badly prepared. To be cost effective (and providing there are no outstanding reasons), such as a reduced burden or for geological anomalies, the hole should be filled with explosives from the bottom to the base of the stemming. A common method of calculating the burden, although not very scientific, is to use the same number of metres for burden as the drill hole is in inches, for example, a three-inch drill hole would dictate a burden of three metres. An empirical law of blasting states that the burden is always between thirty and forty times the diameter of the borehole. Multiplying the diameter of the drill hole by forty for the first blast will provide an element of safety and corrections can be made on any subsequent blasts.

Some authoritative bodies believe that to achieve the most effective fragmentation it is usual for the spacing to equal or extend up to, 1:5 times the burden. The writer has yet to see evidence of the effectiveness of this belief and is of the firm opinion that increasing spacing only causes less breakage at the toe and a greater variance in fragmentation. It should also be borne in mind that if the initiation sequence is anything but in straight lines, the ratio of burden to spacing will be altered. For example, the holes shown on the drawing are spaced equally, but if the holes were fired with the front row (1234) first, followed by the second (5678) row, (blue lines) the burden and spacing would be 1:1, however, if the holes were fired at an angle (green lines) with (4) as the first line, (3 and 8) as the second line, (2 and 7) as the third line, (1 and 6) as the fourth line and ending with (5), the ratio would

Figure 11-13 [ref Mills] Changes in burden and spacing



be 1:1.41. If this concept is not understood, it is possible to off-set the holes and then adjust the firing order and end up with a very small and perhaps dangerous hole loading, because of this the writer recommends a square drilling pattern with the burden and spacing being equal. An empirical, circular argument states that; the drill hole diameter dictates the burden relative to the height of the bench which should be 2.5 to 5 times the burden; That is,

- a two-inch drill hole requires a two-metre burden and a 5 to 10.0 metre bench
- a three-inch drill hole requires a three-metre burden and a 7.5 to 15 metre bench
- a four-inch drill hole requires a four-metre burden and a 10 to 20 metre bench

It can be said that, within reason, the greater the bench height, the more cost effective is the blast. A further complication when calculating burden and spacings is that all explosives, to a greater or lesser degree produce more relative energy with an increase in hole diameter. Therefore, per gramme, explosives in a 200 mm hole will be more effective than the same explosives in a102 mm hole. Because of this increase in energy, a fixed ratio of explosive to rock cannot be used, and consideration will have to be given to this effect when changing drill hole sizes.

Mass of explosives

Before calculating the blasting ratio, the volume of the drill hole must be established, this is required in order to determine the amount of explosives which will actually fit into the hole. The following is an easy to use formula for calculating the volume of one metre of drill hole; where

- V = volume in litres
- d = hole diameter in millimetres
- E = explosive charge in kilograms
- D = density of the explosive
- 0.785 = the constant to convert the area of a square to that of its inscribed circle
- 1000 = conversion factor

method	V =((d x d).785) / 1000
eg	V = 76 x 76 x.785 / 1000 V = 4.53416 litres rounded off to 4.5 litres

To convert the volume to the amount of ANFO which can be loaded per metre of blast hole simply multiply by 0.8 this being the loose density of ANFO.

eg 4.5 x 0.8 = 3.6 kg

For a seven metre face with 0.5 metres of sub drilling and 2.5 metres of stemming, 5.2 metres of the hole is available for explosives. Therefore the column charge is; 5.2 x 3.6 = 18.00 kg of explosive.

Burden and Spacing

If the mass of the explosives is divided by the blasting ratio, the total amount of rock can be calculated ;

- volume = 18 / 0.4
- volume = 45.0 cubic metres of rock

next, divide the volume by the total length of the drill hole excluding the sub drilling but including the stemming.

- volume / length
- 45.0 / 7 = 6.43

then, find the square root of the result.

- $\sqrt{6.8} = 2.53$

A typical, and ideal, burden and spacing for the example shown above is 2.53 metres.

11.1.6 Cost of blasting

As the cost of explosives in Africa rapidly fluctuates, the following costs are based on the price of explosives in the UK (1998). The calculations consider a small blast of twenty holes, this being the maximum that can be fired using electric delay detonators. The following is an approximation of the cost of explosives and accessaries in the UK, priced ex works.

Electrical exploder	£ 400.00
Electric delay detonators (each)	£ 8.73
Connecting wire (reel)	£ 6.40
Trunk wire (100 metres)	£ 15.00
Nonel underground detonators (each)	£ 4.50
Nonel surface detonators (each)	£ 2.34
Detonating cord ten gramme (1,000 m)	£ 655.00
Detonating cord forty gramme (1,000 m)	£ 995.00
Ammonium nitrate (1,000 kg)	£1,000.00
Emulsion, Delta plus (1,000 kg)	£2,500.00
NG explosives (1,000 kg)	£2,680.00
Pentalite 800 gramme (each)	£ 7.50
Fuel oil (litre)	£ 0.60

Electrical system

This system requires the use of an exploder, trunk wire, forty detonators and one reel of connecting wire @

Connecting wire	£ 6.40	£ 6.40
Detonators	£ 8.73 x 40 =	£ 349.20
Pentalite boosters	£ 7.50 x 20 =	£ 150.00
80 kg Anfo per hole	£ 80 x 20 =	£ 1,600.00
5 litres of diesel fuel	$ 3.00	£ 3.00
TOTAL		£ 2,108.60

20 holes would produce (at a density of 2,3 tonnes per cubic metre) 9,200 tonnes, therefore the cost per tonne is 1,934/9,200 = £00.21 per tonne. Firing would require the use of an exploder and a reel of truck wire at £ £400.00, these items are re-usable.

Chemical system

This system requires the use of an exploder, trunk wire, relays, twenty metres of forty gramme cord per hole, 120 metres of ten gramme cord and one electric detonator.

10 gramme cord	£ 0.65 x 120	£ 78.00
40 gramme cord	£ 0.99 x 400	£ 396.00
Connecting wire	£ 6.40	£ 6.40
40 relays	£ 5.00 x 40	£ 200.00 (estimated as not available in UK)
Detonators	£ 5.00 x 12 =	£ 60.00
80 kg Anfo per hole	£ 80 x 20 =	£ 1,600.00
5 litres of diesel fuel	$ 3.00	£ 3.00
	TOTAL	£ 2,343.00

20 holes would produce (at a density of 2,3 tonnes per cubic metre) 9,200 tonnes, therefore the cost per tonne is 2,343/9,200 = £00.25 per tonne Firing would require the use of an exploder and a reel of truck wire at £ £400.00, these items are re-usable.

NONEL

This system requires the use of an exploder, trunk wire, forty down the hole detonators, twenty surface detonators one electrical detonator, twenty pentalite boosters and one reel of connecting wire @

Surface detonators	£ 2.34 x 20 =	£ 46.80
Down the hole dets	£ 4.50 x 40=	£ 180.00
Connecting wire	£ 6.40	£ 6.40
Detonators	£ 8.73 x 1 =	£ 8.73

Pentalite boosters	£ 7.50 x 20 =	£ 150.00
80 kg Anfo per hole	£ 80 x 20 =	£ 1,600.00
5 litres of diesel fuel	$ 3.00	£ 3.00
	TOTAL	£ 1,994.00

20 holes would produce (at a density of 2,3 tonnes per cubic metre) 9,200 tonnes, therefore the cost per tonne is 1,994/9,200 = £00.22 per tonne. Firing would require the use of an exploder and a reel of truck wire at £ £400.00, these items are re-usable.

Summary of costs

The cost of blasting 9,200 tonnes by the methods described above, per tonne are ;

Electrical	£ 00.21 x 2 = 00.42 in US$ 00.70
NONEL	£ 00.22 x 2 = 00.44 in US$ 00.73
Chemical	£ 00.25 x 2 = 00.50 in US$ 00.83
The cost per mined tonne	US$ 00.73.

11.2.0 STORAGE AND DESTRUCTION OF EXPLOSIVES

11.2.1 Explosives store

From the time that explosives leave the care of the manufacturer and until the time of use, the method of storage must be rigidly enforced. The site of the store must be positioned at an approved distance from habitation or areas where the public have access. A general distance from habitation, railways, public places and highways is provided in the British Explosives Order 1951, the distances shown below do not require the permission of the public. The store should also be positioned in a site where it will not be damaged from fly rock or ground vibration.

Explosives stores must be secure, there are two schools of thought on this,

Very secure

- Steel or concrete structure very resistance to unapproved entry, these are generally of massive construction

Secure enough

- Moderate steel construction encased in a concrete building with modern security systems

The main problem with being "very secure" is that, should there be an explosion, the blast will be slightly contained and the store will become, in effect, the casing of a bomb, with the likely effect of pieces of shrapnel travelling long distances at high speed. A more lightly constructed store will be destroyed at the beginning of an explosion and the reduced mass is not likely to fly as far. It should be remembered that few magazines will contain an explosion of more than a few kilograms of high explosive.

Provided that the store is secure from theft, it should be good enough, In a typical quarry, an average store could be ripped apart in a few seconds by the use of an excavator, regardless of its construction. Generally, the store should have a steel door with either internal locking bolts or internal hinged and mortise locks, two, high security five or more lever locks, and a separate weakened locking handle (many doors are ripped from their hinges by the application of a truck and chain to the locking handle as this is often the strongest part of the door.

Figure 11-14 [ref HMSO] Quantities and safe distances

Store type	Quantity	distance
A	75 kg	25 metres
B	150 kg	40 metres
C	450 kg	90 metres
D	900 kg	140 metres
E	1,800 kg	210 metres

The store should contain only explosives, and these stored in separate units according to type. Store licenses should be posted immediately inside the door of each store and the store rules should also be conspicuously posted at each store. The operation of the store must be directed by a competent person who appreciates all phases of the operation and all applicable regulations.

Every store must be inspected at intervals no greater than every three days to determine whether there have been any attempted or unauthorized entries. Any thefts or attempted thefts should be reported immediately to the proper authorities. Records must be maintained so that all explosives received can be traced. Doors should be kept closed and locked except when materials are being handled and during inspections. Keys should be carefully controlled and held only by competent persons. Sites must be posted with warning signs reading "Explosives - Keep Off." Smoking or carrying matches or cigarette lighters must not be permitted in or around magazines or trucks containing explosives. The stores should be used exclusively for storing explosive materials, detonators, and the equipment required to operate the stores. Any ferrous equipment should be covered with protective paint. Floors should be clean and should be swept frequently and the sweepings removed to a safe place and burned.

Detonator store

The detonator store can contain; detonating caps, detonating relays, electric detonators, safety fuze and igniter cord. Some bodies allow the storage of the firing device or equipment. The contents of the store must be protected from, fire, flame, shock, friction, and extraneous electricity either as current or radio waves.

Rules for a detonator store

1. Only detonators are to be kept in this store, i.e., blasting caps, electric detonators, safety fuse and nonexplosive blasting supplies. Ferrous tools and implements should not be kept.

2. Packages containing detonators should be handled carefully and not dropped or thrown down. They must not be slid along the floor or over each other, or handled roughly in any manner.

3. Metal bail hooks must not be used.

4. Corresponding grades and brands should be stored together in such manner that brand and grade marks are visible, this to allow them to be easily counted and checked and the oldest stocks readily seen.

5. The oldest stocks should be piled to the front of the store and used first.

6. The opening of packages or packing or re-packing of detonators should not take place within a distance of 15 metres of the store.

7. Extreme care must be used in opening or closing packages of safety and electric detonators.

8. Ferrous tools must not be used to open or close packages of detonators.

9. Loose safety or electric detonators should not be kept in the store, they must not be taken out of their original package until required, then the package should be closed.

10. If artificial light is needed, only a safety flashlight or other approved source of light should be used.

11. Smoking should be prohibited within 15 metres of the store and matches, lighters or other flame producing devices are proscribed.

12. Shooting should be prohibited and firearms or cartridges proscribed

in or near the store.

13. The interior of the store must be kept clean, and the surrounding area clear of dry leaves, grass, undergrowth, trash, and debris.

14. The store must be maintained, if a leak develops in the roof or walls it must be repaired at once.

15. No unauthorized persons should be allowed in or near the store.

16. The doors should be kept closed and securely locked.

Rules for an explosives store

The explosives store can contain; mixed ammonium nitrate and fuel oil, Nitroglycerine type explosives, water gel explosives, emulsion types explosives, cast and other types of primers and detonating cord. The contents of the store must be protected from, fire, flame, shock, and friction, high explosives are not sensitive to radio waves. This store is not to contain; any form of detonator, gunpowder, ferrous tools or other sparking metal implements in this magazine. A full inventory of the contents of the store must be kept at all times.

1. Packages should be handled carefully, not dropped or throw down, or slid them along the floor or over each other, or handled roughly in any manner.

2. All explosives and detonators must be stored in closed boxes, which should be put in upright and stable stacks. Damaged explosives should not be repackaged within 15 metres of a store or other explosive materials.

3. Primed cartridges should not be made up or kept in a magazine with other explosives.

4. When a new shipment is received, stocks should be arranged so that older stock will be used first. Corresponding grades and sizes should be stored together, and the cases arranged so that the grade and size markings are visible for easy counting and checking.

5. No packages are to be opened or sealed within a distance of 15 metres of the store. The packages of explosives must be stored at least 100mm away from the walls of the store.

6. No loose explosives or open packages of explosives should be kept in the store. All packages should be sealed.

7. If artificial light is needed, use only a safety flashlight or approved light.

8. No smoking or carrying of matches, lighters or other flame producing devices within 15 metres of the store.

9. The interior of the store to be kept clean, and the area around store clear of dry leaves, grass, undergrowth, trash and debris to prevent fire.

10. Store to be kept in good repair.

11. No unauthorized persons to be allowed within 15 metres of the store.

12. The door of the store is to be closed and securely locked except when opened for transacting business.

13. Boxes that have contained explosives should be burned and never reused.

11.2.2 General

The rules and legislation vary in each country, for example, in Zambia the rules are virtually all written to apply to the use of explosives underground, in the UK and its European neighbours the rules have become daunting and ever changing, in Malawi there are virtually no rules and those that they have are not enforced, in Jamaica the rules and legislation describe gunpowder, shipping and harbour masters. The United Nations has provided explosives with a hazard classification or division number, in this case, the greater the number the lower the transportation risk. The number can be followed by a letter that further defines the goods.

Figure 11-15 [ref 46 , UN] United Nations hazard numbers

Divisions	division number
Substances and articles which have a mass explosion hazard	1.1
Substances and articles which have a projection hazard but not a mass explosion hazard	1.2
Substances and articles which have a fire hazard and either a minor blast hazard or a minor projection hazard or both, but not a mass explosion hazard	1.3
Substances and articles which present no significant hazard	1.4
Very insensitive substances which have a mass explosion hazard	1.5
Extremely insensitive and with no mass explosion hazard	1.6

The United Nations has also issued a table of compatibility groups;

Figure 11-16 [ref46] UN compatibility groups

Description	UN number	Examples
Explosives blasting type A	0081	Nitro-glycerine based explosives
Explosives blasting type B	0031 0082	Factory made ANFO
Explosives blasting type D	0084	Plastic explosives
Explosives blasting type E	0241 0332	Cartridged water gels and emulsions
Detonators electric for blasting	0030 0255 0456	Electric instantaneous, short delay and delay detonators
Detonators non-electric for blasting	0029 0267 0455	Plain detonators, capped fuses and detonating relays
Detonator assemblies, non-electric for blasting	0360 0361	Shock tube detonators and connectors
Cord detonating, flexible	0065 0289	Detonating cord
Boosters	0042	Cast primers (pentolites)
Blackpowder	0027 0028	Gunpowder, fuse cord

Explosives and detonators are safe to handle when in good condition and treated with great care, However, they are perishable goods and will deteriorate if kept in unsatisfactory conditions. Should this happen, it may become necessary to destroy them and in this case, expert assistance is positively recommended. When stored in suitable conditions, the following are recommended;

Figure 11-17 [ref 46] Shelf life

Use by times	
Nitroglycerine based explosives	One year
Water gel type explosives	One year
Emulsion type explosives	One year
ANFO	Six months
Detonating cord (PETN)	Several years
Cast primers (PETN and TNT)	Several years
Delay detonators	Eighteen months
Instantaneous detonators	Two years
Sealed shock tubes	Up to two years

Explosives that require disposal may be fresh material from containers broken during transportation, surplus material for which here is no further need, or they may consist of material which has deteriorated or which has become unfit for use through some sort of damage. Deteriorated or damaged explosives may be more hazardous than those in good condition and, hence, require special care in handling and disposal. Explosive stocks should always be rotated in the magazine so that older material is used first. When properly stored and cared for, commercial explosives, blasting agents and blasting accessories will remain in good condition for a considerable time. Deterioration will occur after prolonged storage, particularly under conditions of high moisture and high temperature or if the materials are improperly treated, in which case, they may deteriorate rapidly.

The absorption of moisture is the principal reason for the deterioration of explosives and blasting agents. They contain certain ingredients which are hygroscopic and moisture is consequently absorbed if they are exposed to dampness or excessive humidity during storage. Moisture generally reduces the strength and sensitivity of explosives and blasting agents and in the case of explosives also causes exudation of the liquid ingredients and eventually renders the material unfit for use.

Moisture is also the chief cause for deterioration of most blasting accessories, particularly blasting caps, safety fuse and delay elements of Detonating Relays. Prolonged exposure to extreme moisture or high atmospheric humidity will reduce the efficiency of most blasting accessories and in some cases, may result in these materials becoming hazardous to handle. It is the responsibility of every explosives user to destroy such explosives and blasting accessories found on his operation to be unfit for use or to have deteriorated into a dangerous condition. He must never, under any circumstances, abandon these materials either by burying them or by any other means.

11.2.3 Nitro glycerine based explosives

Gelignite explosives will deteriorate take a state of increasing danger and will rapidly become unstable. The first signs of deterioration are apparent when the cartridges become soft and damp, this is often followed by a strong smell of marzipan. White crystals may also appear on the outside of the cartridge. Explosives that have absorbed an excessive amount of moisture are usually soft and mushy. The cartridge paper has a greyish or whitish appearance and crystals of sodium nitrate or ammonium nitrate are sometimes found on the outside of the cartridges. Cartridges that have been soaked in water are usually swollen.

Exudation of nitroglycerin from gelatin type explosives is evidenced by the formation of small beads of clear oily liquid at the ends of the cartridges or by an oily film appearing on the surface of the wrappers. In cases of pronounced exudation, nitroglycerin may have leaked from the cartridges into the case and been absorbed by the sawdust packing or collected in pools in the waxed lining paper; in extreme cases, it may have run out or seeped through the bottom of the case. The hygroscopic salts will draw moisture from the atmosphere and for watery drops or crystals on the outside of the packaging that will have a salty taste. If the explosive is of considerable age, it may be very dark brown, sometimes almost black in colour, and the cases may frequently be discoloured by brown stains. In some instances, the explosive may fail to detonate and in others, it may burn instead of detonating thereby giving off poisonous fumes. Eventually the remaining gel will break down releasing oily droplets of free nitroglycerine, these may form on the outside of the cartridge. Nitro glycerine based explosives in a deteriorated condition are dangerous to use and should be promptly destroyed.

Handling

A careful examination should be made of the magazine, wearing clean rubbers over the shoes to prevent the possibility of friction from shoe nails or grit. The floor of the magazine should be examined for traces of nitroglycerin and if any spots are found, they should be washed with nitroglycerin destroyer or with a strong solution of sodium or potassium hydroxide. If neither of these materials is readily available, coal oil or kerosene liberally applied to the floor will temporarily desensitize the nitroglycerin sufficiently to allow the floor to be walked on while the explosives are removed from the magazine. The floor must then be washed with nitroglycerin destroyer or caustic sodium or potassium hydroxide as described above. Smoking must be prohibited in the vicinity or within the magazine and rubber gloves should be worn to avoid staining the hands from contact with the cleaning solution.

The stacked cases of explosives should be examined to ensure that none are in danger of falling over; if any cases are found in such a position, they should be propped up carefully until such time as they are to be moved. Every precaution should be taken to avoid friction of any kind when moving the cases. They should be lifted gently from the pile, not slid off. They should then be carried outside of the building and gently set down on a bed of sawdust, straw or other cushioning material. If this operation has to be carried out in cold weather, extra care should be taken to be certain that the cases can be readily moved and are not frozen in place or to each other. No force of any kind should be used in an attempt to move them.

Any cases of explosives showing signs of nitroglycerin stains should be washed with nitroglycerin destroyer or with a strong solution of sodium or potassium hydroxide, if neither of these materials is available, kerosene may be substituted. If the cases are to be taken to the destroying grounds by truck, they should be separated from each other and from the bottom and sides of the truck by means of straw, hay, or other cushioning, fresh material being used for each load. Any truck used for this purpose should be well washed out with nitroglycerin destroyer or with a strong solution of sodium or potassium hydroxide after completion of the work.

There are two commonly used methods for destroying deteriorated explosives; by detonation or burning. They should never be buried as there is always the possibility of the cache being later uncovered. Nitroglycerin is very chemically stable and will retain its sensitivity for years.

NG destroyer

This liquid should be used freely to decompose the nitroglycerin thoroughly at a solution of two gallons for each pound of nitro-glycerin. The remover loses effectiveness as it ages, so that it should be stored no more than 60 days in a closed container in a dark place. If the magazine floor is covered with material impervious to nitroglycerin, the floor should be thoroughly swept with dry sawdust, and the sweepings removed to a safe place and burned. It is not safe to subject them to an open flame or other potential sources of ignition. Adequate ventilation and/or approved respiratory equipment will be required as will, protective gloves and eye protection. The area must be thoroughly ventilated after use until there is no trace of fumes in the area.

Detonation Method.

This is the quickest and simplest method of destroying explosives. However, it has two undesirable features. In certain instances, some cartridges may be so insensitive that they will fail to detonate and will be merely scattered about by the explosion of the others. It also results in considerable noise and concussion to which residents in the vicinity may object.

Careful attention should be given to the choice of a location as far away as possible from damageable property where the ground is free from boulders and small stones which might be blown about by the blast. The amount of explosives to be detonated at any one time depends on various conditions; the proximity to sensitive structures and wild animal retreats, total amount of explosives to be disposed of, means of transportation available. Usually one to four cartridges should be shot first, later increasing the quantity after assessing the noise and concussion with respect to adjacent property and the reaction of the nearest residents. It is not necessary to open the cases as the contents can be exploded by firing a charge of fresh explosives placed in contact with the outside of the cases. This charge should be made up with two cartridges tied together, with one of them primed with a long length of capped fuse or with an electric blasting cap. The fuse should be long enough to give personnel sufficient time to reach a selected place of safety. Care should be taken in lighting the fuse to ensure that the spit does not point toward the cases since this might cause a premature explosion. All approaches to the locality should be guarded.

Destroying by Burning.

It is recommended that not more than 5kg of explosives be burned at one time, unless the person in charge is widely experienced in work of this kind. The choice of location should be such that, in the event of the explosives detonating instead of burning, surrounding property and lives will not be endangered. Cases should be opened only at the site and the contents removed; the possibility of a filled case exploding due to the concentration of heat is always much greater than when the explosives are removed and spread out. Only wood implements should be used to open the cases; metal of any kind for this purpose is highly dangerous.

A bed of combustible material should be prepared on which to place the cartridges. Any dry material such as wood shavings, straw, hay or excelsior is suitable. The bed should be two to three feet wide and of sufficient length that the cartridges may be spread out in one layer without overlapping. The paper lining and sawdust packing can be spread out with the explosives, and the cases burned separately. The whole bed should be drenched with kerosene to assist the burning. An ignition train of combustible material should be made leading up to the explosives, so that when the train is ignited it will burn against the wind. It should be long enough to allow the operator plenty of time to reach a place of safety before the flame reaches the explosives.

A new location should be selected for each lot to be burned as it is not safe to place explosives on hot ground from the preceding burning operation. After the fire has burned out and the ground cooled (it should not be approached under any circumstances before this), the ashes should be raked over with a wooden rake to make sure that no unburned explosive remains. If a fly is found, it should be burned on a fresh bed of combustible material after a liberal application of kerosene. When explosives are to be burned, great care should be taken to ensure that no cartridges are primed with detonators, as these will explode in the fire and detonate the remaining unburned explosives.

11.2.4 Emulsions, water gels and ANFO

All these explosives decay to a state where they become less likely to detonate. Ammonia is highly soluble in water and may form an alkaline solution called ammonium hydroxide. Ammonium nitrate is highly soluble in water and heating of the water solution decomposes the salt to nitrous oxide (laughing gas), because of this it can easily be destroyed. The sensitivity of ANFO is gradually reduced, when moisture is absorbed, to a point where it cannot be initiated with even the strongest primer. As more moisture is absorbed, it forms a sludge and if sufficient volume of water is present it will dissolve completely, separating from the fuel oil. ANFO may dissolve entirely, or be rendered harmless by exposure to excessive moisture, it can be destroyed by emptying the contents into a large body of water, or alternatively, by emptying it into a pit and pouring water over it. Water gels are similarly affected in the early stages of deterioration. The bulk of the ingredients will eventually dissolve and the gel structure will fail. A residue, perhaps containing aluminium and or crystals of monomethylamine-nitrate (Man $CH_6N_2O_3$) may be left that is mostly harmless. Man has a low sensitivity to impact and its crystals are hygroscopic and can be reduced in water. Any water gels found in a shot hole will eventually become safe, however the detonator and the primer will probably not be so care must be exercised.

Emulsion

The water phase will seperate from the structure, this can be recognised by finding crystals of ammonium nitrate on the outside of the explosive, also the cartridge will become hard and less pliable. The same comments apply to emulsions as they do to water gels found in a shot hole. The first preference for disposal should be that water gels and emulsions are destroyed by detonation and exploded at some suitable location either in the open or covered with earth or sand. If it is not practical to dispose of them by detonation, they can be destroyed by burning, but due to their high water content, considerable difficulty is likely to be encountered. It is strongly recommended that disposal is not carried out unless the person in charge is thoroughly experienced in this type of work.

Water gels

Deteriorated water gels may also exude a liquid, this is mostly water and dissolved salts. In general, to remove this liquid, the magazine floor must be washed with water (and soap if necessary), but always consult the explosives manufacturer for the proper procedure.

11.2.5 Blasting accessories

The appearance of deteriorated blasting accessories will depend upon the product and the conditions to which it has been subjected. In some cases little physical change will be apparent. The shells of Detonators and Relays that have been stored for long periods exposed to moisture or high atmospheric humidity may become discoloured or corroded. Decomposition may have set in and it is advisable that they be destroyed. Detonating cord, Safety Fuse, Hot Wire Lighters, Igniter Cord and Connectors, etc., when exposed to similar conditions should be closely inspected for signs of deterioration and be destroyed if they are not satisfactory for use.

When handling these products every precaution should be taken to avoid friction or shock of any kind when removing boxes of deteriorated blasting caps from the magazine. The boxes should be lifted carefully from the pile or storage shelves and gently set down outside the building on a bed of cushioning material. If this operation has to be carried out in cold weather, extra care should be taken to be certain that the boxes can be readily moved and are not frozen in place or to each other. No force of any kind should be used to free them or used in an attempt to move them and they should be kept separate from each other when transporting them to the place of disposal and protected from shock and friction by means of a bed of suitable cushioning material.

When destroying blasting caps, the most suitable method is to tie them in compact bundles with the charged ends together, then place them in a box or bag and bury them under half a metre of earth; and finally, explode them with a good electric blasting cap or, for still greater assurance, with a cartridge of explosive primed with an electric blasting cap. Never attempt to detonate the caps without burying them as this may introduce some hazard from shrapnel and some might fail to explode and be scattered about by the explosion of the others. The same general procedure should be followed when destroying electric blasting caps, delay electric blasting caps, etc., except that the leg wires must be cut off about 25 mm above each cap. It is not advisable to attempt to cut the wires from more than one cap at a time for reasons of safety. After detonating the caps, the site should be carefully examined before abandoning it to ensure that all detonators have been destroyed.

It is unsafe to throw deteriorated blasting caps or electric blasting caps into streams, wells or ponds, or to bury them, as they might be discovered in some manner and cause serious accidents, even after some years have elapsed. Similarly they should never on any account be abandoned or left in any location in an unlocked magazine. Other blasting accessories should be destroyed by burning. To dispose of detonating cord a bed of combustible material should be prepared and the cord strung out in parallel lines about 15mm apart. Cord should not be burned on the spool. When selecting a site, it should be remembered that the cord is a high explosive and may detonate upon burning. Similar precautions should be observed as are outlined earlier by the destruction of explosives. Safety fuse, hot wire lighters, igniter cord and connectors, etc. should be destroyed by placing moderate quantities on combustible material in the form of a bonfire. The site should be selected so as not to introduce any hazard to nearby property. In the interests of maximum safety, all deteriorated explosives, blasting agents and blasting accessories should be promptly destroyed.

11.3.0 CHILANGA CEMENT WORKS

11.3.1 Introduction

The writer was appointed manager of this and two other quarries in 1996, on arrival the site was found to be virtually non-productive and urgent steps had to be taken to prevent the whole factory from being closed down. A contract had been issued to a company to remove overburden, this was rapidly changed and the plant used to develop a new access road and quarry development. An important part of the new quarry plan would be a method of blending the in situ rock. The past experience of the writer was used to write a mine plan that depended on the removal pre-identified blocks of material each with a mass of about 20,000 tonnes. Advanced principles of blasting were incorporated to allow the material to be efficiently blended prior to loading.

This site consists of a very basic, but interesting quarrying operation, however, notwithstanding this simplicity, by 1995, poor quarrying performance had resulted in the cement works being operated on a reduced throughput. Often, one kiln was intentionally stopped and the other was placed on a schedule of slow running. The kilns required approximately eight hundred tonnes of rock each per day to produce four hundred tonnes each, the remainder being lost on ignition. To achieve full cement production, required a daily output from the quarry, which including losses and wastage amounted to 2,000 tonnes per day, in practice, production of cement was satisfied by a quarry output between 1,000 and 1,500 tonnes. The main problems with quarry production were;

- the inability to supply sufficient rock
- the rock to be of a suitable size
- the rock to be within a measured level of consistency

From the point of view of a blasting engineer, the problems were identified as;

- rapid changes of structure ranging from loose calcitic crystals to hard dense marble with subsequent changes in chemical values
- extensive bedding planes and jointing
- inappropriate bench height reinforced by legislation
- incompatible and worn out primary crusher
- unskilled drillers
- unreliable and inaccurate drilling equipment
- untrained management and staff

Most of the problems were addressed by careful management and a restructuring of the systems, however, blending of the rock involved more significant problems. The most important criteria, as with all cement production was to provide a consistent feed stock, this to be achieved from a variable deposit and in a way that could be sustained. Without blending, the supply of rock to the factory would be too unstable to be usable and without a rock high in calcium carbonate to "sweeten" the lower grades, much of the deposit would be wasted. To enable the whole of the deposit to be exploited and ease the significant problems of blending the feed stock at the cement works, the writer devised and implemented a cost effective method of blasting that caused the rock to be mixed and blended in the blast. This resulted in a supply of rock that remains significantly more homogenous than that previously available. It is believed that the described method of blending the rock is a necessary development in blasting hard rock of variable grade for cement and lime production and unless found somewhere that the writer has not visited, the method of blending rock in situ is unique to Chilanga Quarry.

11.3.2 History

Prior to early 1996 and the arrival on the site of the writer, normal blasting consisted of one or more shots being fired out every day, each with a round of about thirty 76 mm diameter holes. The designed burden and spacing were 2.3 and 2.7 metres respectively and a depth of between three and six metres was aimed for. The actual burden and spacing varied from half to three and a half metres, drilled to a depth of between one and six metres. Charges consisted of a detonator sensitive base charge/primer of ammon gelignite and a column charge of ANFO, sometimes, date expired Magnadet primers were used. Stemming was not considered important and usually the explosive column reached the collar of the drill hole. Initiation was provided by eight gramme detonating cord extending through the ANFO to the primer. Surface timing was achieved using 25 millisecond delay relays positioned between the holes and the holes were coupled by detonating cord, the blast was fired by use of a safety fuse and blasting cap attached to the surface detonating cord.

As could be expected, the results from this type of blasting were very poor, with an average recovery of two to three hundred tonnes, this being less than sufficient to operate one kiln for one day. Because of the imbalance of the burden and spacing versus hole depth, there was little throw or heave and the toes were not broken. The lack of stemming guaranteed massive air overpressure and to be slapped by the wave of overpressure at a viewing distance of one kilometre was truly awesome. The flyrock was outstanding because of, the height achieved, its velocity, and distance travelled, one could estimate that at least one third of the rock went upwards for some considerable distance. Complaints from the surrounding land owners were both expected and received. Because most of the energy was released to the atmosphere, ground vibration was minimal.

The poor results achieved by this standard of blasting, in particular, the uneven floor and tight rock pile caused unacceptable damage to the front end loaders and the dump trucks. The tyre replacement schedule and maintenance costs were particularly appreciated by the manufacturers of the equipment. Poor fragmentation and a high percentage of oversize rock ensured that the primary crusher was virtually unable to operate. The plan and design and method of the blasting was provided by ZCCM who despite being experts in underground mining, have little expertise in surface blasting and was being carried out under the instruction and supervision of the appointed quarry manager, who was a chemist, with no knowledge of quarrying and even less of blasting.

The method of blasting was being implemented in order to conform to a quarry plan produced by senior mining engineers on the staff of ZCCM which called for several small and discrete areas of the site to be quarried simultaneously and the results blended. This type of mining is common within the underground ore industry and falls within ZCCM's own area of expertise. Most of the areas were too small to allow the heavy plant to operate and although showing some originality of thought, the practicality of mining many small sites has little practical value within the surface mining industry where typical heavy machines are more suited to working on a capacious scale.

Failure to keep up with demand and difficulty encountered in digging the blast caused quarry output to fall to a level where demand outstripped supply and all attempts to blend the product fell by the wayside, further exasperating cement production. By 1996 it had become apparent that the method of mining, if sustained, would in all probability cause the demise of the operation and plans had been made to progressively close the plant in favour of developing the factory at Ndola. By the time the writer was employed to redevelop the quarry, there was a marked lack of systematic management directly in place, the workers had become dispirited and all attempts to follow a blasting plan and advance development had ceased.

11.3.3 Developments

After changing ownership, the new operating company decided to replace the quarry manager with a suitably qualified consultant (the writer) and allow the development of the quarry to proceed in a more orthodox manner. The function of the writer at this stage was to assess the existing situation and its plan and implement a schedule for the redevelopment of the quarry. To address the situation, standard principles of traditional bench development were implemented, while at the same time, modern techniques of blasting were introduced, these involved a great deal of experimentation to develop a method of blasting which whilst remaining legally and environmentally acceptable, gave very good fragmentation and blended the various grades of material.

The changes in the blasting were to take place alongside a new development in the quarry and an in house and overseas training programme for quarry personnel. A totally different and more imaginative approach to quarrying was taken than the ZCCM model with regard to blending rock whereby, instead of simultaneously extracting from several discrete areas, a more feasible system was developed, where a block of 20,000 or more tonnes was blasted in a single 200 or more hole shot. The blending was designed to occur in the secondary phase of the blast, that is, in the heave and throw, whereby the rock would be drawn from the sides of the blast mixed with the remainder and heaved and projected outwards from the centre of the blast.

New quarry development was structured around two, seven metre benches, one developed at the current floor level and one sub level, these were to be accurately planned and operated with extended faces, the actual quarrying plan is provided in chapter eight. The purpose of establishing the two benches was to provide a larger vertical area from which limestone could be extracted thus maximising the potential for blending, it would have been better for blending purposes if the bench had been worked at fourteen metres, however, the legislation in Zambia called for a maximum bench height of seven metres (the maximum height that a Caterpillar 988B bucket could reach) and the depth of the holes was regularised at seven point five metres, this including half a metre of sub drilling. Similarly, the principle of working of very long benches provided a better opportunity for blending. Although, initially seeming complicated, the new quarry development provided a suitable vehicle to train the staff in both quarrying and modern blasting techniques.

To establish the optimum burden and spacing and therefore the energy to work ratio, experiments were carried out using a selection of measurements. These ranged from 2.0 to 2.8 metres and it was found that at distances close to 2.0 metres where, because of the greater explosive ratio, it would possibly be expected that fragmentation and throw would be increased, this was not the case. In fact, both fragmentation and throw was reduced. Without having sophisticated high speed photographic equipment available to provide detailed interpretation of the results it is fair to assume that the preceding row of holes had insufficient time to fully evacuate the rock before the subsequent row fired. This would in effect extend the burden and therefore increase the distance for the compressive wave produced on the following row to travel before reflecting into a tensile wave thus reducing fragmentation.

The lack of breakage would reduce the overall throw from the blast. Extending the distance towards three metres gave reasonable fragmentation and throw. A compromise between effectiveness and the ability of the drillers to operate effectively was established at 2.5 metres. The only exceptions to this burden and spacing when modifications were made to accommodate irregularities in the face. When used with a chevron initiation pattern, the holes have a burden to spacing ratio of 1.41.

The ratio of charge to burden was found to release sufficient energy to fracture the rock and still produce enough gas pressure and volume to heave the rock to a height of approximately 15 metres. Examination of the images in this section will show the landed site and attention should be drawn to the "pull" achieved from the new face. Detailed examination will show that there is virtually no back break.

11.3.4 Initiation system

To implement the conversion to the new blasting techniques required changing from the existing initiation system to a much more a sophisticated system that had almost unlimited capacity in terms of the number of holes that could be fired. A decision was taken to change from the combination of electrical detonators and detonating cord to the NONEL system of surface and down the hole detonators. The NONEL system is fully described in chapter ten, fundamentally it consists of a surface delay arrangement, incorporating the "Snapline surface connector system" this is available with the following delay times; 0, 17, 25, 42, 67, 109 and 179 milliseconds (m/s). These detonators were used to apply a delay between respective holes, or lines of holes. Using the NONEL system somewhat reduces the areas for experiment, this is because there are only four short period surface delays available, these being 0, 17, 25 and 47 milliseconds. Using test blasts, it was found that in most areas of the site, a delay of 25 milliseconds between rows gave the best results by ensuring that, particularly if the rock was intact, the individual rows fully developed their detonation products and effects before the next row fired and where the rock was excessively fractured, a 47 millisecond delay proved more effective. The 17 millisecond delays were the least effective.

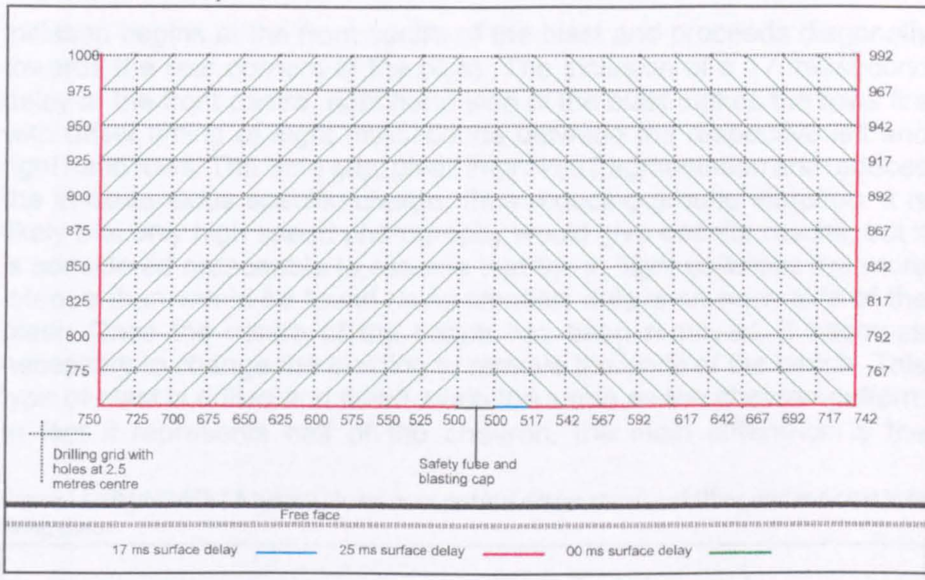
Down the hole initiation was achieved using the NONEL Unidet system of in hole delays, this is a relatively new development in detonator design, the main principle is to reduces the risk of missfires by delaying the initiation of the main or column charges until all of the surface system has been employed. The delay is known as the "burning front" and the method of delay is known as "advance initiation". Three delay periods are available, these being 400, 450 and 500 milliseconds. At Chilanga a 500 millisecond delay was used, this provided the maximum delay and therefore the largest number of holes that can be fired before main charge initiation begins to take place.

Chevron pattern

The complete face was too long to fire in one blast and to achieve optimum results, particularly with regard to blending it was necessary to designate an area close to the centre of the face and design a suitable blast. A chevron pattern was found to be most suitable and in accordance with the principles described earlier in this chapter, a three stage system of breakage was envisaged;

- reflected shock wave
- torsion and stress from the gas pressure
- collisions in the throw

Figure 11-18 [ref Mills] A 2:1 ratio chevron blast design showing drilling grid, blast hole firing times and chevron sequence



To achieve the third stage of breakage, the blast was planned and timed so that material from different parts of the blast were thrown together and to be successful, the blast must use an aspect ratio of not less than 2:1. A chevron pattern is entirely dependent on the timing of the rows within the blast, and to produce the chevron the initiation is timed to progresses from the front centre to the outside rear. A typical blast pattern is shown below and the description is given in the same steps as the shotfirer would proceed, it assumes the viewer to be looking towards the blast and directly in front of the initiation point, essentially, the blast is divided into two sections and proceeds as follows;

- The centre of the front row of the blast is established and the holes each side of the centre line are connected to the initiation detonator. Other than this, the two sides of the chevron can virtually be treated as separate blasts.

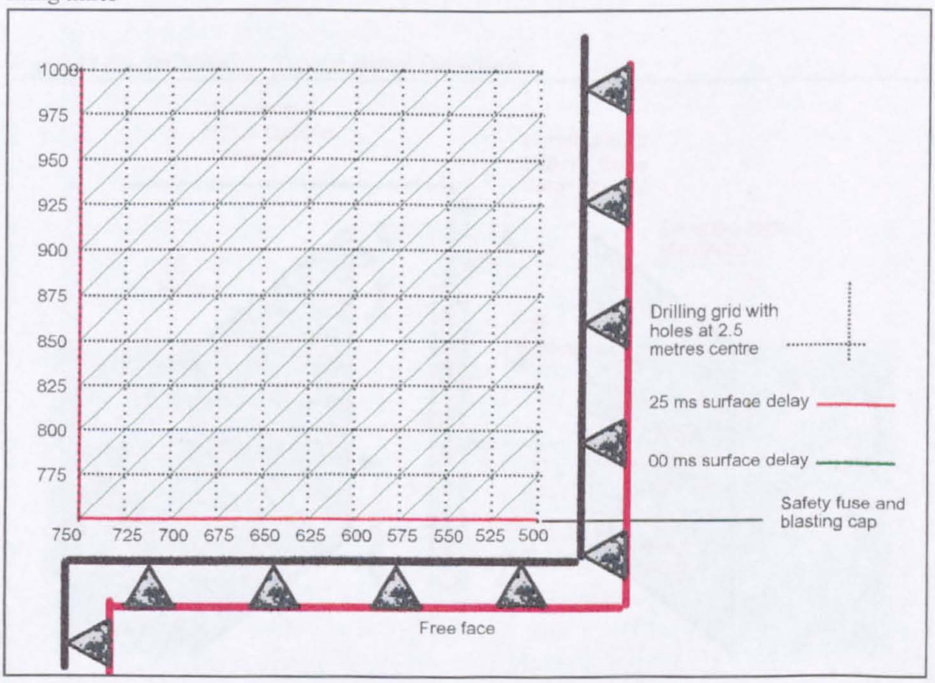
- Left hand side, starting from the centre line, the first row or number one hole is fitted with a zero delay connector, the following holes are joined outwards with 25 millisecond snapline connectors (red lines).

- Right hand side, starting from the centre line, the process is a mirror image of the eastern side, except for the number one hole which is fitted with a 17 millisecond delay (blue line).

- From the number one hole on each side of the blast, heading inwards, the rows of holes are identified and connected by instantaneous snapline connectors (green lines). The rows radiate outwards away from the centre at an angle of 45% to the face.

Initiation begins at the front centre of the blast and proceeds diagonally towards the rear corners of the blast. The inclusion of a 17 millisecond delay at the front centre, right hand side of the blast makes the rows fire with offset timing of eight milliseconds between the respective left and right hand rows. This time gap, gives improved fragmentation and reduces the instantaneous specific charge, thus reducing ground vibration. It is likely that only high speed photography would give definite results, but it is considered reasonable to assume that the in flight collisions are more intense than would be found using identical delays on each side of the blast. Once the centre of the bench has been removed, it becomes necessary to change the blasting to remove the ends of the bench. This type of blast is drilled and timed much the same as the chevron pattern. In fact it represents half of the chevron, the main difference is the

Figure 11-19 [ref Mills] A typical corner or angle blast design showing drilling grid and blast hole firing times



improved aspect ratio. The results from this type of blast are less satisfactory from those obtained from the chevron, the main considerations being the missing third stage of breakage from the mid air collisions. Although the availability of the extended free face benefits the reflected shock wave, the first row will often be out of shape and suffering to some extent from over-digging, this tends to allow the rocks to depart the blast without being shattered

11.3.5 Charging

The main charges were made consistent by the using a detonator sensitive water gel or emulsion explosive as a primer and base charge, the main charge or column charge was ANFO. As pumped and bulk explosives were not available the ANFO was site mixed. Providing that the depth of the hole was similar, the charging of each hole was identical to its neighbour. In practice, each hole was inspected for its accuracy and if found to be correct, was loaded with a water gel or emulsion primer, into this was inserted two 500 millisecond delay Unidets, the column charge of ANFO was mixed and loaded and finally, the hole stemmed with suitable crushed rock material brought from the plant. Occasionally, the gentlemen loading the holes with ANFO would often get into an animated conversation and forget to stop their loading. As the men covered up their mistake with a few inches of stemming, the result was not always discovered, this often led to fly rocks departing the scene from around the collar of the hole. Generally, the standard of drilling and charging was sufficiently high to enable good results to be achieved.

Figure 11-20 [ref Mills] Typical charged blasthole

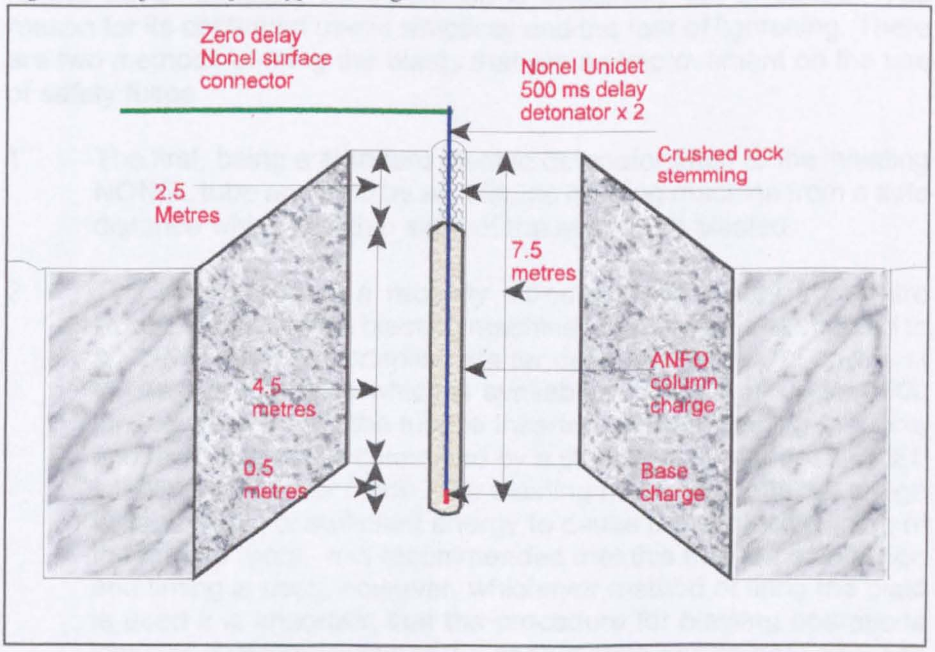


Figure 11-21 [ref Mills] Blasthole statistics

•	Hole depth	7.5 metres
•	Hole diameter	76 mm
•	Sub drilling	0.5 metres
•	Angle	zero
•	Azimuth	zero
•	Burden	2.5 metres
•	Spacing	2.5 metres
•	Stemming	2.5 metres
•	Explosive column	5 metres
•	Volume of rock blasted	44 m^3
•	Tonnage of rock blasted @1 m^3 =2.5 tonnes	110 tonnes
•	Column charge (ANFO)	15 kg
•	Primer (water gel)	3 kg
•	In hole detonators x 2	500 m/s delay Unidets
•	Surface detonator zero delay Snapline	

Blast Initiation

Initiation was achieved by the use of safety fuse and blasting cap (a "number six" power detonator). In most countries the use of safety fuses has for some time (excluding Zambia where it is the preferred method), been prohibited, mainly because, from lighting the fuse until the blast fires (about seven minutes) the operation is effectively out of control. The reason for its continued use is simplicity and the fear of lightening. There are two methods of firing the blasts that are an improvement on the use of safety fuses.

1. The first, being a standard electric detonator fitted to the initiating NONEL tube and fired by an electric blasting machine from a safe distance which is within sight of the area to be blasted.

2. The second, being a recently introduced accessory from Nitro Nobel, the Dynostart blasting machine, this device is connected to a NONEL tube and snapline starter detonator block. An option, is to use a starter tube which is available in lengths of up to 1,500 metres. One end of the tube is inserted into the blasting machine and the other end is connected by a sleeve to a standard NONEL snapline connector block. The blasting machine produces a high voltage spark of sufficient energy to cause the explosive lining of the tube to react. It is recommended that this method of initiation and timing is used, however, whichever method of firing the blast is used it is important that the procedure for blasting operations includes a determination of the danger zone and failsafe methods for the withdrawal of personnel.

Figure 11-22 [ref Mills] Blast summary

Number of blast holes	242
Hole diameter	76 mm
Sub drilling	0.5 metres
Angle	zero
Azimuth	zero
Burden	2.5 metres
Spacing	2.5 metres
Stemming	2.5 metres
Explosive column	5 metres
Length of blast	52.5 metres
Width of blast	25 metres
Area of blast	1312.5 m^2
Volume of rock blasted	9187.5 m^3
Tonnage of rock blasted @1 m^3 =2.5 tonnes	22,968 tonnes
Column charge (ANFO)	15 kg
Primer (water gel)	3 kg
Total explosives per hole	18 kg
Total explosives in the blast	4,356 kg
In hole detonators x 2	500 m/s delay Unidets
Total number of in hole detonators	484
Surface detonators zero delay NONEL Snapline	
Total number of surface detonators	240
Total number of electric detonators	2
Price of explosives (approximate)	£5,000.00
Price of surface detonators	£561.00
Price of in hole detonators	£2,178.00
Price of electrical detonators	£18.00
Total cost of the blast	£7,757
Cost per tonne	£0.34

Summary

By 1997 this quarry had been developed into a model site, that was used by the Zambian Department of Mining as an example of how to develop and operate a site. The local producers of explosives visited the site to learn the principles involved in carrying out the blasts and went on to teach others. The following pages show images of two blasts at Chilanga RP3 quarry the first series of pictures can be identified by their brown tint, this was caused by the smoke and dust present in the atmosphere (prior to the blast). The image directly below shows a large blast of 250,000 tonnes, carried out by the writer at an open strip mine in Zambia, the same techniques were used as those described in the text. The results of the fragmentation are clearly visible, as is the cleanness of the cut on the new face.

Figure 11-23 [ref Mills] A 250,000 tonne blast

Figure 11-24 [ref Mills] A 10,000 tonne blast at Chilanga.

The image above shows the development work being carried out to open the new faces. In the background, a small wheeled loader is moving lateritic overburden. The degree of fragmentation and heave can be judged as can the extension and raised level of the rock pile.

11.3.6 Chilanga blast 1

Figure 11-25 [ref Mills] Pre blast preparations

This image shows part of the bench being prepared for the blast, addition holes were drilled to remove the area in front of the supervisor.

Figure 11-26 [ref Mills] Pre blast view

This image shows the full bench that is to be blasted, the varying materials can easily be identified by their colour.

Figure 11-27 [ref Mills] Blast sequence 1

This image shows the start of the blast with about six rows having fired on each side of the chevron. To the left of the blast can be seen the surface detonators firing. Careful inspection to the centre right of the dust cloud will show the material being lifted towards the left. The stemming can be distinguished from the rock by its brown colour.

Reference 11-28 [ref Mills] Blast sequence 2

The blast is now about half way through its sequence, again the direction of the blast can clearly be seen, especially to the left where the rock is being thrown to the right.

Figure 11-29 [ref Mill] Blast sequence 3

The next image in the sequence shows even more clearly the material being throw inwards, again the brown dust is the stemming. The dark cloud is badly mixed ANFO.

Figure 11-30 [ref Mill] Blast sequence 4

This is almost the end of the blast, the sides of the blast can still be seen to be thrown inwards and the dust cloud in now vertical and beginning to collapse.

Figure 11-31 [ref Mills] Blast sequence 5

The bench after the blast, the rock pile can be seen to be low to the right and high towards the centre. The area directly in front of, and to each end of the face is virtually free from blasted material. The blending in this blast was excellent.

Figure 11-32 [ref Mills] Blast sequence 6

This picture shows the writer and two quarry supervisors standing close to the centre of the rock pile. The quality of fragmentation, heave and positioning of the blast can be seen. The rise above original bench level averaged 3 to 4 metres.

11.3.7 Chilanga blast 2

Figure 11-33 [ref Mills] Blast sequence A

The start of a blast, the surface detonators can be seen firing as a white dust cloud at the surface of the blast.

Figure 11-34 [ref Mills] Blast sequence B

The detonators have all fired and the main charges are beginning to fire.

Figure 11-35 [ref Mills] Blast sequence C

The mushroom shape of the dust cloud shows the blast being directed towards the centre and close inspection at the left hand side shows the rock being thrown inwards.

Figure 11-36 [ref Mills] Blast sequence D

The blast is now complete and the dust cloud is starting to collapse within itself.

Figure 11-37 [ref Mills] Blast sequence E

The height of the heave or lift can be seen in this image, the rock has been thrown well clear of the new face and heaved about 3 to 4 metres more than the original level.

Figure 11-38 [ref Mills] Blast sequence F

The pull away from the corner can clearly be seen in this picture, as can the degree of fragmentation and the clean cut of the standing faces.

11.4.0 NDOLA WORKS

11.4.1 Description

Chilanga Ndola Works operate a shallow quarry directly adjacent to their cement plant, the quarry covers a level area of approximately one square kilometre and has mining rights on a vast amount of undeveloped land.

Figure 11-39 [ref Mills] Ndola works quarry showing the face and flooded lower bench

Development to date has consisted of sub-ground level works of two, seven metre benches, currently, the lower bench is intentionally flooded and used as a source of water to the plant. Geologically the deposit is uncomplicated and without significant variation, The exception to this, is a band of material that is located close to the crusher, this material is very similar if not identical to the material designated L1 by the Ndola Lime Company. When both kilns are operating, the requirement from this site is 1,000 tonnes of high quality limestone each day at seven days per week. The benches have been effectively operated, are established, level, and stable This mine is extremely easy to operate and despite there being no geological data or mining plan, the development to date has been satisfactory. The only aspect that makes this

Figure 11-40 [ref Mills] A dump truck and lateritic in-fill

development less than perfect is that the limestone in the upper bench has proven in places to be severely eroded and in-filled with lateritic soil, also, the lower bench is close to the water table and suffers from the drill holes being filled with water. The crusher at Chilanga Ndola Works is a single pass, multiple rotor hard rock impactor with a capacity of up to 400 tonnes per hour and a capability to accept rocks of one cubic metre and crush to passing 25 mm, because of this, fragmentation in blasting is rarely a problem.

Drilling

Figure 11-41 [ref Mills] Angle drilling with a drifter rig

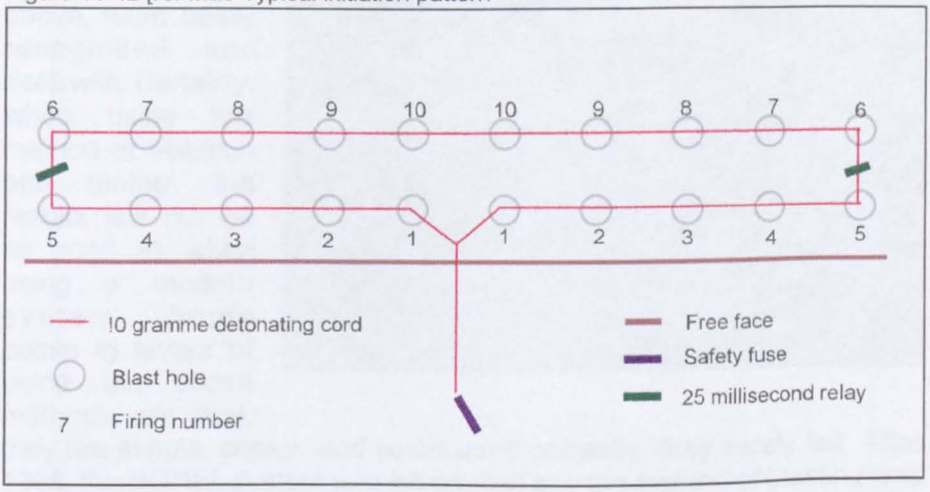
Drilling operations were carried out using an Atlas Copco 601 drifter type rig, drilling holes of 79 mm diameter to a depth of 7.5 metres (0.5 metres sub drilling), at a burden and spacing of 2.3 and 2.7 metres respectively, an angle of about ten degrees was aimed for and an azimuth of zero. After 1996, this changed to an Ingersoll Rand Drifter rig drilling vertical holes to the same diameter with a to a burden and spacing of 2.5 metres. The angle of the mast can clearly be seen on the image, it would seem that on this particular hole, the angle was excessive.

Blasting

Prior to 1996, a typical blast would consist of two rows, each of 20 holes. The blast would produce 4,347 tonnes of rock and blasting took place twice weekly. The holes were loaded with a base charge of gelignite, water gel or emulsion, depending on what was available and the column charge consisted of pre-mixed and packaged ANFO. Initiation was made by running a length of ten gramme detonating cord to the booster which was located in the bottom of the blast hole, the cord extended from the top of the hole where it was connected to the surface initiation system. Timing of the blast was made by using ten gramme detonating cord and millisecond delay relays, the holes were timed to fire in two lines, with the second lines of holes being delayed by 25 milliseconds. This type of blasting has many disadvantages, with the risk of missfires and dead pressing being the most obvious;

Although described as being instantaneous, detonating cord does detonate in a direction away from the initiator, and with this firing pattern, the initiation will begin at the safety fuse which is positioned between the holes marked number 1, the blast holes will then be initiated in the sequence shown above, remembering that a twenty-five delay is positioned between holes five and six. The blast holes marked number ten will detonate approximately 20 to 30 milliseconds after blast holes number 1, the variation is due to the potential inaccuracies found in the relays. As it takes less than two milliseconds for the shock wave generated in a blast hole to travel two metres, the explosives in the second row will experience the shock waves generated from the front row. In this instance, it is likely that any explosives containing a fluid medium in the second row will be crushed by the shock wave and rendered insensitive to detonation with the high possibility of a missfire.

Figure 11-42 [ref Mills Typical initiation pattern



Ten gramme detonating cord does not generate sufficient energy to cause ANFO to detonate, therefore, if the cord is entering the blast hole at the top and travelling down through the ANFO column charge to reach the booster, it is almost certain that most of the ANFO will either be totally destroyed by the detonating cord, or dead pressed and rendered unable to detonate thus reducing the potential energy available for work.

The trunk lines of detonating cord can be and often are severed before they have initiated the whole of the blast, at this site, there has proved to be a 5% risk of the lines being cut, and an in an average blast of 100 holes, finding up to 5 unfired holes was common. There may be several causes, for the cord being cut, these being;

- rocks thrown into the air and landing on the cord, the rocks may have been thrown from the cord itself or from a previously fired blast hole,
- the speed of reaction of the cord, in this instance, the detonation

front is travelling so fast that it is unable to go around corners, this may cause the cord to discontinue the detonation at a sharp angle or a knot,

- lines of cord layed close together but with a difference in timing may cause the later cord to become damaged without actually detonating it,

- movement of the ground can pull the lengths of cord apart before they detonate.

Figure 11-43 [ref Mills] A clean area, note the dust.

Although potentially dangerous, misfires caused by the above, were easily recognised and dealt with. Certainly, when using this method of initiation and timing, the results will not be as good as when using a modern system. Some points in favour of using the above methods are that, they are simple, cheap, and when used correctly, they rarely fail. After 1996, the NONEL system was introduced and the method of blasting was changed to that used at Chilanga works. This gave a significant improvement in the throw of the blast and the ability of the loader to clean the face.

11.5.0 NDOLA LIME COMPANY

11.5.1 Requirements

Figure 11-44 [ref Mills] The flooded Fox cut quarries

Ndola Lime Company operate this large site to produce limestone for conversion to lime and all of the main mining area is to the north of the plant in an area that extends about one kilometre north and two kilometres to the east and west. The daily requirement for rock (should all of the plant be working) is between 3,000 and 5,000 tonnes per day. Several factors cause the blasting to be more complicated than normal, the main ones being;

- a wide variation in rock type. The plant operates two kilns, one rotary and one vertical, each with an approximate product rate of between 500 and 600 tonnes per day. Because of its more efficient use of fuel, the vertical kiln is the most cost effective of the two. To obtain best performance, it has been found that the vertical kiln requires a specific type of rock. Geological surveys have established that less than 20% of the total rock mass is of the required type and of this, most of it is embedded in less useful rock and to selectively blast the preferred rock is almost impossible,
- a requirement for the rock to have high thermal integrity. When used in the vertical kiln, it is of paramount importance for the rock to remain intact throughout the calcining process. Failure of the rock will result in the fines blocking the passage of the gases and thereby causing the calcining process to fail, Inappropriate blasting techniques will produce micro-cracking in the rock, thereby reducing its integrity,
- although originally planned to be operated by rope face shovels, extraction has been by rope shovels, Caterpillar 992B and 988B wheeled front end loaders, each requiring a different blasting technique,
- the mine was (until 1999) owned by the Zambian Consolidated Copper Mines Group of companies and staffing, particularly at a

senior level was carried out from within this group. Unfortunately, although believed to be competent in underground mining, the group have in recent times displayed little expertise in surface mining, particularly with regard to blasting techniques. A consequence of this, is the staffing of the mine with inappropriately skilled personnel, these being either unable or unwilling to apply more applicable techniques,

- because it has for many years been suggested that this mine is sold, there has been little investment in mobile plant and mine development, because of this and other reasons the mine has fallen into a state whereby to establish an efficient operation will require a massive investment in skills and finance

- since breaking into a major aquifer in 1996, the Fox cut, being main source of rock has been completely flooded, with mining resorting to the inferior material in the old Fox cut.

Development

Although being in operation for some thirty years, the development been badly planned with the rock being won on a day to day basis. Development has consisted of extraction from several locations, these being;

- Mwatesi quarry
- Old Fox cut quarry
- Fox cut quarry
- New Fox cut quarry

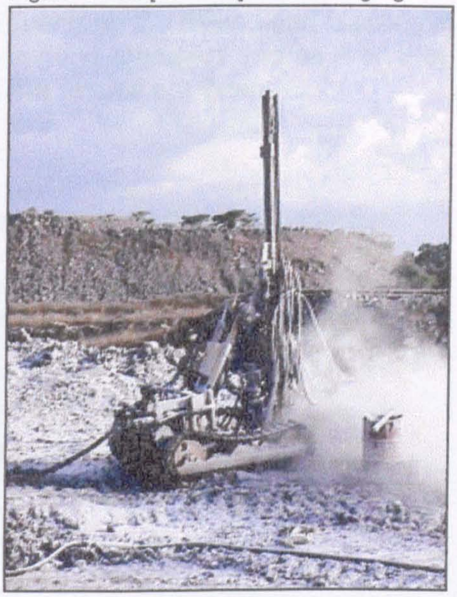
Of these locations, the first major site at Mwatesi quarry was successfully developed and subsequently abandoned, the reason given for the abandonment was the friability of the rock. This probably coincides with the installation of the vertical kiln and the need for a more stable material.

The main deposit extends from the east to the west in a slight crescent shape and the second development was an advance south-eastwards to a site identified as the Old Fox cut. From this site, folding has caused an increase in the width of the deposit and development proceeded in an easterly direction following the anticline until reaching the Lake Ishiku protected area where the deposit attenuated. The area between the Fox cut quarry and the protected area was identified as the New Fox cut and worked until 1996 at which time it broke into a major aquifer and became flooded. Since 1996, mining has returned to the Fox cut quarry and the New Fox cut has been abandoned.

Possibly, as a result of the frequent changes of sites and a general lack of expertise, the development of the mine has been very unsatisfactory, this has shown itself in the ad-hoc development of benches and bench development. The lack of detailed planning has caused the blasting operations to become very haphazard.

Drilling

Figure 11-45 [ref Mills] NLC drilling rig



A range of drilling equipment is in use ranging from drifters to down the hole equipment, most are in poor condition and none have any means of measuring angles. The drilling is informal with little attention given to the geometry of the blast, observations over a protracted period have found burdens and spacing within the same blast ranging from one metre to four metres. The angle of drilling is intended to be vertical but has been seen to vary from minus to plus fifteen degrees, the azimuth is often found to be different on each hole. The poor drilling is probably due to a lack of skilled supervisors, badly maintained equipment and untrained drillers, these factors are usually combined with, uneven floors, varying bench heights, unstable faces and overbreak from the previous blasts. Clinometers operating though two planes would enable the operators, even with a minimum of training to drill a vertical hole. The appointment of an experience foreman who was able to mark out the site would assist the drillers in siting the rig in the correct location.

Currently, the results of the drillers work is such that, effective blasting cannot take place. Should the drillers were able to accomplish it, and assuming that the hole diameter was standardised at 76 mm, a burden and spacing of 2.5 metres could be applied.

Blasting

Standard practice in this mine is to work several benches or sites at any one time and because of this, several patterns of drilling and regimes of blasting will be employed, this being dependent on the individual drilling and blasting teams. Whichever team is operating, the standard method of work is to load the blast hole with a base charge that has had a length of detonating cord attached, the end of the cord will remain on the surface and the hole will be charged with either ANFO or a cartridged explosive, this depending on what is in the store at the time. Stemming consists of fines from the drill and usually occupies between one and two metres of the top of the hole. On completion of loading, the holes will be connected together in lines from the free face and either detonated together or the rear lines will be delayed using relays.

Conclusion

From the above information it can be seen that the drilling and blasting in this mine is not being carried out to a good standard and as would be expected from this type of approach, the results are poor.

The main observations are:

- very high noise levels, air overpressure and ground vibration
- high degree of overbreak
- large variance in the levels of fragmentation
- tight rock pile and corners that are difficult or impossible to dig
- uneven breakage on the floor
- inconsistent throw
- high levels of fly rock
- unwanted blending of the grades of rock

11.6.0 PORTLAND CEMENT COMPANY

11.6.1 Description

Portland Cement Company operate this site to produce limestone for conversion to cement clinker and for transportation to their grinding plant at Blantyre for cement manufacture. The gross daily requirement for rock to the crushing plant 2,000 tonnes per day. The quarry has been in operation for the production of limestone for cement clinker production since 1960. The quarry has never been exploited systematically, nor has it received had sufficient investment to allow for proper development.

Figure 11-46 [ref Mills] A typical view of this spoiled quarry.

The management of the company had, in keeping with other state enterprises been appointed for its relationship with President Banda and other political contacts rather than for any apparent skills in company management. The quarry had been particularly neglected in the area of management and had never operated under any regime that could be identified as standard mining practice. Being a state owned company, there had never been any need to comply with international standards of safety and the quarry was a very dangerous place to work. The quarry was taken over by CDC in 1995 and a major reconstruction was started, one of the main priorities, as seen by the writer was the incorporation of safety policies in the workplace. When producing the feasibility study for the purchase of this quarry, the writer attempted to identify the current condition of the quarry and define a programme for rectifying deficiencies.

The object of the programme was to;

- Review the current environmental and safety conditions
- Evaluate the effects of proposed plans
- Define required actions

Due to a previous lack of funding, quarry development stagnated and by 1996 the ratio of limestone to waste and overburden in material being mined was probably 80% waste to 20% limestone. Because of this, the quarry had virtually reached the stage where for financial reasons further extraction from the quarry could no longer be justified and the operation had degraded to one of winning limestone from any area that was workable.

Figure 11-47[ref Mills] Remnants of past bench development.

The following is a list of some of the problems found with the quarry, for legal reasons only some of the deficiencies can be mentioned. One of the most obvious results of poor operating practices is the sterilisation of all the lower benches.

The specific reasons are;

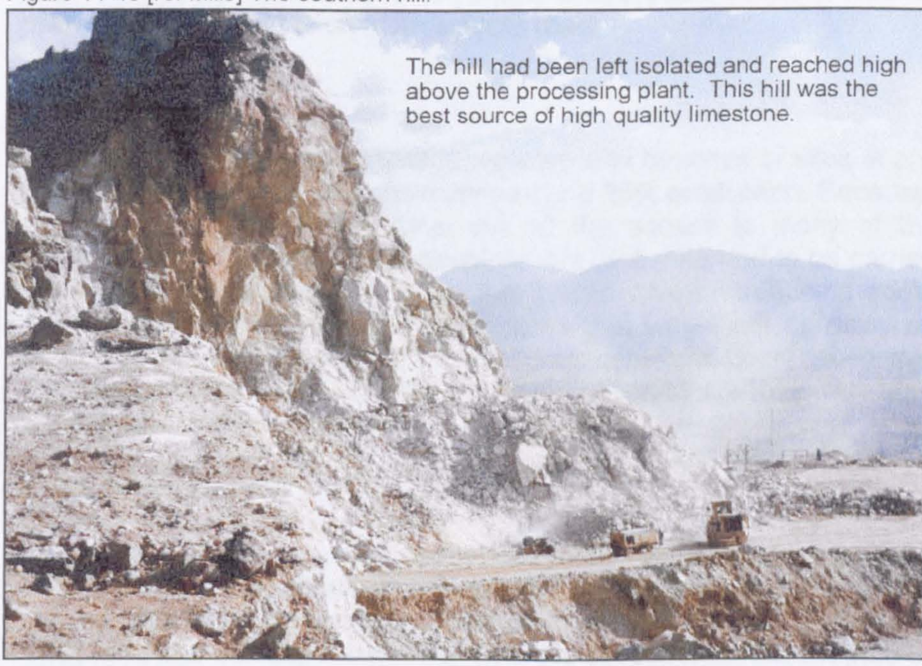
1. The height of the benches and the instability of the faces has made any further operation in this area extremely hazardous.
2. Although not presently enforced it can be assumed that mining regulations will soon dictate bench heights which will be less than those currently operated.

3. The access roads are too narrow and the gradients are unsatisfactory.

4. There is insufficient room for the plant to operate effectively.

The haulage was carried out using thirty-five tonne dump trucks. Depending on the route chosen, one of two service roads is used, one descended to the east of the ridge and one to the west. Both roads as constructed were not suitable for use by heavy duty dump trucks. The gradients were too steep, no drainage or effective culverts had been installed, the carriageway width was too narrow, there were no safety barriers and the surfacing was inadequate. For safety reasons, gradients of the mine haul roads should not exceed 1:8 and should ideally be no steeper than 1:10. These gradients are internationally accepted as safe practice in quarries.

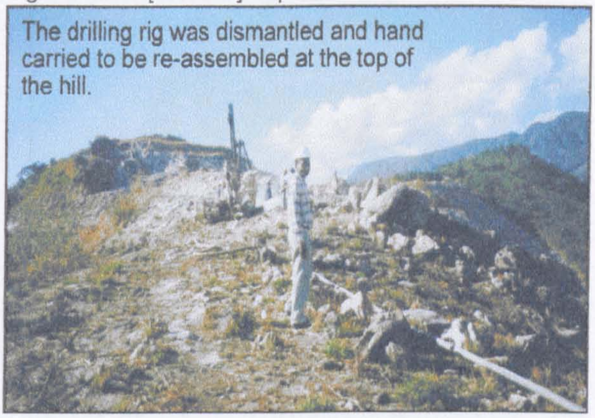
Figure 11-48 [ref Mills] The southern hill.



Drilling

The company operated several rigs and had sufficient drilling capacity for both development and production. Since 1996, the mine has owned three Ingersoll Rand 345 tracked drill rigs, two configured as drifters and one as a DTH. The preferred hole diameters are 76 mm for the drifters and 102 mm for the DTH. The machines were supplied new from South Africa and primarily designed for work underground, drilling short holes and as such were not fitted with clinometers or dust suppression equipment, these items have since been supplied from the UK. The drillers were competent are their work was accurate with respect to geometry and depth and to keep things simple, the angle of drilling was be vertical.

Figure 11-49 [ref Mills] Top of the southern hill.



To develop the souther hill, one of the new rigs was completely dismantled and hand carried to the top of an eighty metre peak, where it was re-assembled and used to blast an access road.

Blasting

Standard practice in this mine is to work several benches or sites at any one time with a split of 75% development and 25% production. Because, previous mining operations have cut off the access to many of the production benches, extensive development work has had to be carried out to re-open the access routes, this has involved introducing some innovative drilling and blasting procedures that would not normally be found in a production mine. One of the main problems being faced was that it was impossible to place the rigs in a position where they were able to reach and drill the required site.

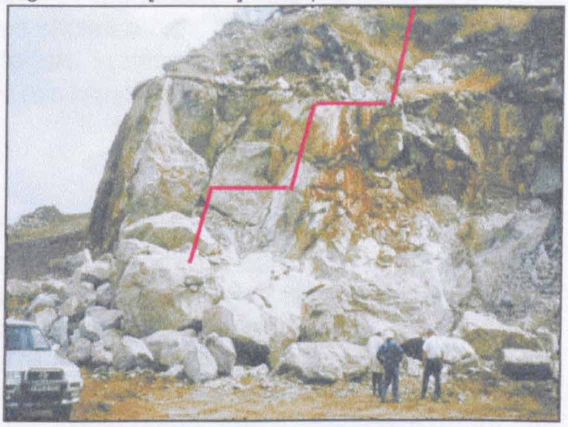
Development blasting

Because some of the benches were completely unaccessible, a blasting technique known as "Snake hole blasting" or springing was employed. This consists of drilling one or more lines of horizontal holes into the base of the face and blasting it out. The recovery of rock from this method of blasting is high, unfortunately, the blocks of rock are usually too large to crush and have to be secondary blasted.

Snake hole blasting can be very dangerous, this is because the free face is the same as the entrance to the hole, in effect producing a gun barrel or cannon. This can lead to high velocity fly rock escaping the hole. Providing this is recognised, measures can be taken to evacuate the site prior to blasting. To be effective, the holes must be fired from the bottom, axial initiation will not work, this is because, with axial initiation, the top of the hole is blown out first, thus releasing the gas pressure before the charge has fully fired. It is almost impossible to completely guarantee the

the effects from snake holes, and extreme care must be taken when using this technique. The snake holes could be effectively charged whatever the angle by using compressed air to blow ANFO into the hole. In the image shown alongside, the purpose of blasting was to widen and make safe the access road (shown on the left) and to begin cutting a series of benches that are marked in red, this was completed in 1997. The drilling rig used was a drifter, making holes of 76 mm diameter to a depth of six metres.

Figure 11-50 [ref Mills] Rock pile from snake holes.

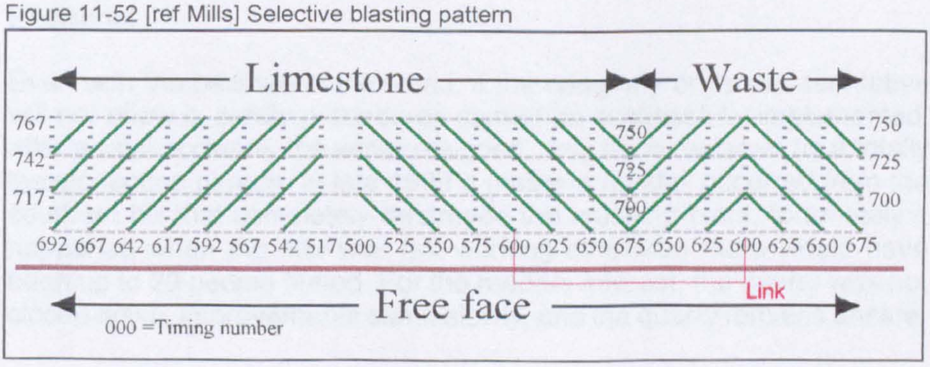


Production blasting

The geology of this site has already been provided. In fact the site is very complicated with limestone only forming about 50% of the rock mass. Because of this, it is not cost effective to plan a blast in the conventional sense, instead it is more effective to design the blast around the various rock types. Two methods of blasting have been used, the first being to blast selectively and only drill and blast the limestone, leaving the other rock types in place to be blasted and excavated later, the second method is to drill and load all the holes regardless of the material and design a firing pattern

Figure 11-51 [ref Mills] Typical intrusions.

Figure 11-52 [ref Mills] Selective blasting pattern



where the limestone is projected in one direction and the other rock types in another. The drawing above shows a typical blasting pattern for use in an area that has differing materials, in this instance, the two materials will be separated by the heave of the blast into two distinct piles.

Secondary Breaking

Any boulders which were thought to contain a reasonable amount of limestone were carried to an area close to the crusher for further reduction by hand drilling and secondary blasting. This method of reduction was labour intensive, used high maintenance equipment, was time consuming, disrupted the operation of the quarry and judging by the number of holes in the roofs of the buildings, including the plant (over 500 metres away) it was very dangerous. In 1997 the writer purchased a heavy excavator and hydraulic breaker to reduce the oversize rocks.

Conclusion

From the above information it can be seen that the drilling and blasting in this quarry was very complicated and diverse with no two blasts being the same. Because of this it was not possible to provide a standard drilling pattern nor was it possible to standardise on a set method of timing the blast. The choice of both explosives and initiation systems was dictated by what was available, unfortunately, all explosives imported into Malawi are handled by a government agency whose sole interest is profit, it is probably due to this, that much of the explosive was as much as three years time expired when it arrived on site. It is fair to say that in very difficult circumstances, the blasting and drilling teams were trying their best however the following was apparent;

- very high noise levels, air overpressure and ground vibration
- high degree of overbreak
- large variance in the levels of fragmentation
- tight rock pile and faces that are difficult or impossible to dig
- uneven breakage on the floor
- very high levels of fly rock
- unwanted blending of the grades of rock

Postscript

Even with the best will in the world, if the company or its representative will not allow it, safety procedures cannot be successfully implemented, after a fatal accident, the writer resigned, only to be replaced by a totally inexperienced person. In late 1999 a massive rockfall occurred from the southern hill that completely destroyed the crushing plant, fortunately it happened when the site was not working otherwise there would have been up to 20 people buried. For the readers interest, the quarry was not closed down, improvements came slowly, and the quarry remains unsafe.

Chapter Twelve

Excavation

and

Loading Equipment

12.1.0 MOBILE PLANT

12.1.1 Rope shovels

Historically, most quarries were at some time serviced by some form of rope shovel loading onto either small trucks or tubs operating on a local "Jubilee" railway line. These machines have been around since 1835 and were originally powered by steam and latterly by diesel or electricity. Rope shovels are still used in larger mines and dominate the market in the 400 tonnes and upward categories, in this role, they are very cost effective, especially as the service life of a rope shovel can extend to thirty years or more making them a very good long term investment. These machines are able to dig from the tracks upwards in well broken material, they are not able to dig below track (or grade) level nor can they dig hard packed rock. The maximum digging effort is produced when the bucket is halfway through its digging cycle and approximately level with its pivot point. Two rope shovels are located at the Ndola Lime Company quarry, one has not worked for many years and the other is parked perhaps ready for scrapping. Two rope shovels are standing at the Chilanga Works quarry, both have been scrapped. One machine is standing at the Ndola Works quarry, it too has been scrapped.

Figure 12-1 [ref Mills] An 800 tonne rope shovel in Zambia

12.1.2 Hydraulic face shovels

A hydraulic face shovel is a hydraulic excavator configured for the bucket to dig away from the machine with the opening for the bucket at the front and the loading to be from the bottom of the bucket using a clamshell arrangement. For more than fifty years, a variety of manufacturers have produced large hydraulic face shovels. Since then, cost benefits and developments in power transmission elements and kinematics have improved the machines so that they are now commonplace and their use now challenges the predominance of the rope shovel in the 50 to 400-tonne market. Hydraulic powered excavators have been steadily improving, with a modern machine being equipped with computer controlled variable flow/pressure swash-plate pumps, separate pumps and circuits for the slew and travel motors, electronic engine speed sensing fuel controls, pressure compensation, and parallel kinematics in the loading and digging attachments. These improvements have enabled them to achieve a faster rate of loading than a comparable rope shovel, also they are more flexible to operate and can more easily move around the site.

Figure 12-2 [ref Mills] A 400 tonne excavator in Zambia.

A service life of about ten to twenty years is typical of hydraulic face shovel, they require a smaller capital investment than a rope shovel but are always more expensive to operate and maintain. In terms of capacity or loading tonnage, a hydraulic face shovel will be approximately half the cost of a rope shovel, the rope shovel will however, last twice as long and be more economical to operate. These machines can dig slightly below grade level and are able to operate with fairly hard packed or badly blasted rock. With this type of machine it is usually possible to operate a bench height equal to the pivot point of the boom and dipper, for example, in the photograph, the permissible bench height would be about ten metres. Portland Cement Company operate a 40 tonne hydraulic face shovel at their Changalume quarry.

12.1.3 Hydraulic excavators

A hydraulic excavator is in many areas identical to a hydraulic face shovel, the main differences are in the configuration of the digging mechanism, with an excavator, the digging motion is one where the open end of the bucket faces the driver and the digging action is towards the driver. The other main difference is the size, with face shovels generally being produced larger and heavier than excavators, the most effective weight for excavators being in the 50 to 200 tonne class of machines.

Reports published by Mannesmann Demag [ref 78] and Liebherr [ref 77], both major manufacturers of hydraulic excavators used data provided by mine operators and other equipment manufacturers to show that in both normal and adverse face operations, loading cycle times can be up to 10 to 15% faster with an excavator than a wheeled loader, and a bucket fill factor of 5% more can be expected. Hydraulic excavators will provide, fast loading cycle times, the ability to dig below grade, cope with toes and poor blasting and give excellent lift and crowd forces. With this type of machine, it is often possible to operate benches with heights of up to fifteen metres, this is permitted because of the ability of the machine to both be able to reach upwards almost fifteen metres and to reach down almost fifteen metres. Although not directly related to a production rate, excavators provide the following advantages over wheeled loaders;

Figure 12-3 [ref Mills] A 100 tonne excavator (Indonesia)

- greater reach to enable compliance with bench height legislation
- more break out power
- ability to operate a rock breaking device
- significantly improved fuel consumption
- the driver can see into the bucket

A suitable excavator for primary mining would be configured for operation in an extreme environment this, applies particularly to aspects of safety such as Roll Over Protection (ROPS) cabs, Falling Object Protection (FOPS) and windscreen protection. For operation in a typical small quarry the machine will have an operating weight of 75 tonnes, the ground engagement equipment such as buckets and teeth must be suitable for extreme service and the undercarriage designed for high strength and high ground pressure. To achieve high ground pressure, the undercarriage on excavators designed for quarry work is shorter than normal, the chains are stronger, the pads are narrower, have fewer cleats or bars and are of heavier construction. Generally the machine will have more ground clearance and the belly plates are stronger than normal. An advantage of using hydraulic excavators is that they can readily be fitted with rock breaking tools such as, hydraulic hammers, and pincers, smaller machines can be fitted with a mast and used as a blast hole drill.

12.1.4 <u>Wheeled loaders</u>

The introduction of the large wheeled front end loader brought mobility paired with an acceptable loading capacity, compared with a rope machine, the unit price was dramatically reduced. The first machines had a rigid chassis with either front or rear wheel steering, in the nineteen sixties, new developments saw the introduction of the articulated chassis with centre pivot steering and rigid mounted axles. It was this innovation that lead to the family of machines that is now common in mining for without centre pivot steering and its inherent stability, it is not possible to operate large machines.

Figure 12-4 [ref Mills] A front end loader (Montserrat)

Wheeled front end loaders are primarily designed to load large quantities of rock from loose stockpiles. In order to fill the bucket, the whole machine has to move and to prevent damage to the machine, the bucket has to be wider than the machine, this does not always enable the optimum shape of bucket to be employed, hence the development of spade edge buckets. Due to its method of operation and in order to be productive, a wheeled loader has to be constantly travelling, this involves accelerating and decelerating, with much of its usefulness being dependent on the tightness of the rock pile and the skill of the driver. In a report published in 1989 by, Demag [ref 78], they calculated that in an eight hour operating period, a wheeled loader accelerates 2,400 times, decelerates 1,800 times and penetrates the material 600 times.

Should the machine be used to break out and load blasted, but tightly packed rock, the efficiency of the machine will reduce. This is because the bucket has to achieve its initial penetration by the inertia of the machine running into the rock Thereafter, penetration is achieved by the torque of the engine being transferred to motive or tractive effort through the grip of the tyres. An average machine will have about 40% of the engine power available for pushing, the remainder is used by the other functions of the machine. Should the tyres loose grip, the pushing effort is lost, in practice, the operator will usually attempt to penetrate the rock until traction is lost at this time, more grip will be provided to the front wheels by crowding the bucket and pushing down on the wheels until traction is regained. Crowding the bucket has the result of redistributing the weight of the machine, often resulting in overstressing the front wheels and lifting the rear wheels off the ground, a

Figure 12-5 [ref Mills] Sometimes the day does not go well, as this 966E driver found when he went for firewood. (Tanzania)

sight commonly seen with badly trained operators. Extreme use of this facility can damage the machine and has been known to cause the machine to overturn. Caterpillar suggests the service life for the 988B front end loader, loading trucks from a bench in good digging, loading and carrying on poor surfaces with slight adverse grades, to be 12,000 hours or about five years.

12.1.5 Dump trucks

Figure 12-6 [ref Mills] A 35 tonne Wabco dump truck (Dubai)

Originally, haulage in quarries consisted of horse drawn trucks or tubs these being either on road axles on more probably on narrow gauge railway lines. In the UK, it is frequently possible to see the path of these tracks as they radiate outwards from the quarry face, much as fingers extending from the hand. The use of these trucks gave way to motor powered trucks and by the end of the nineteen fifties, most quarries were using heavy duty road going vehicles. In the UK, trucks at that time were small when compared to today's machines, they were usually dual purpose road vehicles in the ten to twenty-five tonne capacity range, mostly produced by Foden and AEC, purpose built quarry machines were found, these being Aveling Barford and Terex/Euclid. The first company to specialise in off road dump trucks was the Euclid Road Machinery Company of Euclid Ohio. They began manufacturing these trucks in 1934. Since that time, trucks have got bigger and more complex with the biggest ever being the Terex Titan 33-19, this being twenty metres long and eight metres wide, with a gross vehicle weight of 500 tonnes. Dump trucks have diverged into two categories, these being, rigid chassis and articulated chassis, rigid trucks being built with capacities from thirty to several hundred tonnes and the articulated trucks being built up to fifty tonnes. The articulated trucks were developed in Europe and to a large extent have still to be accepted in the Americas, the main manufacturers of articulated trucks are, Aveling Barford (UK), Bell (South Africa), Volvo (Sweden), Moxey/Komatsu (Norway), Thwaites (UK), Terex (UK), O&K (Germany), Randon (Brazil), Hydrema (Denmark) and Caterpillar/DJB (UK). The range of heavy rigid dump truck manufacturers is wider, with machines being produced by, LeTourneau (USA), Liebherr (Germany), Caterpillar (USA), Terex (USA), Euclid (USA), Komatsu (JAPAN), Wabco (USA), Belaz (Russia), Perlini (Italy) and many others [ref 56].

Articulated trucks

Figure 12-7 [ref Mills] Articulated truck (Montserrat)

These machines were developed in Europe for bulk muck shifting in arduous conditions, being constructed to withstand a high level of abuse and have outstanding performance in arduous conditions. Mine operators have found that they can take advantage of the robust build of these machines for operating in hard rock mining conditions that would normally be unsuitable for the deployment of rigid chassis trucks. Depending of the weight and the expected conditions, the machines can be supplied with two or three axles, with front wheel, four wheel or six wheel drive. The trucks are usually driven by the front axle and drive to the other axles is optional, The front and rear of the truck are connected behind the cab in a pivot arrangement, this allows the chassis to pivot in a horizontal and vertical plane but also in the vertical plane, should the machine become unstable the body of the truck can and often does fall onto its side, this happens without upsetting the cab and can usually, but not always, be recovered without any damage. Being articulated, gives the trucks many advantages over their more conventional competition, The main advantage, is the ability to operate in inaccessible locations on unprepared roads, lower loading height and an improved ratio of gross to tare weight.

Figure 12-8 [ref Mills] Overturned truck (Jamaica)

Rigid chassis

Rigid chassis dump truck are the main haulers used in the surface mining industry, they have been in production for many years and because of this, have reached a high state of development. These trucks are available with carrying capacities of up to 400 tonnes and having bodies lined with abrasion resistant steel are suitable for hauling hard rock. Typically the 20 to 150 tonne range of trucks will have the equivalent of two axles and be mechanically powered, this system is conventional in the sense that a diesel engine drives the rear axle through a semi-automatic transmission. The rear axle may be fitted with traction aids that will limit slippage to one set of drive wheels. Larger trucks may be fitted with a diesel engine coupled to a generator, the generator will power two motors each mounted within the rear hubs, an improvement to this system was developed in Nchanga Ndola, where on the climb out of the mine, the trucks took power directly from a lines suspended from catenary towers, this enabled the vehicle engine and generator to be disabled and boosted the available power to the motors, in this instance improving the haul speed by ten Kilometres per hour. These trucks are generally, reliable, long lasting, and fast, a serious disadvantage is the empty or tare, weight of the truck, which can often, and usually does, exceed the load weight. They can only be effective on well prepared and maintained roads, in wet conditions they can be very dangerous. Typical slope values are 15% for trucks up to 50 tonnes capacity and 10% thereafter. Tyre failure can be a significant problem and as tyre manufacturers maintain a controlled marketing system, the high cost of tyre replacement often represents the greatest operating expense.

Figure 12-9 [ref Mills] A rigid truck (Malawi).

Figure 12-10 [ref Mills] A truck resting. (Malawi)

12.1.6 Dozers

The work of a dozer in a modern day quarry is increasing, this is largely because of, improvements to the design of dozers, thus enabling them to carry out more complex tasks, the requirements to increase the stability of tips, and to cater for environmental legislation, such as landscaping. A secondary duty of the large dozer is ripping. It is said that the first ripper was a device pulled through the ground by oxen and used by the Romans when building the Appian Way. The most useful rippers for hard rock are the single shank, parallelogram type. This consists of a heavy frame which is fitted to the machine, sometimes with a reinforced block to allow a second machine to push, a single tine and a tooth on the end. Hydraulic rams force the tine into the ground and another set of rams enable the operator to adjust the angle of the tooth. Changing the angle of attack provides a way of regulating the penetration, generally the tooth should, once penetration is achieved, be parallel to the surface. A fifty or sixty tonne machine is ideal for working in a large quarry. The machine should be fitted with a straight blade, extreme service ground engaging tools and extreme service undercarriage. Even with good maintenance and suitable equipment, the cost of maintenance will be high and ultimate life of the machine will be short. Careful observation of the picture of the Cat D8L that was operating and owned by Changalume quarry (on the right) will show that the machine was not fitted with a ripper, not fitted with a ROPS cab, and the driver was not wearing a safety helmet. The usefulness of a dozer without a ripper in a hard rock environment, is very limited.

Figure 12-11 [ref Mills] A Cat D8L at (Malawi).

Grader

It is fair to state that the most popular grader in Southern Africa is the Caterpillar series. Caterpillar started producing these machine in 1928 when they bought out the Russell Grader Manufacturing Company of Minneapolis, Minnesota [ref 56]. If the operation is large enough to justify the capital expenditure, a grader is an absolute must, particularly in Africa where it will be required to maintain not only the mine roads, but will perform a social service by building and maintaining roads to the villages. Graders are rated by the size of the blade, with a twelve to sixteen foot blade being the average, Champion produced a machine in 1975 with a twenty-four foot blade. Under normal usage, one grader will be required for 30,000 kilometre tonnes of daily haulage. For use in the African environment, the grader must have drive to all wheels and be reasonably easy to operate. Although often fitted, a scarifier is not usually required. The largest grader built was produced by Umberto ACCO of Italy, it had a

Figure 12-12 [ref Mills] A Cat 14G grader (Tanzania)

1,000 horse power engine and a 33 foot blade. The machine shown in the picture was used by the writer to construct about 90 kilometres of road to service a gold mine in Tanzania [ref 56].

12.2.0 CHOICE OF EQUIPMENT

12.2.1 Considerations

The type of mobile plant used in a mine should be of the correct capacity, ability and size. It must be selected to agree with the type of operation and environment in which it is expected to operate. For example, in African conditions, it would be advantageous to have the fuel systems of the engines calibrated for operation at high altitude, typically, a diesel engine will lose 1% efficiency for every 100 metres increase above sea level and most of central Africa is above an elevation of 1,300 metres [ref 23].

Most of the equipment manufacturers have computer programmes dedicated to selecting the correct combination of loaders and haulers, a typical programme has been used in the case studies. When making a choice of equipment, several factors can come into play which may change the original thinking;

- changes in blasting legislation
- changes in required tonnages
- unavailability of funding
- difficulty of digging
- development of new types of equipment
- unforseen changes in the type of operation
- environmental issues
- inability to get the machine to the site
- expected machine life,

There are many other factors which can modify the original thinking, but generally unless the concept is completely wrong, the changes will be small, for example, changing a wheeled loader for a face shovel, or using a back actor with a hammer instead of secondary blasting. Face loading often accounts for over half the quarry production costs and many studies have taken place to select which type of machine is most cost effective in the hard rock environment.

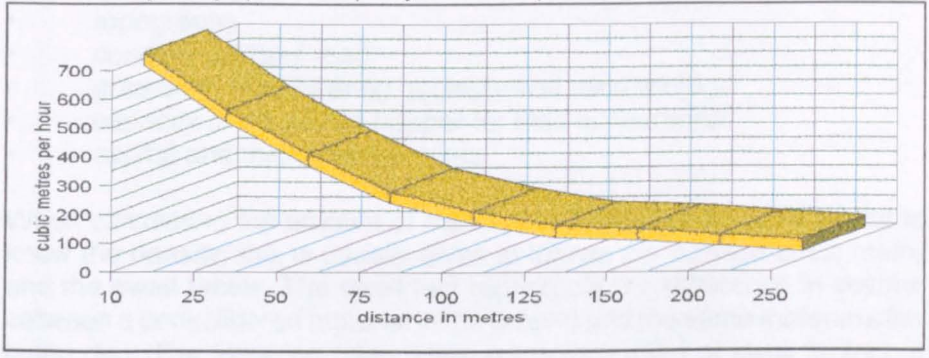
Before selecting a face loader, it is normal to establish the difficulty of digging, for example, it may be more cost effective to rip and doze to a wheeled loader, than to drill and blast to a tracked excavator. To establish which system is most appropriate, the material to be extracted can be described as;

- easy, typically, soil, shale or clay overburden
- rippable, hard clays, weathered, soft or layered rocks

• rock, this can only be excavated after blasting

and, although geotechnical investigation will provide a guide to digging, it is often difficult to decide between the cost effectiveness of ripping or blasting marginal rocks. For a 350 hp dozer the following is considered reasonable;

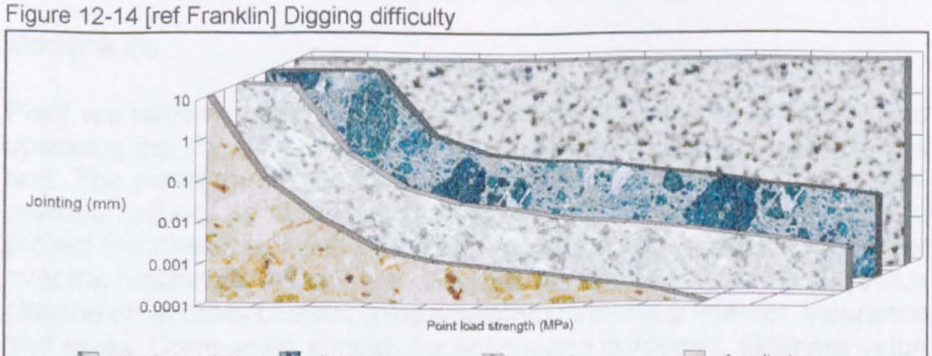
Figure 12-13 [ref 23] Optimum capacity for a 350 hp dozer



Seismic velocity m/s	Difficulty
• 0 to 500	ripping not required
• 500 to 1,000	easy ripping
• 1,000 to 2,000	medium ripping
• 2,000 to 2,500	hard ripping
• 2,500 to 3,000	very difficult ripping to blasting
• 3,000 and higher	blasting required

The graph is a representation of one proposed by Franklin in 1971, although of some interest, the writer believes that there are far to many variables for a graph to predict if a rock can be dug or ripped. Generally, if material can be easily ripped, then this is the method to use, if not, then it should be blasted and loaded.

Figure 12-14 [ref Franklin] Digging difficulty



Haulage

The selection of haulage equipment will depend on;

- quantity and block size of material to be carried
- workshop facilities
- ground conditions
- haul distance
- topography
- condition of haul road
- gradients, road bearing capacity and road width
- capacity of discharge hopper for load acceptance
- capital and maintenance costs

When calculating the amount of material to be hauled it will be useful to know the density, this is usually given in tonnes per banked cubic metre and the swell factor. The swell fact represents the difference in volume between a consolidated material in the ground and the same material after being dug. The following table gives some indication of swell factors, it should be noted the smaller the fragments, the greater the swell.

Figure 12-15 [ref Anon] Density and swell factors

Material	density t/Bm³	Swell factor	Material	density t/Bm³	Swell factor
andesite	2.94	1.67	phosphate	3.21	1.50
basalt	2.94	1.64	porphyry	2.74	1.67
breccia	2.41	1.33	quartz	2.59	1.67
dry clay	1.91	1.35	quartzite	2.68	1.67
earth dry	1.84	1.35	sandstone	2.42	1.61
gneiss	2.71	1.67	schist	2.64	1.61
granite	2.69	1.72	shale	2.64	1.50
limestone	2.61	1.63	siltstone	2.42	1.61
limonite	3.80	1.55	syenite	2.64	1.67

Machine life

Plant will become uneconomic to operate when the cost of owning and operating the machine exceeds the cost of owning and operating a new unit. The costs should be calculated against hours worked as an old machine will not be able to operate the same hours as a new one. To protect the investment and be able to replace it, the owner must recover over the machine's useful life an amount equal to the loss in resale value plus the other costs of owning the equipment including interest, insurance and taxes. Companies should, for accounting purposes, estimate value loss in advance and recover their original equipment investment by establishing realistic depreciation and replacement schedules.

Machine depreciation should not be based or related to any tax considerations, but rather be a simple straight line write-off based solely on the number of hours or years the owner expects gainfully to use the machine. Accordingly, it is imperative that careful consideration is given to the selection of depreciation periods, and that owning and operating cost calculations are based on useful machine life. It is recognised that various factors can influence depreciation periods, such as, local economic conditions and the availability of foreign exchange. Much of the plant fleet that is operated in the case studies should have been replaced many years ago and that the continued use of the equipment is contributing a disproportionate amount to the operating costs of the companies [ref 23].

Maintenance

Figure 12-16 [ref Mills] Plant workshop at Malawi.

The cost of buying a machine is small when compared to operating and maintenance costs, this is especially true in Southern Africa where the machine and spare parts suppliers are seen to apply a virtual monopoly where they make it difficult and in some cases impossible for the machine users to purchase spare parts and consumables from outside of their local dealer network. In all the case studies, the local suppliers operate a pricing structure whereby the cost of a spare part could be up to 400% greater than the cost of the same part purchased in America. For those who are not familiar with purchasing spare parts; the supplier will always ask the customer for the serial number of the machine, then the number will be entered into a computer and within seconds the computer will give the history, locality and ownership of the machine, if this is outside of the dealers concession area, the customer will not be supplied with the parts and will be advised to go to the local dealer.

Tyres seem to present a particularly difficult situation, with only a small number of manufacturers servicing the market through their appointed agents. Efforts to purchase the tyres from outside of the user countries will be met with a firm refusal from the suppliers, unfortunately, this has in the past led to the pricing structure being abused, also, to win the order, the local suppliers are willing to promise deliveries that cannot possibly be met.

In recent years, the producers of heavy plant have introduced ever more sophisticated machine management systems, such as Command Control Steering, Automatic mode transmissions, cushioned bucket stops and kickouts, computerised engine management systems and computerised transmission systems. A typical example is the Caterpillar "Maestro Electronic Control System". Should these electronic systems fail (as they often do) they require the use of skilled technicians with computerised equipment to fix them. To purchase this equipment and train a technician to operate it is often outside of the ability of the average user, therefore to repair the machine requires the services of the supplier. The introduction of these sophisticated systems has not benefited the Third World User.

A system of machine examination that does work, is the regular analysis of the various lubricating oils, interpretation of the results can be carried out by the dealer, or if independence is paramount, then the oil companies will provide the information. From the interpretation of the oil analysis it is possible to identify the beginnings of component failure, thus enabling prompt remedial action to be taken. Failure to maintain the equipment will result in the excessive emission of pollutants and vehicle emissions, road traffic contributes to the major atmospheric pollutants in the following proportions;

- Hydrocarbons 36%
- Nitrogen oxides 51%
- Carbon monoxide 89%
- Lead 70%
- Black smoke 42%
- Carbon dioxide 19%

Acceptable emissions according to European Community Directive 88/77/EEC are;

- Carbon monoxide 4.0 g./k Wl
- Hydrocarbons 1.1 g /k Wl
- Nitrogen oxides 7.0 g /k Wl
- Particulates 0.15 g /k Wl

In Zambia it is possible, with a lot of difficulty to persuade the local dealers to provide a reasonable back up of spare parts and service, whereas in Malawi the same is almost impossible. The difference is mainly due to the population of the machines and the willingness of the dealer to invest in the country.

Leasing

Throughout the life of any quarrying project, the question will arise of whether to purchase, or hire plant and equipment. Ownership costs will employing skilled artisans, operating a sophisticated workshop facility and carrying a substantial amount of spare parts. Alternatively, it may be possible to lease machines from either a competent owner who will supply all of the above, or the main machine dealer may offer a leasing service and already have the facilities available to look after the machines. The option of buying or leasing should raise the following questions;

- is the business large enough to justify a large capital investment
- will money be available
- will financiers want to make the money available at start of mining
- is there a tax advantage
- is it cost effective
- will specialist machines be required
- will some of the machines only be required for a short time
- will the mine have sufficient infrastructure and skills required to maintain the machines
- are skilled operators available, can local people be trained
- will the machine suppliers be able to provide a repair service
- how much investment will be required for purchasing, warehousing, stock control, holding of obsolete stock
- will the equipment have any resale value or become redundant

If in answer to any of the above, there is some doubt, the company should consider leasing the equipment. When leasing or hiring, these questions may be considered;

- are the machines available
- will they be maintained
- is it cost effective
- who will provide the operators
- will the company hold spares
- who will insure them
- when will they be renewed
- who will be in control
- what rates will be applicable
- are there tax advantages
- will the machines be in company colours, is this important
- how often will the rate be reviewed

Almost all of the major manufacturers will offer to lease machines, in most of Europe, this is the direction in which the industry is proceeding, in Africa however, the service offered may not be compatible with that required by the operator and sometimes this can only be appreciated when the deal has been done. Due to the fact that the major equipment dealers in Africa, also operate their own mines, sometimes in direct competition to the users of the machines, the thought of inviting the competition to participate in operating the mine, may not be acceptable.

12.3.0 CHILANGA WORKS

12.3.1 Plant fleet

Chilanga operate with the absolute minimum of mobile plant, with perhaps the exception of the dump trucks, Although being cost effective, operating a minimum fleet can cause problems, for example, should one of the major items fail, particularly if this unit happens to be the 988B the whole works will stop. Most of the plant is well beyond its economic age, but with the exception of the D8K the fleet still performs reliably. This information was correct up to 1999 when the plant fleet at Chilanga consisted of;

Caterpillar 988B loader

Figure 12-17 [ref Mills] A Cat 988B at Chilanga

Face loading is carried out by one Caterpillar 988B front end loader, this is a typical machine for operation in a small to middle sized quarry. The Cat 988B was produced from 1976 until 1993 and can be described as; a heavy duty articulated loader with an operating weight of 43,457 kilograms, fitted with a direct injection fuel system, turbocharged eight cylinder diesel engine, displacing 18.00 litres and developing 375 HP (280 kW) at 2200 RPM. A variable capacity torque converter gives the operator control over the amount of torque directed to the drive (rimpull) as opposed to the hydraulic systems. Torque is directed to the axles via a Caterpillar planetary type power shift (semi-automatic) transmission, with four forward and four reverse ratios. The machine produces a breakout force of 36,330 kg [ref 23].

The front axle is fixed and the rear axle can oscillate through an arc of 26 degrees. Reduction gear units generate torque at the hubs to reduce the amount of mechanism carrying high torque. A failsafe system of braking is used with multiple, oil immersed spring applied discs being fitted as service brakes. The spade-edge, heavy duty bucket has a heaped capacity of five cubic metres. At an average bucket fill factor for blasted rock of 75%, the load in the bucket will be 3.75 cubic metres. This equates to 5.62 tonnes at a density of 1.5. The overall operational height of the machine is 6.753 metres, this dimension is important in that the bench operating height corresponds directly to the maximum reach of the machine which will work the face, (the greater the reach, the higher the working face) [ref 23].

The efficiency of the machine can be considered by comparing the amount of work expended; with a typical cycle distance and an average weight. A total of 49,077 kg is carried 100 metres to load 56.2 kg.

Caterpillar 988A

For duties within the cement works, such as loading coal and crushed limestone a Caterpillar 988A is available. This machine is also available for operating in the quarry and serves as back up to the 988B.

Figure 12-18 [ref Mills] A Cat 988A at Chilanga

The Caterpillar 988A is a wheeled front end loader. It was the first articulated loader built by Caterpillar and was produced from 1963 until 1976. Of several units that were purchased, one of these is remaining it dates from 1975 and has a designed operating weight of 35,800 Kg. The machine is fitted with a bucket of about five cubic metres capacity and is powered by a six cylinder, turbocharged diesel engine giving 325 horse power [ref 23]. The machine is fitted with a general purpose bucket, the purpose of fitting a rock bucket is to give the machine more penetration and break out power. An essential prerequisite when using a machine with little break out power such as a front end loader for bench operation. On a very good day, a 988A has a breakout force of 21,380 kg. The 988A was never a very successful machine, this primarily being due to engine design. The engine was prone to overheating and subsequent cracking of the cylinder head, an expensive repair as the head carried two cam shafts and associated complex valve gear. This machine has had several engines fitted.

Fiat Hitachi excavator 400LCH.2

Chilanga operate one of these machines, it was purchased in 1996 to enable the quarry to comply with mining legislation regarding bench height and to operate as a rock hammer, however the hammer has proved to be unreliable and because of this, the full economic value of the machine has not been realised. The Fiat Hitachi 400LCH is one of a new generation of machines, it is powered by a 285 hp, direct fuel injection, turbocharged diesel engine, giving an excellent power to weight ratio. The engine has a capacity of 13.8 litres. To reduce fuel consumption, an auto-idling device automatically reduces the engine speed when implement controls are in neutral. The hydraulic system consists of multiple swash plate pumps, these sophisticated variable flow/pressure pumps are controlled by a computer aided engine-pump system. The control system is fitted to all modern excavators to improve efficiency. Failsafe, spring applied brakes are fitted to the track drives. The machine has an operating weight of 45,400 kg, can reach to a height of ten metres and to a depth of more than seven metres. Breakout force is 21,500 kg. A hydraulic rock breaking hammer has been supplied with the excavator but has proved to be unreliable. Being below the 75 tonne weight threshold that is normally recognised for heavy mining, the machine is fitted with a light duty general purpose undercarriage system. To comply with any mining regulations that may be introduced, the cab is fitted with falling rock protection and windscreen protection. Generally, the machine is under utilised.

Figure 12-19 [ref Mills] Fiat-Hitachi 400LCH

Caterpillar 769C dump trucks

Chilanga operates three Caterpillar 769C dump trucks. Caterpillar entered into the off highway, rigid dump truck market in 1962 with the 35 tonne 769, followed in 1970 with the 50 tonne 773 and expanded in 1975 with the 777, now a full range of trucks are produced up to the 330 tonne 797. The Caterpillar 769C dump truck is a two axle, rigid chassis, heavy duty rear dump tipper truck. Power is provided by an eight cylinder turbocharged Caterpillar 3408 diesel engine which is rated at 450 hp (336kW). Transmission of power to the rear axle is by a hydraulically operated multiple disc, semi-automatic gearbox with seven ratios. Suspension is provided by four oil-pneumatic cylinders. It is heavily constructed with an empty weight of about 30 tonnes and has a maximum load capacity of 36 tonnes [ref 23]. This gives an all up weight of 66 tonnes. A more typical load is about 28 tonnes, giving an all up weight of 58 tonnes. The haul distance at Chilanga is 9 kilometres each way. Simple calculation will soon show that to haul the 28 tonnes of product 9 kilometres, requires moving the 30 tonne weight of the empty truck 18

Figure 12-20 [ref Mills] A Cat 769C at Chilanga

kilometres, therefore, the total cost of winning 28 tonnes of material at the crusher, is moving 88 tonnes 9 kilometres. The above reasoning is intended to show that two thirds of the money spent in hauling rock is used for moving the trucks back and forth and only one third of the money is being used productively. Now that revised blasting practices have improved fragmentation, consideration could be given to using trucks of lighter construction, for example, road going articulated tipper trucks.

Caterpillar D8K Dozer

Chilanga operates one of these machines. A D8K dozer weighs 32 tonnes and is powered by a six cylinder in line, turbocharged diesel indirect injection engine giving an output of 300 bhp. Although it was regarded as a large powerful machine 20 years ago, this size of dozer is probably the smallest which could be considered for operating in a hard rock environment. A reasonable output from these machines in loose conditions is about 400 loose metres per hour on a thirty metre push to a tip. Caterpillar recommends that when operating this type of machine, in continuous high impact conditions, working on rock surfaces, and pushing and dozing in rock, the effective service life of the machine should be 15,000 hours, about eight years or less.

12.3.2 Production and Costs Estimate [ref 24]

The following production and cost analysis has been calculated for Chilanga Works quarry, when operating one 988B front end loader and three 769C dump trucks. Output required 1,600 tonnes per scheduled 14 hour day. Fleet availability 72.0% and operator efficiency 90%. Trucks fitted with 18.00-33 E3 tyres and restricted to 40 kilometres per hour, rated at a tare weight of 31.34 and payload of 27.00 tonnes.

Figure 12-21 [ref 24] Cat 988B loading details

Loader	
One Caterpillar 988B	
Bucket fill factor	102.7% at 5.5 loose cubic metres
Tonnes per pass	8.77 at 1,540 tonnes per loose cubic metre
Passes per hauler	3.08
Hauler payload	27. tonnes
Percentage of payload	106.1%
Loader cycle time	0.60 minutes or thirty-six seconds
First bucket dump	0.30 minutes or eighteen seconds
Truck exchange time	0.70 minutes or forty-two seconds

Figure 12-22 [ref 24] Cat 769C Cycle details

Truck cycle time	
Loading time with exchange	2.80 minutes or two minutes forty-six seconds
Haul time	11.49 minutes or eleven minutes thirty seconds
Dump and manoeuver	4.00 minutes
Return time	9.0 minutes
Potential cycle time	27.29 minutes or twenty-seven minutes eighteen seconds
Waiting time	0.00
Total cycle time	27.29 minutes or twenty seven minutes-eighteen seconds

Figure 12-23 [ref 24] Chilanga hauling details

Haulage						
Segment	Distance metres	Rolling resistance %	Grade %	Speed limit, kph	Potential speed, kph	Cumulative time, minutes
Haul	9,000.00	1.70	2.00	60.00	46.39	11.49
Return	9,000.00	1.70	(2.00)	60.00	69.03	9.00

Potential production for the 988B loader is 579 tonnes per hour and the potential output for the trucks is 178 tonnes per hour. The production cost is US $ 2.08 per tonne, and the required hours of operation are 14.0 at 115 tonnes per hour achieved [ref 24].

12.4.0 PORTLAND CEMENT COMPANY

12.4.1 Plant fleet

Changalume quarry operated with an abundance of mobile plant and was still unable to either maintain the existing plant, or supply the crushers with sufficient rock. In 1998 a further batch of equipment was hired, consisting of Cat 988 loaders and Cat D8L dozers. Most of the plant is well beyond its economic age, but with the exception of the D8 L still performs reliably. This information was correct up to 1999 when the plant fleet consisted of;

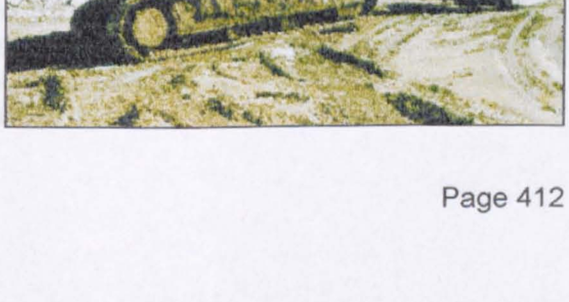
Caterpillar D8L Dozer

Figure 12-24 [ref Mills] A Cat D8L at Changalume

This is probably the most popular heavy dozer ever produced, the quarry operates one and at the time of writing had hired a second machine. In rock configuration, the machine weighs 42 tonnes and is powered by an eight cylinder, turbocharged direct injection engine, giving a power of 370 bhp [ref 23]. A reasonable production from this machine with a thirty metre dozing distance to a tip is 900 loose cubic metres per hour, instead of being used in its true application as a dozer, the quarry use the machine mostly for sorting the limestone from non limestone rock. The unit is in poor condition and should have been replaced some years ago.

Komatsu face shovel

Figure 12-25 [ref Mills] A Komatsu 400 at Malawi in the wrong application, the machine is virtually scrap.



Changalume operate one of these machines and it is a typical case where, the quarry operator having little knowledge of machines has purchased an excavator that is virtually useless. It is fair to state that because it is too small for the duties expected from it (45 tonnes), the life of the machine will be reduced.

Should the machine continue to operate on the quarry face, it is expected that the cost-effective life of the machine will be in the order of two or three years, after one year, the bucket is scrap and the engine and undercarriage need replacing. A face shovel or an excavator is a good machine option for quarrying, however, a suitable machine in this installation would weigh from seventy-five tonnes upwards and be built to operate in a harsh and dangerous environment. The Komatsu is a small lightly constructed machine with no safety guarding or rock protection and is completely out of place in this large and dangerous quarry.

Caterpillar 245B excavator

Figure 12-26 [ref Mills] A Cat 245B at Malawi

Caterpillar have been building excavators since 1972 when they produced the 225, these were followed in 1973 by the 234, 1974 the 245 and 1976 with the 215. Probably in an effort to gain expertise, the company merged with Eder of Germany and Mitsubishi of Japan. In 1994 the 350 tonne, 1,470 horse power 5320 was introduced. The 245B is fitted with a six cylinder in line turbocharged direct injection engine, which, depending on the model produces between 325 and 385 horse power. Typical operating weight is 65.745 tonnes. This heavy duty excavator has a reach of twelve metres and is able to work in areas of the mine which have been made unsafe by previous operators. The driver is protected from falling rocks hitting the machine from either above or from the front by a substantial safety guard and rock deflector. Changalume quarry purchased two of these robust machines in 1996, they are both configured as back actors. One is a standard 245B excavator and one is a 245ME, the ME stands for mass excavator. The 245B is fitted with a Krupp 2,500 heavy duty rock hammer and is used to help develop the mine and to eliminate the need for secondary blasting. The 245 ME is used as a loading tool and is most useful in areas of the mine which are either too high or too difficult to dig with a wheeled front end loader. The loading time with the 245ME into a 35 tonne dump truck is the same as with a 988F loader [ref 23/56].

An advantage of the 245 over the later "3" series of excavators is the minimum of computerised control that is installed, the experience to date of the writer is that the computers take too much control from the operator and sometimes this can be unsafe.

Caterpillar 988F front end loader

Figure 12-27 [ref Mills] A Cat 988F at Malawi

The Caterpillar 988 series of front end loaders is fully described in the section on Chilanga works. The machine was purchased new in1995 and apart from being generally updated, the 'F' version has a slightly more powerful engine than the 'B' version that is rated at 400 horse power. The weight has been increased by one tonne and the breakout force is also increased by one tonne. This particular unit has the unusual feature of the steering wheel being replaced with a joystick. Because it has proven to be reliable, the usage of this machine has been high and a new engine was needed in 1998. The quarry did operate two 988B loaders, one of these was damaged beyond repair and the driver killed when it ran backwards over one of the unprotected faces, the other machine remains in service as a back up for the 988F. It is likely that this machine will soon require replacing.

Caterpillar 769C dump trucks

Figure 12-28 [ref Mills] Improved eastern haul road

The quarry operates two of these trucks, they are fully described in the section on Chilanga Works. A third truck serves as backup for the 769's this is a 35 tonne Komatsu that is occasionally serviceable. The whole site is only 1,000 metres long with a gross elevation of some 200 metres, because of this, the road inclines tend to be steeper than would normally be acceptable. The average haul distance is less than 500 metres. As the quarry is developed it can be assumed that the haul roads will improve and the cost of maintaining the trucks will fall.

Caterpillar D350 articulated dump truck

The Euclid Road Machinery Company of Euclid, Ohio made headlines in 1971 when they brought out a 1,850 horse power, gas turbine powered all wheel drive articulated dump truck. Caterpillar entered into the articulated truck business in 1985 when it bought out DJB Engineering Limited, a company established in 1973, the Caterpillar trucks are still built at the DJB plant. The quarry operates one of these light duty trucks, it was purchased in 1996 to haul silica material from a source approximately three kilometres from the quarry, to the cement works, and for road maintenance. Up until that time, the rigid trucks had been used, with disastrous results to the drive components and tires. More recently, the truck has been used to supplement the services of the rigid dump trucks in the main quarry [ref 23/56].

Figure 12-29 [ref Mills] A Cat D350 and 245B

Maintenance and spare parts

For various commercial reasons, the plant dealer is unable to provide a reasonable service to the quarry and because of this, virtually all the spare parts and consumables are imported from the UK. The standard of maintenance skills is not generally suitable, this results in greater down time than would otherwise be expected. Due to the poor maintenance of the haul roads, the trucks are fitted with the toughest of cross ply tyres.

12.4.2 Production and Cost Estimate [ref 24]

The following production and cost analysis has been calculated for Changalume quarry, despite there being a vast amount of plant allocated to this quarry, the following has been calculated considering the plant fleet to consist of one 988F front end loader and two 769C dump trucks. A production figure of 3,000 tonnes is used, because approximately 50% of the rock is waste, 50% is lost in ignition and cement production is 750 tonnes. Trucks fitted with 18.00-33 E3 tyres and restricted to 40 kilometres per hour, rated at a tare weight of 31.34 and payload of 23.63 tonnes. Output required 3,000 tonnes per scheduled 20 hour day. Fleet availability 56.3% and operator efficiency 80%.

Figure 12-30 [ref 24] Haulage details

Haulage						
Segment	Distance metres	Rolling resistance %	Grade %	Speed limit, kph	Potential speed, kph	Cumulative time, minutes
Haul	500.00	3.00	(10.00)	40.00	75.00	0.75
Return	500.00	3.00	10.00	40.00	24.95	1.13

Figure 12-31 [ref 24] Loading details

Loader	
One Caterpillar 988F	
Bucket fill factor	80% at 6.1 loose cubic metres
Tonnes per pass	7.54 at 1.540 tonnes per loose cubic metre
Passes per hauler	3.40
Hauler payload	23.63 tonnes
Percentage of payload	100%
Loader cycle time	0.60 minutes or thirty-six seconds
First bucket dump	0.10 minutes or six seconds
Truck exchange time	0.70 minutes or one minute six seconds

Figure 12-32 [ref 24] Truck cycle details

Truck cycle time	
Loading time with exchange	2.60 minutes or two minutes thirty-six seconds
Haul time	0.75 minutes or forty-five seconds
Dump and manoeuver	2.0 minutes
Return time	1.13 minutes or one minute-eight seconds
Potential cycle time	6.48 minutes or six minutes twenty-nine seconds
Waiting time	0.72 minutes or forty-three seconds
Total cycle time	7.20 minutes or seven minutes-twelve seconds

Potential production for the 988F loader is 545 tonnes per hour and the potential output for each 769C is 438 tonnes per hour. The production cost is US $ 0.88 per tonne. Required hours of operation are 17.0 at 177 tonnes per hour achieved [ref 24].

12.5.0 NDOLA WORKS

12.5.1 Plant fleet

Ndola works, in common with Chilanga Works operates with the bare minimum of plant. In 1998 the fleet consisted of two 988B loaders and three 769B dump trucks. The daily requirement of run of quarry rock is approximately 2,000 tonnes and the distance from the quarry face to the crusher is one kilometre.

Caterpillar 769B dump truck

The quarry operates three of these vintage rigid chassis trucks, they were produced between 1967 and 1978, have a 309 horse power engine and a capacity of 32 tonnes. These trucks suffered from having the same engine as the 988A and fail for exactly the same reasons, in fact the engines in this application probably does last a little longer than that in the loader, this is probably due to it not having as many accelerations and decelerations in a given work cycle. The economics of operating these trucks must be questioned, for example, it is rare to find more than two out of the three running. It is likely that before long, spare parts will be very difficult to obtain and they are already very expensive.

Caterpillar 988B

The works operates one and the quarry operates one of these machines, their description has been fully covered in the section on Chilanga Works. The second machine is always available to support the primary machine.

Maintenance

Although Ndola Works has its own fully equipped workshop, the main Caterpillar dealer is within 30 minutes drive of the site, because of this, most of the technical work is carried out by the dealer, similarly, spare parts are also easily obtained.

12.5.2 Production and Costs Analysis

The following production and cost analysis has been calculated for Ndola Works, when operating one 988B front end loader and three 769B dump trucks. Output required 1,600 tonnes per scheduled 8 hour day. Fleet availability 59.4% and operator efficiency 89%. Trucks fitted with 18.00-25 E3 tyres and restricted to 40 kilometres per hour, rated at a tare weight of 28.03 and payload of 23.13 tonnes.

Figure 12-33 [ref 24] Haulage details

Haulage						
Segment	Distance metres	Rolling resistance %	Grade %	Speed limit, kph	Potential speed, kph	Cumulative time, minutes
Haul	1,000.00	2.00	1.00	40.00	55.37	1.50
Return	1,000.00	2.00	(1.00)	40.00	71.80	1.50

Figure 12-34 [ref 24] Loading details

Loader	
One Caterpillar 988B	
Bucket fill factor	100% at 6.3 loose cubic metres
Tonnes per pass	9.71 at 1.540 tonnes per loose cubic metre
Passes per hauler	2.38
Hauler payload	23.13 tonnes
Percentage of payload	100%
Loader cycle time	0.60 minutes or thirty-six seconds
First bucket dump	0.10 minutes or six seconds
Truck exchange time	0.70 minutes or one minute six seconds

Figure 12-35 [ref 24] Truck cycle details

Truck cycle time	
Loading time with exchange	2.00 minutes
Haul time	1.50 minutes or one minute-thirty seconds
Dump and manoeuver	2.0 minutes
Return time	1.50 minutes or one minute-thirty seconds
Potential cycle time	7.00 minutes
Waiting time	0.87 minutes or fifty-three seconds
Total cycle time	7.87 minutes or seven minutes fifty-three seconds

Potential production for the 988B loader is 694 tonnes per hour and the potential output for each 769B is 595 tonnes per hour. The production cost is US $ 0.86 per tonne. Required hours of operation are 6 at 280 tonnes per hour achieved [ref 24].

12.6.0 NDOLA LIME COMPANY

12.6.1 Plant fleet

Ndola Lime Company operates with the large mixed fleet of mostly old equipment. Apologies are made for the lack of photographs as for many years it has been impossible to take photographs without being put in jail for spying. Indeed, on two occasions, Ndola Lime Company went to a lot of trouble to try to get the writer expelled from the country. Most of the images are showing similar machines at other sites.

Plant list

- One 1600E P&H electric rope front shovel
- One 1400E P&H electric rope front shovel
- One Cat 992B front end loader
- Three Cat 988B front end loaders
- One Cat 950E front end loader
- Two Cat 769C dump trucks
- Four Terex 3307 dump trucks
- One R35 Terex dump truck
- Two air compressors
- Two drifter drill rigs
- One DTH drill rig
- One Cat D10N dozer
- One Cat D8K dozer
- One Cat 16G grader
- One Terex R15 water tanker
- One Wabco 65 tonne water tanker (on loan from Nchanga)
- One Coles 30 tonne mobile crane

P&H 1600E electric rope shovel

Figure 12-36 [ref Mills] P&H 1600 at NLC quarry

The Harnischfeger Corporation (P&H) originated with Alonzo Pawling and Henry Harnischfeger in 1884 and the first electric shovel, the 1200WL was built in 1933, since then they have become the market leaders in rope shovels. The quarry operated this electrically powered machine for more than twenty-seven years and it has now been left in a broken down condition. To restore it to working condition, it requires a total overhaul of the undercarriage and other substantial repairs. Although showing on the books as the main quarry excavator, it has probably been unofficially scrapped. A large rope shovel such as a P&H 1600E is a very suitable machine for a high output operation, where long high faces are used and forward development is slow. Particular care has to be used when blasting to this type of machine. In general, the rock pile has to be high but well broken.

The usual failure on a face shovel is neglect of the undercarriage, this is partly due to its cost and partly due to the fact that it does not have to move very often and even then it does not have to hurry. It should be borne in mind that even when working from a fixed location, the undercarriage is still being worn out. Being electrically powered reduces the flexibility of the machine making it even less mobile. To power the machine requires a substantial investment in sub stations, cables and other accessories. Once set up, these machines are very economical in operation.

P&H 1400E electric rope shovel

In 1998 the quarry was continuing to operate this twenty-two year old machine, at that time, it was working reasonably satisfactorily and will probably continue to do so. The main risk to its continued operation is the growing scarcity of spare parts. The cost of repairing even a modest breakdown would probably extend to more than the value of the machine, otherwise, the same comments apply to this machine as to the 1600E.

Caterpillar 992B

Caterpillar started building these machines in 1968 and brought out the B version in 1973 even though it is twenty three years old the quarry still operates this wheeled front end loader. The machine has an operating weight of 64 tonnes and the engine develops 550 hp, it does not feature the beadless tyres as shown in the photograph.

Figure 12-37 [ref Mills] A Cat 992B at Dubia

The size of the machine can be judged by comparing the size of the man who is seen repairing the bucket (under the arrow). This model of machine was only manufactured over a period of four years and was not particularly reliable, it has a twelve cylinder version of the engine described with the 988A of Chilanga Works. In good condition it was supposed to produce a breakout force of 29,330 kg. The beadless tyres were a very effective device developed by Caterpillar, they gave the machine high digging power and from time to time they would provide excitement by blowing apart, sometimes the whole track unit would be projected tens of metres, the guard above can be seen to be bent upwards from one that has gone bang. Even with light duties the machine should have been scrapped more than ten years ago.

Caterpillar 988B loaders

The company own four of these machines, they range in age from 1980, 1982, 1987 and 1993, of the four only two have been working regularly. These units are the main loading assets and without them the quarry cannot function. Until 1994, the quarry operated a Caterpillar 988C and this was the main loader, when it unit was returned to ZCCM the main loading duties fell onto the smaller 988B's.

Terex 3307 dump trucks

Figure 12-38 [ref Anon] A Terex dumper similar to the ones at Ndola.

The main truck fleet consisted of five Terex 3307, 40-tonne rigid chassis dumpers, these were purchased in 1979. The picture on the right shows a similar truck. The trucks are fitted with a Detroit Diesel, supercharged, sixteen cylinder, two stroke engine type 16V-71 series. This is an interesting engine that basically consists of two V8 engines bolted together, when managing a heavy plant workshop in Ras Al Khaimah the writer overhauled about fifty of them. The engine has been produced since the nineteen thirties, the later versions featured a modified cylinder liner with the top being wet and the lower part being dry, this improvement being carried out presumably to reduce the inherent fault of "cold ring sticking" where to piston rings gum up and stick in the grooves. One of the main objections to this engine is the volume and pitch of the exhaust noise, this being typical of a two stroke engine (it was featured in the Irish folk song of McAlpines Fusileers as the mighty roar of the twenty-four, referring to the Terex TS24 scraper), other problems are, high fuel use and unburned hydrocarbons being produced especially when the engine is cold. Transmission of power to the rear axle is made by an Allison, hydraulically operated multiple disc, semi automatic gearbox with seven forward ratios, lock up type torque converter and a retarder. Suspension is provided by four oil-pneumatic cylinders. An average life for these trucks is considered to be about ten years to fifteen years depending on their maintenance and to date, one unit has been scrapped, and of the other four, only two are generally available. The biggest mechanical problem seems to be the failure of the rear axle with them snapping off at the hubs, this probably being due to the inner cone of the wheel bearing moving on the axle, causing fretting, followed by a stress raiser and complete stress failure.

Terex 35 tonne dumpers

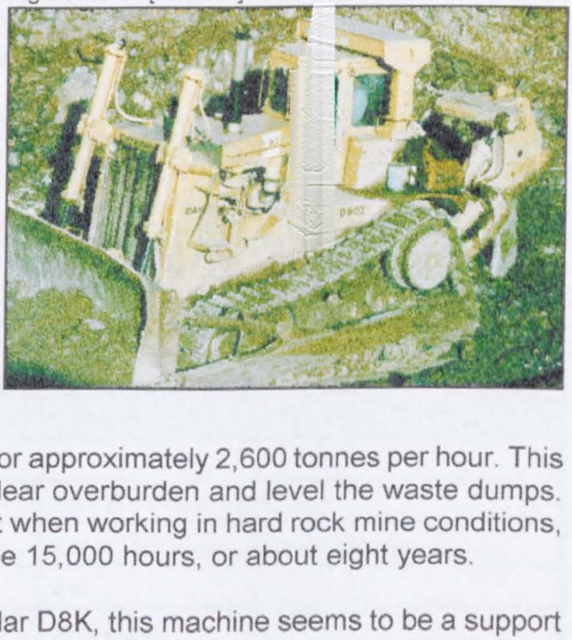
In 1996, the company owned four of these 1972 vintage trucks, when they were new they were a very successful machine. The power unit is a General Motors V12-71 series engine. There was only one of these trucks operating in 1996 and probably none now.

Caterpillar 769C dump trucks

The company operates two of these trucks and they have now become the main hauliers of the fleet. The trucks date from 1993.

Dozers

Figure 12-39 [ref Mills] A Cat D9L in the Falklands.



Ndola Lime Company operates one 1986 Cat D10N dozer. The machine is in reasonable condition. A D10N Dozer weighs 61 tonnes and is powered by a twelve cylinder, turbocharged direct injection diesel engine producing 520 bhp. A reasonable production from this type of machine with a thirty metre dozing distance to a tip is 1,400 loose cubic metres per hour or approximately 2,600 tonnes per hour. This machine is mainly used to clear overburden and level the waste dumps. Caterpillar recommends that when working in hard rock mine conditions, the life of this machine will be 15,000 hours, or about eight years.

A second dozer is a Caterpillar D8K, this machine seems to be a support machine for the D10N. It is well past its useful sell by date and is fit for scrapping.

Workshops

The workshop complex is found to the north of the main office. They are large and comprehensively fitted out. The workshops serve all of the NLC fleet, including heavy plant, light and heavy vehicles. Within the complex are petrol and diesel tanks. Waste petroleum products are recycled.

Observations of the maintenance yard confirm that some of the product ends up on the ground. The drain from the cleaning facility also spills hydrocarbons onto the ground. All the workshop areas require cleaning up.

Summary

By the end of 1998, effectively, the working fleet consisted of four dump trucks, two 988C loaders and one D10N dozer. These machines were totally unable to produce sufficient material for the kilns and carry out the other loading duties.

12.6.2 Production and Costs Estimate [ref 24]

The following production and cost analysis has been calculated for Ndola Lime Company quarry, when operating two 988B front end loaders and four 769C dump trucks.

Output required 5,000 tonnes per scheduled 20 hour day. Fleet availability 64.0% and operator efficiency 85%. Trucks fitted with 18.00-33 E3 tyres and restricted to 50 kilometres per hour, rated at a tare weight of 31.34 and payload of 23.63 tonnes.

Figure 12-40 [ref 24] Haulage details

Haulage						
Segment	Distance metres	Rolling resistance %	Grade %	Speed limit, kph	Potential speed, kph	Cumulative time, minutes
Haul	2,000.00	4.00	1.00	50.00	35.98	3.20
Return	2,000.00	4.00	(1.00)	50.00	70.22	2.40

Figure 12-41 [ref 24] Loading details

Loader	
Two Caterpillar 988B	
Bucket fill factor	100% at 5.5 loose cubic metres
Tonnes per pass	8.31 at 1.500 tonnes per loose cubic metre
Passes per hauler	2.84
Hauler payload	23.63 tonnes
Percentage of payload	100%
Loader cycle time	0.60 minutes or thirty-six seconds
First bucket dump	0.10 minutes or six seconds
Truck exchange time	0.70 minutes or one minute six seconds

Figure 12-42 [ref 24] Truck cycle details

Truck cycle time	
Loading time with exchange	2.00 minutes
Haul time	3.20 minutes or three minutes-twelve seconds
Dump and manoeuver	3.0 minutes
Return time	2.40 minutes or one minute-four seconds
Potential cycle time	10.60 minutes or ten minutes thirty-six seconds
Waiting time	0.00
Total cycle time	10.60 minutes or ten minutes thirty-six seconds

Potential production for the 988B loaders is 1,418 tonnes per hour and the potential output for the 769C trucks is 535 tonnes per hour. The production cost is US $ 1.36 per tonne. Required hours of operation are 17.0 at 291 tonnes per hour achieved [ref 24].

Chapter Thirteen

Crushing and Processing

13.1.0 CRUSHING PLANTS

13.1.1 Introduction

History

Typical first examples of people breaking large hard rocks, consisted of fires burning close to the rocks to heat them and when they had reached the required temperature they were rapidly quenched with water and hopefully, they broke. This practice of using thermal shock to break rock continues in Africa and other third world areas where alternative methods are not available. The next level, or secondary rock breaking, came with slaves and convicts breaking rocks with hammers. This practice is still carried out for profit in many African countries, not by slaves or criminals, but by private individuals operating the most basic of "quarries". Heading south out of Lusaka, Zambia it is possible to see several hundred men and women breaking rocks by the side of the road. Although authorities have made efforts to stop this practice, as "it gives tourists the wrong impression" the practice continues.

Many attempts were made to mechanise the rock breaking process, some successful and some not. Roll crushers were used in the UK as early as 1806 and stamping mills, or mechanised hammers were patented in the USA in 1830, and by 1840, an impact crusher had been developed. In 1858 Eli Whitney Blake developed a double toggle jaw crusher, these machines are still manufactured and are still commonly known as the Blake action crusher. After Mr Blake's invention, came the Gates gyratory crushers, these 1878 crushers are recognisably similar to modern crushers. The cone crusher was patented in 1922 by Symons and is still in production. If the inventors were still alive they would recognise that modern crushers remain similar to the ones they developed many years ago. This does not mean that crusher development has stood still, it just proves that some of the original concepts were very good. Since those early days, crushers have developed into an amazing range of machinery, employing many different methods of stone reduction, for example: single toggle jaw crushers reduce the stone by abrasion, attrition, shear forces and compression; double toggle jaw crushers use only compression and shear forces; cone and gyratory crushers use all of the above and torsional stress; single and double roller crushers use compression; toothed roller crushers incorporate a shearing action and impact, horizontal and vertical shaft impactors apply dynamic energy to the rock to drive it into either anvils or other rocks in high speed collisions.

Generally, the basic data required to chose a crusher is;

- Type of material to be crushed
- Maximum feed size, across corners
- Size analysis of feed material

- Moisture content
- Plasticity
- Bulk density
- Swell factor
- Specific gravity
- Hardness,
- Crushing work index,
- Abrasion index
- Compressive strength
- Friability
- Size of product, its purpose, shape and grading
- Fine product screened out before or after crushing
- Closed or open circuit
- Physical limitation to installation
- Throughput requirements

The large range of crushers ensures that a suitable machine can be obtained to break any type of rock. Generally, there is a trade off between;

- throughput and crushing power, often with the largest throughput coming from the crushers with the least breaking power
- cost and crushing power, the more powerful crushers require more investment in terms of capital and infrastructure costs
- maintenance costs, the most powerful crushers often cost the least to maintain.

Put simply, a large gyratory crusher will require more capital investment than a horizontal shaft impactor of similar throughput, in fact several impact type crushers could be installed fro the same price as one gyratory. However, the life of the impactor will be will be much less than that of the gyratory and the maintenance costs of the impactor will be much greater. Also, the maintenance rebuilds of the impactor will be required far more frequently than those for the gyratory. The main difference between the crushers is the application, an impactor will not cost effectively break hard, tough materials such as granite and quartzite, however, in relatively easily broken rock, an impactor will produce a better shaped and less stressed rock than a gyratory. All roller crushers break the rock by compression, they are only suitable for the softest of rocks, such as chalk and coal. A variation on this theme is the toothed roller, this a development of the coal or tunnelling rock shearer, they carry two horizontal rollers that are fitted with a spiral of teeth. The rock enters the crushing space and is thrown towards the centre of the rollers, at this point, the teeth should rip the rock apart by shearing force. When used in soft rock these machines have an incredible throughput, however, when used in slightly tougher applications, the writer has found them to perform badly, whereby instead of shearing the rock, the teeth chip small parts of it until it is perfectly circular, once this process is complete, the rock falls between the teeth and if operating to plan, the machine stalls, if not, it will break either its teeth or drive. With so many types of crusher available, it is likely that someone, somewhere, will make exactly the crusher that is required for a particular project.

13.1.2 Material

When considering a choice of crusher, a substantial knowledge of the characteristics of the rock is required. For example; is the rock hard or soft, tough or brittle, is it homogeneous or fractured. The mechanical strength of a rock depends largely on its constituent minerals, nature and type of cementation and degree of modification "heat and compression". Most igneous rocks are hard, both as a result of their constituent minerals such as feldspar and quartz because of the very tight bonding that occurred during cooling and crystallisation. Because they may be cemented by a weak material, sedimentary rocks are generally less hard even though they may contain hard minerals such as quartz, a typical example being sandstone, if the bonding material is hard, such as silica the rock may be very hard, a typical example being quartzite. Further information is required of the use that the crushed rock will be put to, for example, rock fill, concrete, asphalt, cement or lime. If lime production is to be considered, especially if a vertical kiln is to be employed, the rock must retain its strength even when it is heated to a point just below its melting temperature, to achieve this, it must not carry the type of internal stresses and damage within its laminations, that is often caused by gyratory and cone type crushers.

One of the first guidelines to assessing a material will be an analysis of its compressive strength. The following table gives typical compressive strengths and valuations. Measuring the compressive strength of a material involves applying a mechanical test to measure the maximum amount of compressive load a material can bear before fracturing. The test piece, in the form of a cube, prism, or cylinder, is compressed between the platens of a compression testing machine and subjected to a gradually applied load. Brittle rock may exhibit great compressive strength. Generally, compressive strength will increase with;

- degree of crystal intergrowth and content of mineral with high compressive strength

and reduce with;

- cleavage plains, micro joints, weathering

Although basalt and quartzite have very similar compressive strengths, this does not necessarily mean that they are similar to crush, in practice, basalt tends to be brittle and breaks easily, whereas quartzite is very tough and difficult to crush. Typically, limestones will fall between 2 and 7 on the Moh's scale of hardness, and have a compressive strength from 10MPa to 200 MPa. A dense limestone should have about 180 Mpa.

Figure 13-1 [ref 114 et al] Rock strengths

Rock type	Compressive strength kg/cm²	Compressive strength Mpa (MN/m^2)	Moh's scale	Valuation
Basalt Quartzite	2,800 to 3,276 2,569 to 6,411	200	7 plus	very high / hard extremely hard
Granite Limestone	1,845 to 2,777 1,621 to 1,968	120 to 200 180	6 to 7	high / hard
Gneiss Sandstone Marble	1,498 1,335 to 1884 1,193 to 1968	60 to 120	4.5 to 6	medium
Porous limestone	400 to 800	30 to 60	3 to 4.5	low / soft
Chalk	200 to 400	10 to 30	2 to 3	very low / soft

The breaking characteristics of a rock describe its behaviour when struck by a hammer, each rock type having a typical pattern of failure dependent on its texture, mineral composition and structure. The following chart gives the Los Angeles co-efficient which is a relative measure for determining the resistance of rock to breaking, the lower numbers give a greater resistance to breaking.

Figure 13-2 [ref Anon] Rock breaking characteristics

Rock breaking characteristics (Los Angeles value)					
Phyllite	17	Mica gneiss	45	Grey granite	53
Amphibolite	19	Mica schist	48	Granite (general)	56
Diorite	24	Gabbro	51	Limestone	66
Quartzite	36	Granitic gneiss	52	Pegmatite	71

A materials work index is a useful guide to its crushing characteristics, whereby the larger the number, the more resistant it is to crushing. The work index is the amount of energy given in kWh needed to reduce one ton of rock from a theoretical infinite size to 80% passing 100 micron. Abrasiveness is a time dependent parameter and a rocks abrasion index refers to the ability of the rock to cause corrasion, or erosion by friction, whereby the greater the number, the more abrasive will be the rock (this often relates to its silica dioxide SiO_2 content).

Los Angeles abrasion value (ASTM C131) describes the ability of a material to abrade a surface, this generally is an indication of the silica or quartz content. Concrete aggregates are required to have a maximum allowable abrasion value of 50% and good limestones should give about 25%. The test consists of a 5 kg sample of dry clean aggregate being accurately graded, placed in a container to be rotated at 33 rpm for 500 revolutions, screened on a number 12 sieve. The retained material is weighed, deducted from the original weight and the result divided by the original weight to give a result.

Figure 13-3 [ref 114 et al] Rock details

Material	Work index	Abrasion index	Silica content %	S.G.
Amphibole	16.30 plus or minus 3	0.20 to 0.45	0 to 5	
Andesite	16.00 plus or minus 2	0.50 plus or minus 0.10		2.84
Basalt	18.00 plus or minus 4	0.20 plus or minus 0.10		2.89
Diabase	19.00 plus or minus 4	0.30 plus or minus 0.10	0 to 5	2.70
Diorite	19.00 plus or minus 4	0.40	10 to 20	2.78
Dolomite	12.00 plus or minus 3	0.01 plus or minus 0.05		2.82
Haematite	11.00 plus or minus 3	0.50 plus or minus 0.30		3.76
Magnetite	08.00 plus or minus 3	0.20 plus or minus 0.10		3.88
Ferro-Silicate	11.00 plus or minus 2	0.25 plus or minus 0.07		4.91
Gabbro	20.00 plus or minus 3	0.40 plus or minus 0.10	0	2.83
Gneiss	16.00 plus or minus 4	0.50 plus or minus 0.10	15 to 20	2.71
Granite	16.00 plus or minus 6	0.55 plus or minus 0.10	20 to 35	2.68
Greywacke	18.00 plus or minus 3	0.30 plus or minus 0.10	10 to 25	
Hornfels	18.00 plus or minus 3	0.70 plus or minus 0.20		2.82
Limestone	12.00 plus or minus 3	0.001 to 0.03	0 to 5	2.69
Marble	12.00 plus or minus 3	0.001 to 0.03	0	2.73
Porphyry	18.00 plus or minus 3	0.10 to 0.90		2.73
Pyrite ore	10.00 plus or minus 3	0.60 plus or minus 0.20		3.48
Quartzite	15.00 plus or minus 3	0.75 plus or minus 0.10	60 to 100	2.71
Sandstone	10.00 plus or minus 3	0.10 to 0.90	25 to 90	2.68
Syenite	19.00 plus or minus 4	0.40 plus or minus 0.10		2.73

The Deval abrasion test consists of a 5kg sample broken into about fifty pieces. The sample is placed in a large cylinder angled at 30 degrees to its axis of rotation, this results in the sample being thrown twice from end to end during each of 10,000 revolutions. The sample is then screened over a number 12 sieve and the amount passing is expressed against the original sample as the percentage of wear.

Actual crushing value (ACV) is an old style test of the strength of aggregates and because the fines can become compacted in the test, weak aggregates tend to show a better result than is true. A figure of less than 12 is very good. For use in asphalt 28 is permissible and for concrete, 35 is permitted. Good limestones will give a value of 25 or less.

The 10% fines value test, is an accurate test of the strength of rock than its (ACV) with an increase in the value of the number (pressure) showing an improvement in the rock strength. In this test, the rock is subjected to an increasing pressure until 10% of the rock has reduced to specified size of fines. A good limestone should give a value of 180 kN or more.

The Dorry hardness test consists of a 25mm cylindrical core of rock being subject to the abrasive action of quartz sand being thrown from a disc, after 1,000 revolutions of the disc the end of the sample is worn away and the result calculated in inverse ratio to its hardness.

The US test of toughness (ASTM D-3) consists of a 25mm high, by 25mm diameter cylinder of rock being stressed by the application of a 2kg hammer. The energy of the blows is progressively increased by raising the starting point of the hammer by 10mm after each blow. The height of fall in centimetres at point of failure is the register of toughness.

Figure 13-4 [ref 114 et al] Rock toughness values

Typical toughness values from a drop hammer test					
Rock type	Value	Rock type	L. A. Value	Rock type	L. A. Value
Dolomite	1.00	Granite(general)	1.50	Quartzite	1.90
Limestone	1.00	Chert	1.50	Diorite	2.10
Andesite	1.10	Gabbro	1.60	Sandstone	2.60
Granitic gneiss	1.20	Basalt	1.70	Fresh diabase	3.00

The UK test of aggregate impact values determines the toughness of the aggregate and is similar to the 10% fines test, the main difference being in the application of the loading, in this case consisting of the impact of a specified standard hammer falling 15 times under its own weight upon a cylinder containing rock. The BS 882 1983 limits of aggregate impact values for use in concrete are up to 25% in normal concrete. Good limestones should give a value of 20% or less.

13.1.3 Work studies

The following are descriptions of various types of crushers that are manufactured and the specific crushers that are used in the case studies.

- Chilanga cement works operates a gyratory primary crusher and a single shaft fixed hammer impactor as a secondary unit.
- Ndola cement works operates a single, high reduction, multi-shaft impact crusher.
- Portland Cement company operates a primary jaw crusher an a single shaft fixed hammer impactor as a secondary unit.

Of all these crushers, by far, the most effective is the multi-shaft primary impact crusher that is installed at Ndola works.

- Ndola Lime company operate a large single toggle primary jaw crusher, a secondary cone crusher and a tertiary cone crusher.

The requirements for a vertical lime kiln are different from those of a rotary cement kiln and the crushers cannot be directly compared, however, the crushers at Ndola can be seen to be compression units, and are probably the worst choice for this application.

13.2.0 GYRATORY CRUSHERS

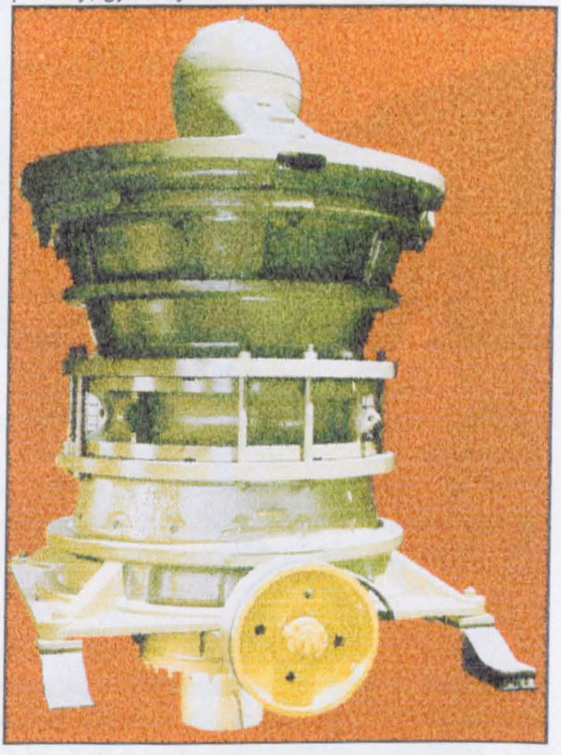
13.2.1 Description

The term "gyratory crusher" refers to a family of both primary and secondary crushers where the head of the crushing ring and the crushing cone are inverted and the cone is supported. This definition can become slightly blurred when used to describe secondary crushers, where the head of the unit is much shorter. The crushing is similar to a mortar and pestle.

The main components of a gyratory crusher are;

1) The stationary crushing ring or outer cone: This is an inverted frustum, or elongated cone that funnels out at the top, it can clearly be seen on the image opposite as the dark gray section. The general slope of the cone is not linear, with the greater angle from vertical being towards the top of the machine. The bottom of the cone is open to allow the rock to pass through and at this point if found its narrowest diameter. The cone is fixed to and within the outer shell of the machine and in some applications, the cone is the outer shell. On larger crushers the cone is made up of many segments. These segments are usually cast or forged from manganese steel alloy.

Figure 13-5 [ref 116] A large capacity Svedala primary, gyratory crusher.

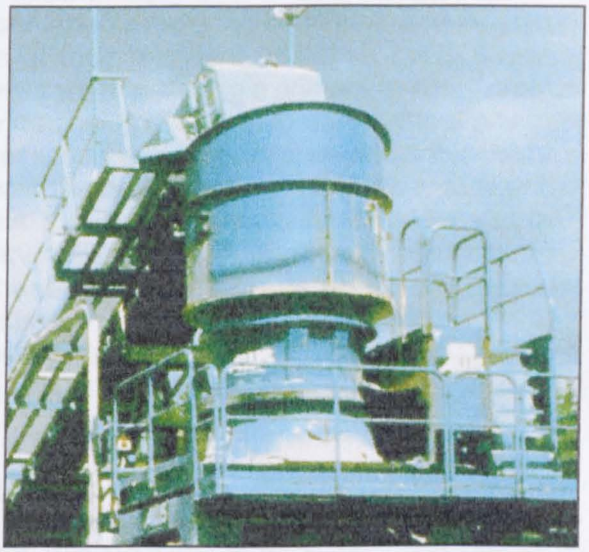


2) The male or inner cone: This unit is also a frustum, but in this case, the cone is upright. It is mounted on a shaft that sits within, but is not fixed to an eccentric bushing, the whole unit is partially suspended from a ball and socket joint that is mounted in the very top of the crusher and straddling the feed opening. This device is known as the spider and commonly has two or three support arms. The shaft is able to rotate freely within the bushing and the bushing rotates at between 100 and 250 rpm. The rotation of the eccentric bushing causes the crushing cone to perform (as

seen from the top) a gyratory motion. A typical angle between the eccentric bushing and its vertical plane is 3 to 5 degrees and a typical throw (the lateral distance moved by the inner cone) can be up to 70 mm. Because of this the shaft mounted male cone will approach and recede from the stationary crushing ring, thus reducing any rocks which are trapped between the two cones. The eccentric movement is greater at the discharge, or lower end of the crusher

3) The drive mechanism. This is simply, a crown wheel mounted horizontally close to the base of the crusher that is supported on either bronze bushes or roller bearings and on which is mounted the eccentric bushing. The drive to the crown wheel is provided by a roller mounted shaft and pinion that is directly powered by the energy source, usually an electric motor. Lubrication is provided by an external source of oil that is cooled and pumped around the machine. Often the oil will pass by a device that can detect metal particles and therefore provide warning of impending major failures, also advantage can be taken on a regular basis to analyse the oil for bronze in order to estimate wear on the bushings. Note; a frustum is any part of a solid shape that is contained between two parallel planes, in this case a cone.

Figure 13-6 [ref 116] A Svedala gyratory secondary crusher.



13.2.2 Applications

Gyratory crushers are generally used as primary crushers in hard rock applications, where they are able to reduce hard and tough rock. They can be designed with special liners to give a very reasonable wear rate. The throw of the male cone can be altered by replacing the eccentric bushing to give greater or less crushing pressure for a given power input, or alternatively greater or lesser throughput. On the basis of energy consumption, a large gyratory crusher is considered to 300% more energy cost effective than a jaw crusher, this is largely because the crushing action does not require several tonnes of jawstock to be lifted for each stroke of crushing. When idling, a large gyratory will use only 30% of its full load energy compared to the 50% required by a jaw crusher.

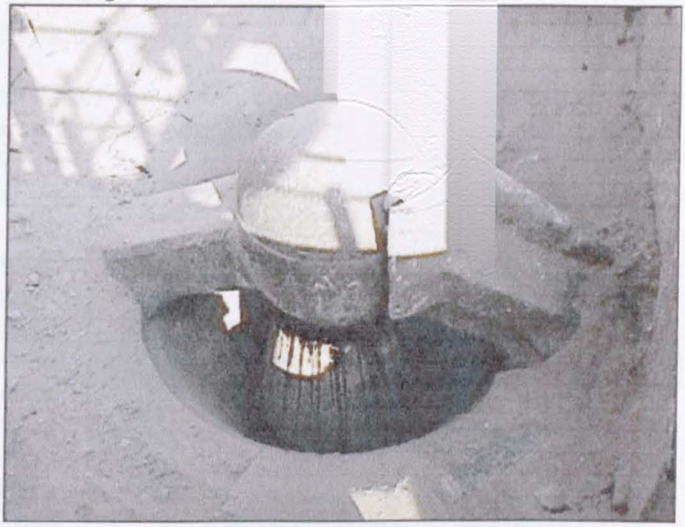
Because of the greater available crushing area of a gyratory crusher over a jaw crusher of similar weight, throughput is considerably higher. This generally means that the economics of operating a gyratory crusher over a jaw crusher becomes more attractive in high throughput operations and fixed crushing units are now produced with capacities of more than 6,000 tonnes per hour. Despite their requiring the most comprehensive of support structures and therefore high capital costs, having low wear rates and being able to crush the hardest of rocks gives large gyratory crushers a major advantage over many other crushers. Because of this, gyratory crushers are often the preferred primary units in hard rock applications. Operators working with lesser durable rocks will sometimes use a gyratory crusher to take advantage of their excellent operating costs and long service life. Cost can be saved when installing a primary gyratory crusher by taking advantage of the machines ability to operate without any form of feeder. Typically, a hopper is built around the upper part of the machine and the trucks tip directly onto the bearing spider. A disadvantage of this being that any small material that would normally be screened out by a grizzly to by pass the crusher will now pass through the crusher, thus reducing throughput. In a hard rock application, operating with a feeder and grizzly, blockages are infrequent, however one of the risks associated with direct feeding and crushing softer rock, is that in wet conditions the material may be pulverised to build up between the cones and eventually block the crusher.

13.2.3 Chilanga Cement Works

The original requirement for this gyratory crusher was approximately 200,000 tonnes per year, this has since increased to 454,000 tonnes per year. Operating 250 days per year this gives a daily throughput of some 1,800 tonnes per day.

The primary crushing plant was installed in 1956 and is a good example of the long service life that can be expected from these units. The crusher is a 30-inch Superior McCaully gyratory that was manufactured by Allis Chalmers of Milwaukee, USA. Since its installation, the machine has on average been rebuilt every five years. Power is provided by a 110Kw slip ring induction motor, turning at the relatively slow speed of 975rpm. In this installation, the rock is fed by an apron feeder to a two roller grizzly, this gives an even flow and rejects the material that is already below grade size. Several assessments have been made to judge the throughput of the crusher and it is estimated that in this application, a capability of crushing a little more than 100 tonnes per sixty minutes continuous operation is reasonable. One of the problems with throughput is the size of the gape (the entrance to the crusher that is visible in the image) which is 30 linear inches, giving an total area of 4,300 square inches. When the plant was installed, the loading of the run of quarry material would have been carried out by a small rope shovel with a bucket size of about one cubic metre and the fragmentation of the blast would have been appropriate to the machine. Nowadays the loading is carried out by a front end loader fitted with a five cubic metre bucket and to avoid blockages the rock must be blasted to produce total fragmentation with no rock being larger than the gape of the crusher.

Figure 13-7 [ref Mills] An overhead view of the primary crusher at Chilanga Works.

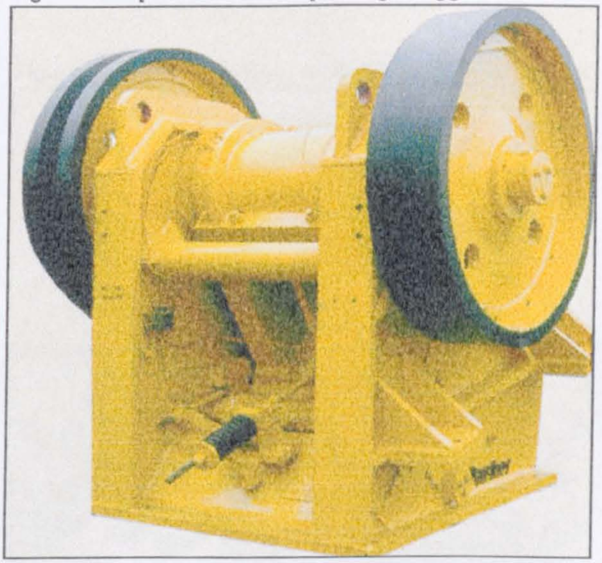


13.3.0 JAW CRUSHERS

13.3.1 Single toggle jaw crushers

These machines are often the most favoured and applicable primary reduction tool, they are found operating throughout the world in all applications, from crushing the softest sedimentary, to the hardest igneous rocks. They are able to accept a lump size equivalent to 90% of the feed opening. The average angle between the jaws of a modern single toggle jaw crusher is 20 to 25 degrees, this will give a reduction ratio of between 4:1 and 6:1, it is often this large reduction ratio that makes them attractive as primary crushers. The size reduction of the feed is carried out between two jaws, one moving and one stationary. The moving jaw moves not only backwards and forwards but also up and down, this being more pronounced at the bottom of the jaw. The rock is mostly reduced by compression and typical crushing values of 345 N/mm^2 can be attained. With this type of machine the jawstock is suspended from an eccentric shaft and is braced by a toggle from the back plate so that the jawstock moves through an elliptical path. The jawstock will conduct most of the crushing action and throw near the top of the jaw. The compressive force will be directly related to the power of the drive motor and the inertia of the flywheels. The movement of the lower parts of the jaw will be largely up and down, in abrasive conditions this causes the jaws to wear quickly. The vertical movement of the jaws helps to feed the crusher and assists in avoiding blockages. Because the whole of the jawstock is lifted in each revolution, a large drive motor is required. Typical energy requirements for crushing one cubic metre are between .35 and .75 kWh.

Figure 13-8 [ref Parker Plant] A single toggle crusher.



In the past, jaw crushers were manufactured from massive castings, to achieve the required strengths, these castings were heavy and cumbersome, in recent years, considerable advances have been made in the design and construction with much of the castings being replaced by fabrications and cast steel sections, this has considerably reduced the weight and increased the strength; the crusher shown above is of fabricated construction. Modern designs have brought about cheaper construction, reduced weight, easier maintenance, and improved component and wear part life. A computer aided design of the eccentric throw, angle of pitman and toggle geometries has, improved crushing force and enabled a flatter elliptical pattern at the discharge end of the jaws. The results of these improvements are, less wear, a better throughput and a smaller power requirement. The jaws are fitted with serrated lining plates which are cast from austenitic manganese steel with an average Mn content of 14%. When crushing limestone, the wear rate will be between 5 to 15 grams per tonne of rock crushed. A rule of thumb to throughput is 0.1 tonne per hour/per square inch of the feed opening. The crusher setting is made manually by closing the gap at the bottom of the moving jaw, this can be a time consuming occupation and generally, adjustments are made to compensate for wear, rather than to alter the gradation. Usually, the output grades down from twice the crusher setting.

Figure 13-9 [ref Kue Ken] A double toggle jaw crusher

13.3.2 Double toggle jaw crushers

Double toggle crushers are designed for use in hard rock applications where the rock has a work index greater than 25, they have more compressional force than a single toggle and better wear characteristics. The main design difference between these and single toggle crushers is the detail of the jawstock. With this design, the moving jaw is suspended from a circular shaft, around which it pivots. The movement of the jawstock is provided in a horizontal plane by a Pitman (cam shaft) turning between two toggle plates. The toggles form a link between the Pitman, the frame of the crusher and the moving jaw. Movement of the jawstock is in an arc extending from the pivot with very little lateral movement around the axis. The maximum travel being at the base of the jaw. There is lower energy consumption when compared with a single toggle crusher, this is because of not having to lift the complete jawstock with each revolution, also fine balance can be achieved in the design to minimise the amount of energy expended in moving steel as opposes to crushing.

Possibly, the best design of this type of crusher is produced by Kue Ken, where the pivot is ahead of the crushing chamber and the jawstock is finely balanced. As there is no vertical movement in the crusher, the reduction is solely by compression, this produces a clean cut with a minimum of fines production and less wear. The disadvantage of not having any vertical movement is that the crusher is not self feeding and consequently, for a given opening a single toggle crusher will have a much larger throughput, in some instances this can be double that of a double toggle machine. The most common double toggle crushers are, the original Blake design and the Kue Ken, with the Blake being much less sophisticated than the Kue Ken but perhaps being more rugged. Tramp metal protection is provided only on some crushers and even then its effectiveness is questionable, The safety device consists of a spring coupling device that connects the flywheels to the eccentric drive shaft, the principle is that, should the crusher be subjected to shock loading, the springs will move out of place and allow the flywheels, together with the drive motor to disengage from the drive shaft and by so doing, allow the crusher to rapidly stop.

The crushers that are most likely to need this device are all the smaller designs and more often than not these units are the least likely to have protection fitted.

13.3.3 Ndola Lime Company

The crushing plant was purpose designed by Bateman of South Africa and built in 1973, the major crushers date from that time. The primary crushing plant consists of a apron plate feeder, feeding to a primary jaw crusher, unusually, no grizzly or scalping screen is fitted to remove the fines prior to primary crushing. The most important criteria for the effective operation of the vertical kiln, is that the rock should be as cubical as possible and free from internal cracking and stresses. These requirements are best catered for by impact shearing instead of compression crushing. For reasons that where not made clear to the writer, all the crushers in this unit reduce the rock by compression, therefore, it is likely that the existing combination of jaw and cone crushers is probably the worst that could have been placed in this complex plant.

Primary crusher

Primary crushing is carried out by a 60 by 48-inch, Pegson Telesmith, single toggle jaw type crusher, it is one of the older designs and consists of very heavy castings, this is emphasized by its massive mounting and foundations. The crusher house is of massive construction and would easily house a unit with a much greater capacity. Typically this size of crusher would be operating with a closed side setting of 200 mm or more, giving a throughput of up to 800 tonnes per hour. In this application, the closed side setting is 112 mm and the throughput is a little more than 300 tonnes per hour. Considering the size of the crusher and its infrastructure, the throughput is very low. A similar sized impactor would have a throughput of approximately 1,000 tonnes per hour. Considering the lack of abrasiveness of the limestone, the requirement for a cubital product with thermal integrity, and the ready market for fine material, the choice of a large jaw primary crusher must be questionable. It would be reasonable to assume that if the crusher suffered a major failure, it would then be replaced by an single rotor impactor.

13.3.4 Portland Cement Company

Until 1996 the primary crushing was carried out by a 30-x 42 inch, Osborne Telesmith, single toggle jaw crusher, mounted on a steel chassis that was once mobile. The crusher was of uncertain vintage but certainly of the older generation with massive castings and a wide nip angle. The nip angle was reduced partly by fitting convex liners. Operating with these liners gave an incorrect shape to the crushing chamber, resulting in a low throughput and a high percentage of fines. The condition of this crushing plant was appalling and despite appeals made by the writer to general management to improve safety standards, an operator was killed while cleaning the main product conveyor. Even after this death, the crusher continued to operate in the same condition until it was scrapped by the writer in 1996.

The replacement primary crushing plant was manufactured by Svedala and featured one of their excellent single toggle crushers. These modern machines are of light construction, incorporating both castings and fabrications, they have a larger than normal throat and a fine nip angle.

13.4.0 IMPACT CRUSHERS

13.4.1 Horizontal shaft impact crushers

This family of machines are often found in medium to light duty, low abrasive applications as high throughput primary crushers. Other applications include, secondary and tertiary crushing, where the rock requires a cubital shape. They are widely used for reducing limestone and other relatively easily broken materials. Impactors offer, low capital costs and high throughput, this can be at the expense of high maintenance cost and a relatively short working life. The crushers basically consist of a horizontal shaft (rotor) that is fitted with replaceable hammers, these are in the form of a heavy cast alloy bar located to extent beyond the diameter of the rotor. The whole of which is mounted in a steel box that is lined with an impact and wear resistant material, At 90% to the axis of the shaft are mounted breaker bars or anvils, these are heavy metal castings that form a surface for impact. The entrance to the crusher is typically rectangular and large when compared to a jaw or gyratory of similar throughput, a heavy chain curtain is suspended from the top of the entrance to prevent rocks from being throw out of the machine. The rotors in a single shaft primary application spin at between 250 and 600 rpm, the Cedarapids plant shown above spins at 550 rpm.

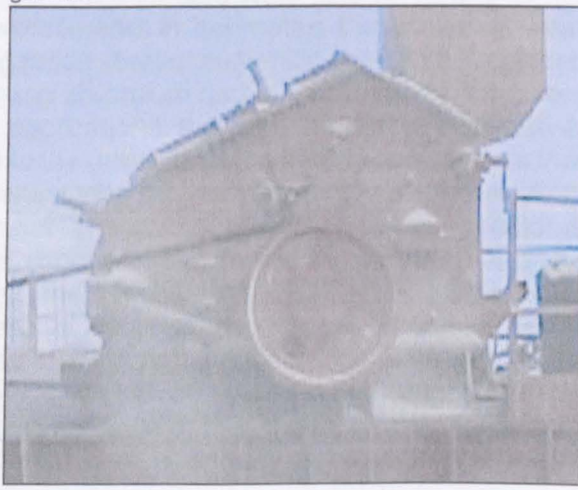
Figure 13-10 [ref Mills] A Cedarapids 4338 single rotor, two hammer impact crusher. Installed by the writer in Jamaica for lime production. The inspection door is open, notice the adjustment panel to the right of the door, the bolts holding this panel are sacrificial and designed to break in case of the ingestion of tramp metal. The chain is to prevent the panel from departing the scene.

Figure 13-11 [ref Mills] Rotor and impact hammer in Cedarapids crusher.

Reduction by impact is a dynamic operation where size reduction ensues in three procedures;

- primary size reduction is by the impact of the hammers against the incoming feed
- secondary reduction is caused by the material striking the breaker bars
- tertiary crushing is caused by collisions of rock within the crusher

Figure 13-12 [ref Mills] A Svedala secondary impactor crusher operating in St. Kitts, note the dust and lack of guards.



Primary impactors often operate with reduction ratios of up to 10:1 per rotor and are often found to be replacing more than one stage in the crushing circuit. They are made in a variety of designs with single, multiple, horizontal and vertical shafts, with pivoting and fixed hammers. They all operate by the rock being struck by hammers with typical tip speeds of 25 to 50 metres per second. These are mounted on a rotating shaft to strike the rock and impart sufficient kinetic energy to either break the rock or project it against breaker bars "or anvils" to be shattered and rebound to be struck again by the hammers in a continuous process until the sized particles are unable to react effectively to the hammer blows and are swept from the crushing chamber. Breaking rock by this method causes immediate fracture, usually along natural cleavage lines, with little retained residual stress or sub fractures, this has obvious advantages in applications where the rock needs to retain its strength during periods of high torsional stress or heat variation. Gradation of the product can be changed by altering the rotor speed, or by adjusting the gap to the breaker bars.

Large multi shaft combined impact and hammer crushers, installed in a single stage, single pass operation are often able to produce a reduction of run of quarry material to 95% passing a 25 mm sieve. In this type of crusher, the feed of up to two cubic metres falls onto two slowly rotating rollers which transport the material to be crushed into the crushing chamber. The slot between the rollers acts as a gap to scalp the undersize fraction from the feed. The rotors run relatively slowly at between 250 and 400 rpm giving the hammers a circumferential speed of 40 metres per second. The energy requirement for a 95% reduction is

1kWh per metric tonne.

The effectiveness of impact crushers is dependent on the feed characteristics, which are;

- compressive strength less than 250 N/mm
- Mohs' hardness less than 4.5
- moisture less than 20%
- silica content less than 15%
- clay content - nil

The above conditions will give an expected wear rate of 10 grams per tonne. In recent years, developments in the metals that make up wear parts has enabled the manganese steel or austenitic steel, to be replaced by nodular cast iron containing chromium carbide, thus giving impactors the ability to operate in applications that are moderately abrasive. Unfortunately, the changes to the make up of the hammers and liners that make them more wear resistant also makes them brittle and more likely to be damaged by tramp metal. The normal protection for an impactor is to have some of the anvils spring loaded, this is incorporated to allow tramp metal to fall through the crusher. Although good in principle, impactors often suffer massive damage through the action of metal travelling at tip speed around the crushing chamber. Many of these machines have been completely written off by an excavator tooth rattling around the crushing chamber, because of this, the input material should be screened by a metal detector, in a primary application this is not possible.

A problem associated with impactors, particularly when used in secondary and tertiary applications, is the amount of dust that they produce, this often gives environmental as well as production problems. Slowing the rotor speed will reduce the amount of fines but this type of crusher will always produce more of this often unwanted product than other types of crusher.

A variation on the horizontal shaft impactor is the vertical shaft family of impactors (VSI). These crushers are used in secondary and tertiary applications, they reduce and re-shape the rock by attrition. In the most successful application of these crushers, the rock is projected against other rock instead of against an anvil, or in one particular instance, the rock is projected against a curtain of falling rock. Because of its method of reduction, a typical VSI can handle rocks with an abrasive ratio of up to 80%. The machine consists of a hollow rotor spinning about a vertical axis, the rotor may be up to one metre diameter and 300 mm deep. Rock enters the rotor from the top and is accelerated by the fast spinning rotor (up to 3,000 rpm) to emerge at very high speed to impact rocks that are either falling past the rotor openings (ports) or to strike material that has previously been thrown onto a shelf that circles the rotor. The design of the rotor, its diameter, its number of openings, its speed, the distance to the shelves and the shape of the rock residing in the shelfs all have to be calculated and specified for any particular application. Most of the major manufacturers have test crushers that can be spun at any speed and easily fitted with a wide range of rotors. The main wear parts to this machine, are tungsten carbide tips that sit on the edge of the ports, these tips can be replaced easily by removing the old units and 'dropping' replacements into place.

A major advantage of this type of reduction is the ability to operate a selective reduction regime, with the less durable material being reduced and the stronger material remaining. This feature is often utilised in ore reduction. The rotor in a VSI may spin at speeds up to 3,000 rpm.

13.4.2 Ndola Cement Works

This quarry operates a large multi rotor, impact crusher produced by FLS

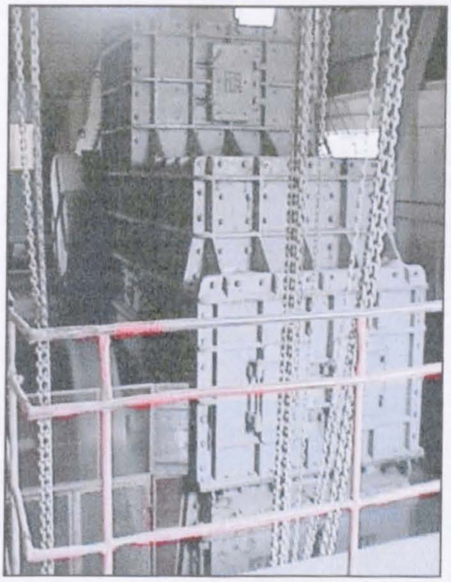


Figure 13-13 [ref Mills] Multi-rotor impact crusher at Ndola viewed from the left side

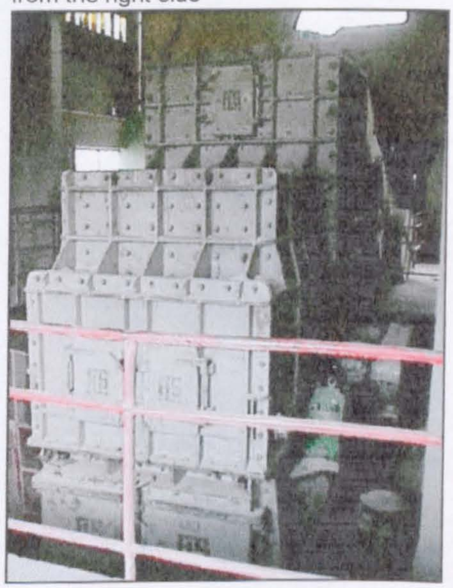


Figure 13-14 [ref Mills] Ndola crusher from the right side

of Denmark. This type of crusher is typically found in modern cement works and is of substantial proportions. Comparative to its throughput it requires massive infrastructure. In front of the crusher is a large hoper that can accommodate the load from two or more forty tonne capacity dump trucks, the rocks are transported from the hopper by an inclined apron feeder. Typically, the size of the run of quarry rock will range from dust, to blocks of one cubic metre and output from the crusher is minus 50mm, this gives a reduction ratio of 10:1 per rotor.

The crusher has four shafts, the first two act as a grizzly and also carry the rocks to the main rotors for reduction.

Although successful, this crushing plant is of massive proportions and relative to is size has a small throughput of about 300 tonnes per hour, an amount easily equalled by the Cederapids crusher shown earlier, albeit with a lesser reduction ratio.

13.4.3 Portland and Chilanga secondary crushers.

Both these plants use a copy of a Hazemag AP-SM single rotor impact crusher. The units are manufactured under license in South Africa and are a faithful copy of the original. The appearance of the units is similar to that of the Svedala crusher shown earlier, with a three hammer rotor and circumferential breaker bars, the crusher in Chilanga has proven to be very successful, with a minimum of repairs, probably due to a lack of maintenance, the Portland crusher has literally worn itself to uselessness. This crusher has required three replacements of the rotor, each being written off by the ingestion of tramp metal, the plant operates without the benefit of a metal detector.

13.5.0 CONE CRUSHERS

13.5.1 Description

Figure 13-15 [ref 120] Telsmith cone crusher. An advanced version of one of the Simons family of crushers. These crushers range in weight from about five to fifty tonnes.

Most cone crushers either are, or were, based on the machines developed in America by the Symons brothers these crushers were patented by Mr E. Symons and have proved to be exceptional in their operation and reliability, with some of the very earliest examples still working. As with all good designs they will be copied and many identical crushers have been produced by a host of companies, some copied legally and some not. The image on the right shows a reasonably modern version of the Simons crusher, this design was, and still may be, produced by the Telsmith Company. An important feature of this type of crusher, is the ring of coil springs, they are designed to allow the shell of the crusher to lift and release tramp metal. The term "cone crusher" refers to a crusher where the head of the crushing ring and the crushing cone are pointed in the same direction, much as, a closed bell within a bell, that is open at each end. This is in contrast to a gyratory crusher where the direction of the cones diverge.

Figure 13-16 [ref 120] Male cone and support

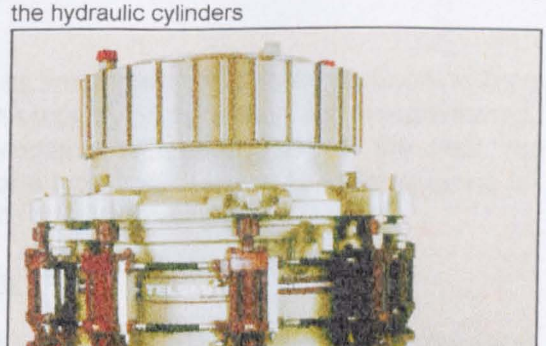
A cone crusher consists of a shell in which is fitted a stationary truncated female cone (crushing ring). Within the shell but independent of it, a male cone is mounted onto a freely rotating shaft. The shaft is vertically offset-mounted in a bronze bearing. The bearing is mounted in a crown wheel driven by a pinion gear. On Symons type crushers the top of the shaft is fitted with a device

Figure 13-17 [ref 120] Telsmith crusher shell. Showing the outer or female cone and the threads for adjustment

which is used to distribute the feed around the crushing chamber. The rotation of the crown wheel causes the male cone to perform a gyratory motion and because of this, the gap between the two cones (crushing chamber) opens and closes. Because of the relatively obtuse angle of the cones, the crushing chamber increases in area as the rock descends, thus allowing for the increased volume of the rock (swell factor). This design feature helps reduce the tendency found in gyratory crushers for the rock to become impacted and adhere to the cones.

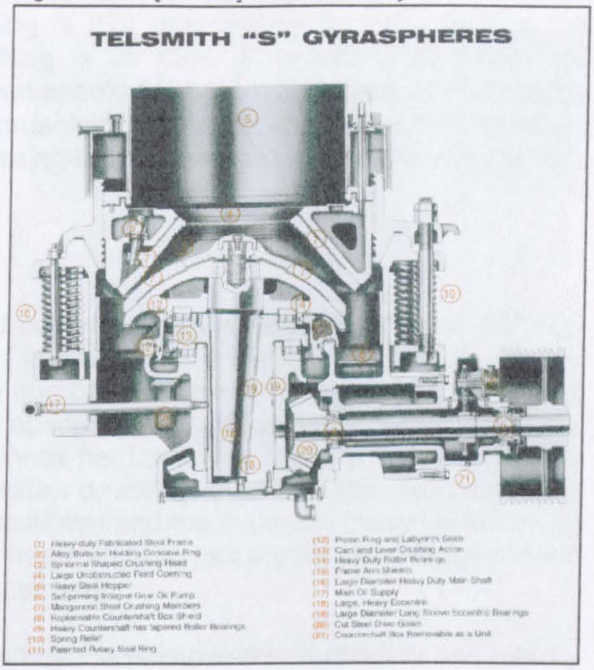
Compared with a gyratory crusher, cone crushers have a greater shaft offset and will rotate faster, this allows the rock to fall by gravity and be struck several times by the rising head of the cone before vacating the crushing chamber The frequency and amplitude of the crushing action can be adjusted by changing the speed of the drive and by altering the amount of offset in the bearing. The geometry of the crushing chamber can be radically altered by fitting any of a wide range of purpose designs of cones and mantles. The size of the crusher is designated by the diameter of the base of the crushing cone. In a secondary application, a reduction ratio of between 8:1 and 6:1 would be normal and in a tertiary application the ratio would be between 3:1 and 2:1. Adjustment of the Simons type design was achieved by screwing down the upper shell of the crusher, effectively reducing the gap between the inner and outer cones, this process was always time consuming and often very labour intensive and difficult. Moreover, adjustments could only be made when the crusher was not crushing (in theory).

Figure 13-18 [ref 120] A Telsmith crusher, showing the hydraulic cylinders



Modern designs use hydraulic systems for adjustment, either by moving the shell and outer cone, or by moving the shaft and inner cone, tramp steel protection on some modern crushers is provided for by replacing the outer springs with hydraulic cylinders, these are interconnected and pressurised by a gas over hydraulic accumulator system. On other systems, over-pressuring the hydraulic adjustment system will cause the pressure to be released and will lower the inner cone. The modern system of instant hydraulic adjustment can be made when the crusher is operating, thus enabling a facility to be applied which senses the amount of electrical load on the drive motor. Having this ability enables a further step to be made to a completely automatic system of adjustment and enhanced performance.

Figure 13-19 [ref 119] Telsmith "S" by section



Application

Cone crushers are often used as secondary and tertiary crushers in hard rock applications They crush the rock by compression and flexural stress, this can cause there to be problems with the quality of the rock, for example, should the rock be of a type that is prone to splitting along its bedding planes or in line with its crystal make up.

13.5.2 <u>Ndola Lime Company</u>

The Ndola Lime Company uses two cone crushers, one for secondary and one for tertiary crushing.

Secondary crusher

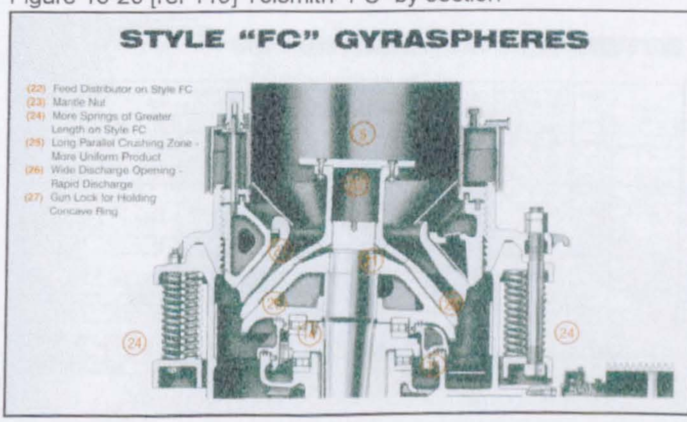
Secondary crushing is carried out by a 66x15-inch "S" type standard cone Pegson Telsmith gyratory crusher. The unit was installed in 1973 and is in good condition, it is found in the number two crushing house. Utilising a long throw spherical head action, and an operating closed side setting of 44 mm the throughput has been calculated at 99 tonnes per hour. The manufacturers show that with an coarse crushing chamber configuration the maximum feed opening is 279 mm closing to 254 mm and the minimum closed side setting is 25 mm. At a setting of 44mm the throughput in an open circuit should be more than stated, at 400 tonnes per hour. With a medium crushing chamber configuration the maximum feed opening is 228 mm closing to 203 mm and the minimum closed side setting is 19 mm.

Tertiary crusher

Tertiary crushing is carried out by a Pegson Telsmith 48-inch "FC" fine crushing gyratory crusher. The unit was installed in 1973, is in good condition and found in the number two crushing house. Utilising a long parallel crushing zone, and operating with a closed side setting of 10 mm gives a throughput of 56 tonnes per hour. The manufacturers show that when operating with a medium crushing chamber, the maximum feed opening is 75mm, closing to 48mm and the minimum closed side setting is 8mm, at a setting of 10 mm the crusher has a potential throughput in a closed circuit of 80 tonnes per hour.

Other than size, the main difference between the secondary and tertiary crushers is the inclusion of a feed distributer on the smaller unit and in the bearing support. Both units are adjusted by rotating the head of the crusher. The head of the crusher is locked into position by hydraulic rams, these can be released for adjustment, a further system of hydraulic rams can be used to turn the head, this system although reliable is obsolete. Lubrication is provided by oil pumped under pressure to the main eccentric and support bearings, to return through a cooler to an external tank.

Figure 13-20 [ref 119] Telsmith "FC" by section



Appendix A

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Appendix B

An Example of a Drilling Log

This is a typical core sample analysis it has been taken from a project in which the writer produced a feasibility study. The core has been logged and samples taken, they have been given a visual inspection and commented upon [ref 90].

NAJAYO CEMENT PROJECT

Borehole SN1O3

Borehole Summary

Type of Borehole	Vertical, cored throughout
Total Depth	60.00 metres
UTM Grid Co-ords	F382700 N2030750 (approx.)
Level of Origin	100 m above sea level (approx.)
Dates of drilling	14/08197-18/08/97
Drilling Contractor	Geocivil SA
Drilling Rig	Longyear 38
Driller	Juan Enerlo
Geologist	Lloyd Boardman (IMCL)
Cores logged by	Lloyd Boardman
Purpose of Borehole	To provide continuous cores for cement raw material evaluation
Core and hole size	Core: HQ
Drilling Fluid	Water and polymer
Casing	All recovered
Comments	Left open and capped, 60m 1.75 inch pvc pipe installed

Borehole: SN I03

Approx. location	E382700 N2030750
Approx. starting level	100 m (above sea level)
Coring from surface	(HQ size)

Partially weathered, weak, smooth, plastic yellow/brown CLAY 0.02 carbonate concretion at 0.58, occasional white calcareous veinlets and small rounded diffuse patches.

SAMPLE: SN 103/1 0.00-1.00

SAMPLE: SN 103/2 1.00-2.00

0.06 carbonate concretionary layer at 1.76, veinlets absent below, generally solid core recovery throughout with rare broken horizons, rare small red/brown irony concretions below 2.40 with some black spotty manganese mineralisation on occasional high angle joints. Becoming generally firmer and blocky below 2.50 with occasional small listric surfaces. 0.05 pale grey carbonate concretionary layer at 2.77.

SAMPLE: SN 103/3 2.00-3.00

SAMPLE: SN 103/4 3.00-4.00

SAMPLE: SN 103/5 4.00-5.00
45 degree angle joint at 5.48, high angle irregular joint 6.53 to 6.69
SAMPLE: SN 103/6 5.00-7.00
iron staining common on joints below 6.70 - 7.00 partially weathered, very weak, smooth pale yellow/brown CLAYSTONE, abundant iron staining and manganese patches, generally solid core recovery, some high angled joints, blocky in part, strong and calcareous with manganese mineralisation 7.60 to 7.75 and 7.92 to 8.02 and for 0.05 at 8.18, 28 and 35 degree joints common 8.70 to 9.40.
SAMPLE: SN 103/7 7.00-9.00
0.03 calcareous nodule at 8.32, rare grey/green streaks below.
45 degree listric joint at 9.60, grey/green streaks increasing in abundance below with rare pale yellow/brown silt laminae.
DIP 17 degrees at 10.75
SAMPLE: SN 103/8 9.00-11.00
0.035 strong calcareous ?concretionary layer at 11.25, becoming greenish below with abundant black manganese films on some joints. 0.04 strong pale grey irregular calcareous nodular layers at 12.56 and 12.74 with sandy CLAYSTONE between, some fine orange sandy laminae below
SAMPLE: SN103/9 11.00-13.15 13.45
Fresh to partially weathered, strong, pale grey nodular LIMESTONE with manganese mineralisation on open joints, irregular top and base, solid core recovery
SAMPLE: SN 103/10 13.15-13.30 13.30
Partially weathered, weak, pale brown, slightly silty CLAYSTONE with rare yellow/brown patches, solid core recovery
SAMPLE: SN 103/11 13.30-14.30 14.30
Partially weathered, weak to moderately weak pale brown CLAYSTONE, grey/green in top 2.00, generally solid core recovery except below I 6.20 for 0.15 with some broken core, some small pyritic aggregates in grey material and rare small shells, large irregular high angle joint iron stained joints and thin laminae at 16.30
SAMPLE: SN 103/12 14.30-16.35
Some broken and fragmentary core below 16.35 to 17.55, generally solid core below and stronger and siltier, occasional grey/green patches below 17.00, high angle iron stained joint 17.60 to 18.00, 0.02 pale grey, strong calcareous nodular layer with manganese mineralisation and Iron staining at 18.02
SAMPLE: SN103/13 16.35-18.35
Fresh, moderately weak to moderately strong, grey with green tinge CLAYSTONE, some pale brown patches in top 0.65, 0.07 pale grey irregular iron stained calcareous nodule at top, generally solid core recovery.

SAMPLE: SN103/14 18.35-20.35

thin black lamina at 20.70, rare comminuted shell debris in patches, below, slickensided joint 21.65 to 21.74

FAUNAL SAMPLE: FN001 at 22.45

SAMPLE: SN103/15 20.35-23.35

becoming slightly paler in colour below 23.65

SAMPLE: SN103/16 23.35-26.35

DIP 16 degrees at 28.55

FAUNAL SAMPLE: FN002 at 28.70

SAMPLE: SN103/17 26.35-29.35

FAUNAL SAMPLE: FN003 at 29.35

FAUNAL SAMPLE: FN004 at 30.05

SAMPLE: SN103/18 29.35-42.35 32.35

Fresh, moderately strong, smooth, greenish grey CLAYSTONE, occasional very small shells and shell debris, curved listric surface at 32.52, generally solid core but with fragmentary and broken core for 0.20 at 33.75, well preserved large ornate gastropod at 33.80, occasional small gastropods below.

FAUNAL SAMPLE: FN005 at 33.80

Core fragmentary for 0.15 below 33.80

SAMPLE: SN103/19 32.35-35.35

Re-drilled core for 0.14 at 35.85, generally solid core below except for 0.16 fragmentary at 38.90, generally weaker and blocky below 38.00

SAMPLE: SN103/20 35.35-38.90

Solid core recovery below 38.90, 0.085 concretionary calcareous layer at 39.54, slickensided joint 39.72 to 39.80, rare thin black lamina below 39.90 and generally paler in colour overall with some very fine silty laminae and occasional small concretionary calcareous nodules, 0.15 concretionary carbonate layer with associated listric surfaces at 41.24 and occasional thin layers below.

SAMPLE: SN103/21 38.90-41.90

becoming smooth in general below 43.50 again but thin silty horizons common below 44.00 and with more shell debris and becoming generally weaker

SAMPLE: SN103/22 41.90-44.90

45 degree slickensided joint at 45.00, thin calcareous concretionary layers and small nodules common in basal 0.85 46.15

Fresh, moderately strong, smooth, grey to dark grey with green tinge, CLAYSTONE, occasional very fine paler silty laminae.

DIP 17 degrees at 46.50

weak and plastic from 47.30 to 47.90

SAMPLE: SN103/23 44.90-47.90
Fragmentary and broken core for 0.10 at 49.20 and for 0.18 at 50.50, silty with some well preserved shells for 0.20 at 49.65
FAUNAL SAMPLE: FN006 at 49.65
0.02 fine sandy lamina at 49.85, generally weaker below 49.70 to 50.12, rare silty lamina below and moderately strong again.
SAMPLE: SN103/24 47.90-50.90
solid core recovery below 50.90 DIP 21 degrees at 51.00 irregular fine sandy laminae for 0.04 at 51.97, occasional convolute laminae below, ?sand filled sub horizontal burrow at 52.63, comminuted shell debris for 0.18 at 53.64
SAMPLE: SN103/25 50.90-53.90
55 degree slickensided joint at 54.12, shell debris common in thin layers below 55.20 and becoming darker and weaker. 45 degree joint at 55.40
FAUNAL SAMPLE: FN007 at 56.50
SAMPLE: SN103/26 53.90-56.90
very rare shell fragments below 56.90
SAMPLE: SNIO3/27 56.9O-60.00 60.00

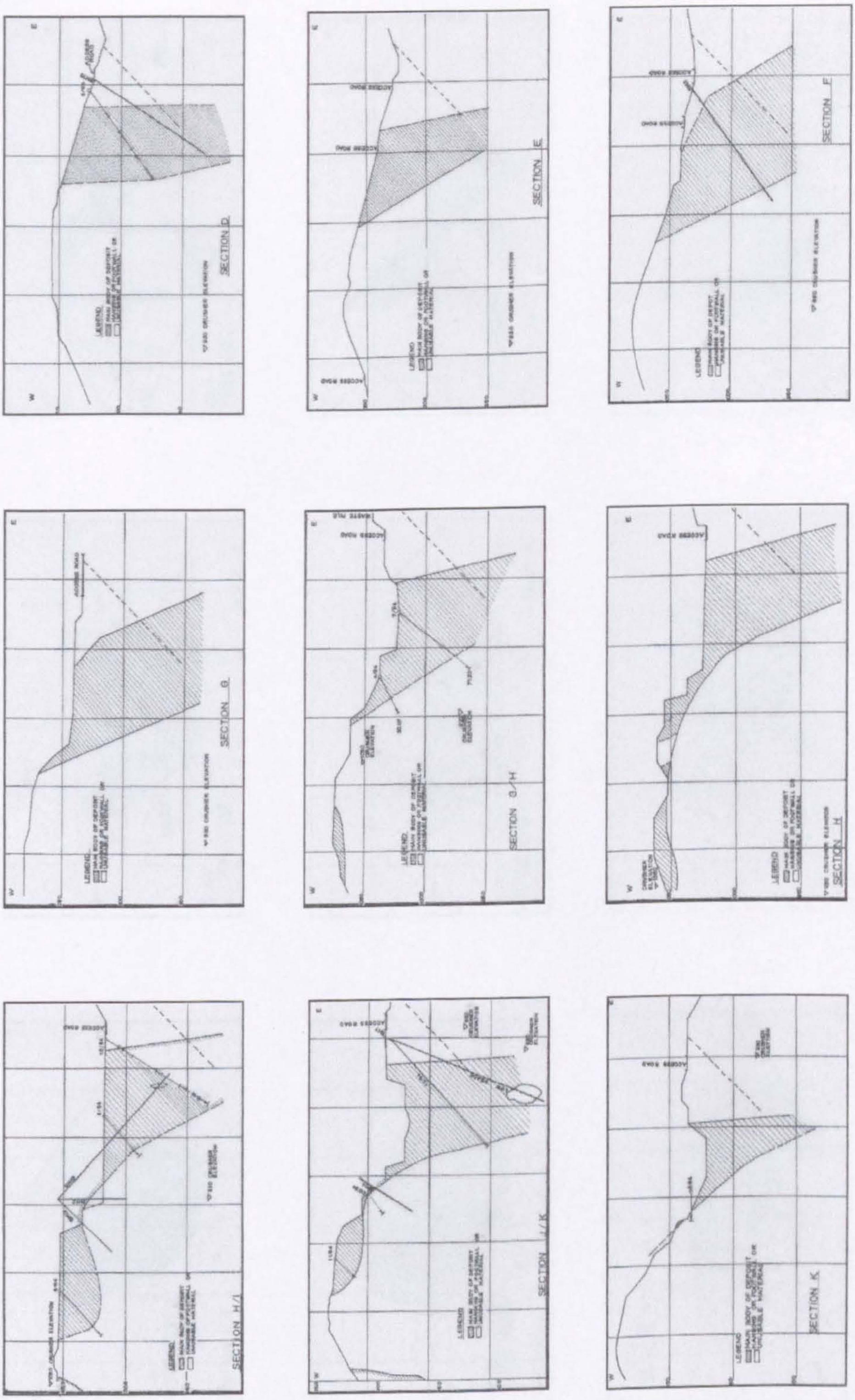
Total Depth 60.00m

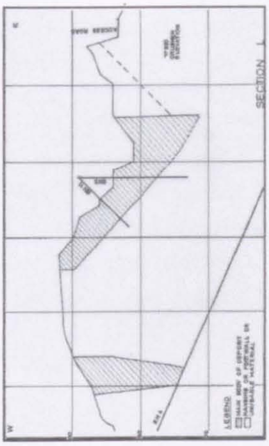
Borehole left open with locking cap and 60m 1.75 inch slotted PVC pipe inserted.
Borehole dipped 5/9/97 - water level 9.75m

Appendix C

Sections of Changalume Deposit

Sections of the Changalume deposit





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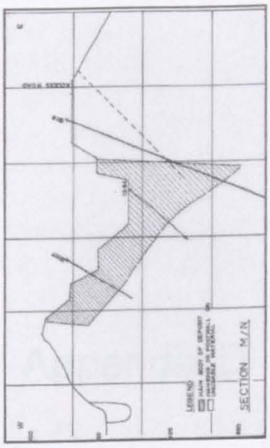

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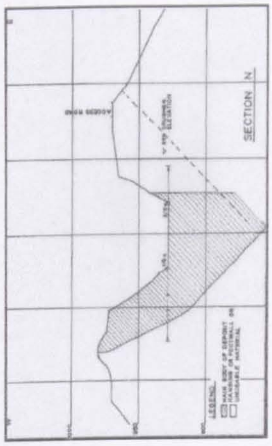

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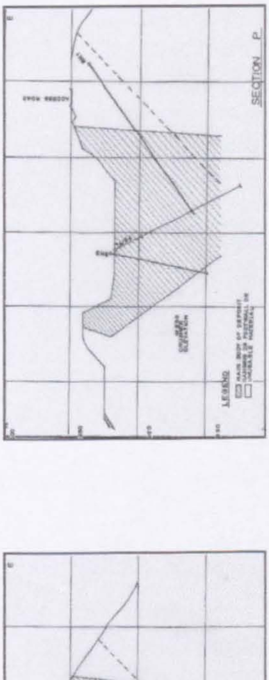

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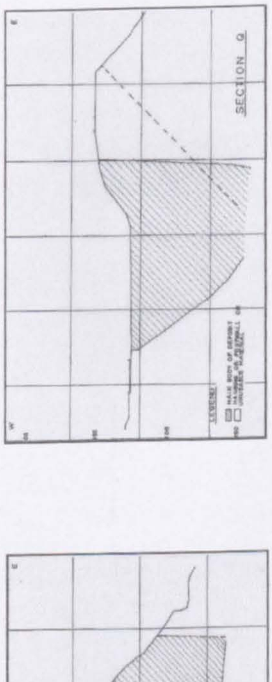

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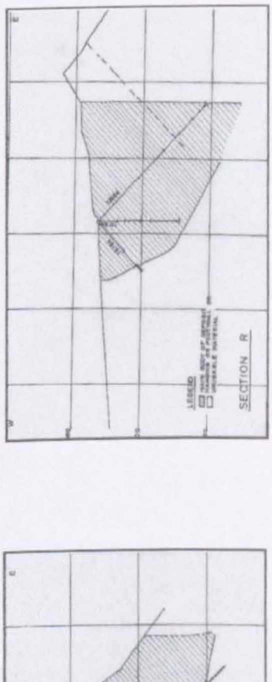

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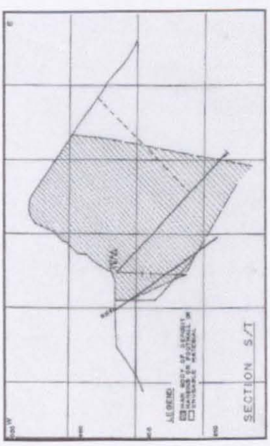

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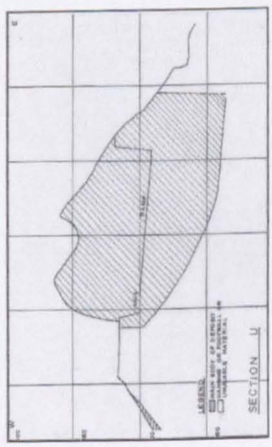

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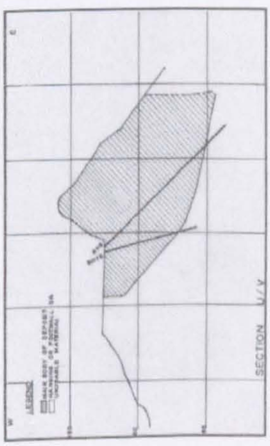


LEGEND
SECTION U/V

Appendix D

Computer Plotting

Computer plotting

In the past, the part of quarry planning dealing with plotting the resource and drawing the actual working plan was a manual process, however, the work has been now being speeded up and to a certain extent improved, by the introduction of computerised facilities. Nowadays most mining plans are prepared on a computer. The first stage of computerisation is to replace the hand drawn maps with a digitised computer image, these can be operated relatively easily and are in wide use, a series of quarry plans for a limestone quarry have been included in chapter 8, they show the original land surface and the progressive development of the project, in this instance the quarry begins at the top of a slope and descends in ten metre benches aligned with the natural contours.

Lerch and Grossman in 1964 published a 3-D algorithm based on graph theory which could be applied to determine the optimum design for a quarry. In 1986 Whittle programming Ltd utilised the algorithm for the first time in a commercial software package. The method produces a computer generated three dimensional block model of the deposit and progressively builds and identifies lists of related blocks which should or should not be mined. The model will display a pit outline to take advantage of the best possible pit slopes to achieve optimum viability from the deposit. A further facility, mainly of value to underground developments or surface extraction of thin seams is the ability of modern computer packages to draw isopachyte maps.

These maps are able to give a visual representation of the thickness of the seam and overburden, they can be produced from the direct input and interpretation of borehole date and operate on the "nearest neighbour" or "area of influence" practices. The maps can also be drawn to give a visual display of the quality of the deposit.

The picture on the next page shows a computer interpretation of an existing coal mine that is located in Indonesia, details such as the intermediate water sump which is situated in the top right hand corner can be seen, as can the haul road and access routes to the various benches. The image was produced by interpretation of information gained from a detailed site inspection that was logged into a computer operating a dedicated programme. This particular image was produced using software produced by Surpac Software Limited, other software such as the MOSS system which was developed from a highway cut and fill programme. Surfer, Datamine, Minescape, Eclipse, Lynx etc, are both powerful and versatile, they are available from specialist software producers.

A typical quarry planning programme will have the facility to;

- generate a grid
- produce a contour map (a two dimensional map of three dimensional data)
- generate surface plots (a three dimensional representation of a grid)
- produce image maps (maps of a grid with scales of colour representing values of height, density, grade of material or any other combinations required to define or simplify interpretation)
- produce a filled contour map which is a an abstraction combining or overlaying all the above plots
- Enable the quarry plot to be combined with country maps for accurate boundary control.

A computer image showing the overhead view from 3,000 metres. Visible are the benches, the final pit walls, entrance road and a mid-level sump.

Appendix E

Rock Mechanics

and

Formulae of Elasticity

Texture

Rock texture refers to the grain structure of the rock and can be classified by such properties as, porosity, looseness, density and grain size. All these factors have a bearing on rock strength.

Structure

Structural properties such as faults, joints, bedding planes, schistosity and rock type contacts, together with dip and strike all influence the structural strength of rock.

Hardness

Hardness can be used as a measure of the engineering properties of a rock behaviour to stress, by indicating how much stress is required to cause failure within the rock.

Rock contacts

Rock contacts form discontinuity planes, where the planes are associated with intrusive rocks they are often gradual and cause little reduction in rock strength. Distinct contacts, may be unstable and appear in association with joint systems.

Faults

Faults are generally shear planes along which, movement and mineralisation has taken place. Fault contacts often consist of fine grained material such as clay and mylonite, that have been formed in the shearing process.

Bedding planes and schistosity

Bedding planes and schistosity reduce rock strength both locally and on a large scale, if the bedding planes are tilted they can cause excavated faces to become unstable, with strata peeling from the face.

Jointing

Jointing has a similar effect to bedding planes, by having extended voids between rock masses, the shattering effect of an explosive shock wave can be disrupted, thereby reducing the compressive and tensile forces. This often results in poor blasting, sometimes accompanied excessive by fly rock.

Dip and strike

The dip and strike of a deposit usually indicate to what extent the deposit will be bedded and jointed and whether the cracks will be filled with calcite or another in fill material.

Rock mechanics

The study of deformation resulting from the strain of rocks in response to stresses is called rock mechanics. When a stress (force per unit area) is applied to a material such as rock, the material experiences a change in dimension, volume, or shape. This change, or deformation, is called strain.

Stresses can be axial, for example, a directional tension or simple compression, or shear (tangential), or all-sided (*for example*, hydrostatic compression). The terms stress and pressure are sometimes used interchangeably, but often stress refers to directional stress or shear stress and pressure (*P*) refers to hydrostatic compression. For small stresses, the strain is elastic (recoverable when the stress is removed and linearly proportional to the applied stress). For larger stresses and other conditions, the strain can be inelastic, or permanent. [ref 47]

Stress relationships

When calculating the forces needed to break rock, some blasting and other engineers refer to Young's modulus and Poissons ratio, below are examples of each, the writer has designed thousands of blasts without recourse to refer to these formulae. This is not to say that it is inappropriate to refer to these formulae, merely that a typical blaster can work quite well without ever having heard of Mr Young and his modulus.

The writer believes that, generally the use of these formulae can be considered inappropriate when blasting more than a single piece of rock, this is because that, on a large scale, rock bodies exhibit a multitude of physical and chemical variations and structural features.

Even on a single rock the assumption of ideal behaviour of elastic strain and homogeneous and isotropic stress and strain falls down because, in reality, on a microscopic scale there are grains and pores in sediments and a fabric of crystals in igneous and metamorphic rocks. Furthermore, conditions such as extended length of time, confining pressure, subsurface fluids, fractures, existing strain forces and reflective faces in the total rock mass affect the rates of change of deformation.

[ref 47]

Variation of some constants (in 10/\6 bars) with rock types and confining pressure								
	At a pressure of one bar				At a pressure of 3,000 bars			
Rock type	Bulk modulus	Young's modulus	Shear modulus	Poisson's ratio	Bulk modulus	Young's modulus	Shear modulus	Poisson's ratio
Granite	0.10	0.30	0.20	0.05	0.50	0.60	0.40	0.25
Gabbro	0.30	0.90	0.60	0.10	0.90	0.80	0.50	0.20
Dunite	1.10	1.50	0.50	0.30	1.20	1.70	0.70	0.27
Obsidian	0.40	0.70	0.30	0.08				
Basalt	0.50	0.80	0.30	0.23	0.80	1.20	0.40	0.25
Gneiss	0.10	0.20	0.10	0.05	0.50	0.70	0.30	
Marble	0.10	0.40	0.20	0.10	0.80	0.70	0.30	0.30
Quartzite					0.50	1.00	0.40	0.07
Sandstone	0.07	0.20	0.08	0.10				
Shale	0.04	0.10	0.05	0.04				
Limestone	0.80	0.60	0.20	0.30				

In elastic deformation, there are various constants that relate the magnitude of the strain response to the applied stress. These elastic constants include the following:

(1) Young's modulus (*E*) is the ratio of the applied stress to the fractional extension (or shortening) of the sample length parallel to the tension (or compression). The strain is the linear change in dimension divided by the original length.

(2) Shear modulus is the ratio of the applied stress to the distortion (rotation) of a plane originally perpendicular to the applied shear stress; it is also termed the modulus of rigidity.

(3) Bulk modulus (*k*) is the ratio of the confining pressure to the fractional reduction of volume in response to the applied hydrostatic pressure. The volume strain is the change in volume of the sample divided by the original volume. Bulk modulus is also termed the modulus of incompressibility.

(4) Poisson's ratio is the ratio of lateral strain (perpendicular to an applied stress) to the longitudinal strain (parallel to applied stress).

The following are the common units of stress:

1 bar = 10^6 Dynes per square centimetre
= 10^5 Newtons per square metre, or Pascal (Pa)
= 0.1 megapascal (0.1×10^6 Pa)

Thus 10 kilobars = 1 gigapascal (*i.e.*, 10^9 Pa). [ref 47]

Young's Modulus

This formula describes the elasticity of a solid and is of interest to blasting technicians when calculating the behaviour of a rocks ability when subjected to a high value shock wave, whereby, if the elasticity threshold of a rock is above the levels imposed by the shock wave, the rock will not break. Young's Modulus is a formula developed by Thomas Young, that describes the elastic properties of a solid undergoing tension or compression in one direction only, as in the case of a metal rod that after being stretched or compressed lengthwise returns to its original length. Young's modulus is a measure of the ability of a material to withstand changes in length when under lengthwise tension or compression. Sometimes referred to as the modulus of elasticity, Young's modulus is equal to the longitudinal stress divided by the strain. The modulus is meaningful only in the range in which the stress is proportional to the strain, and the material returns to its original dimensions when the external force is removed. As stresses increase, Young's Modulus may no longer remain constant but decrease, or the material may either flow, undergoing permanent deformation, or finally break [ref 47].

Typical values of elastic constants and properties			
	Elastic constants		Compressive strengths (kilobars)
Rock type	Young's modulus	Shear modulus	At room temperature and pressure
Shale	0.2 to 0.3	0.15	0.8 to 1.8
Sandstone			0.5 to 2.0
Limestone	0.4 to 0.7	0.22 to 0.26	1.0 to 2.0
Granite	0.3 to 0.6	0.20	1.7 to 2.5
Basalt	0.7 to 0.9	0.30	1.0 to 3.4

[ref 47]

Typical rock strengths with varying temperature & pressure				
Rock type	Temperature degrees Celsius	Confining pressure kilobars	Plastic yield strength kilobars	Ultimate strength kilobars
Granite	500.00	5.00	10.00	11.50
	800.00	5.00	5.00	6.00
Basalt	500.00	5.00	8.00	10.00
	800.00	5.00	2.00	2.50
Limestone	24.00	2.00	4.50	5.50
	500.00	3.00	2.50	3.00
Marble	24.00	2.00	2.50	5.50
	500.00	3.00	1.00	2.00
Shale	24.00	2.00	1.50	2.50

[ref 47]

Appendix F

Properties for Aggregates

PHYSICAL PROPERTIES and SPECIFICATIONS

There are many specifications for aggregates, the most commonly used are those produced by the American ASTM, and the British Standards Institution, each of these can be identified by their respective prefix numbers.

The ones of interest to this document are;

BS 12. Specifications for Portland cement.
BS 63. Specifications for grading, flakiness and strength for roadstones
BS 146. Specifications for Portland blast furnace cement.
BS 410. Specification for test sieves.
BS 812. Testing aggregates.
Part 1. Methods for determination of particle size and shape.
Part 2. Physical properties

- Part 100. General requirements for apperatus and calibration.
- Part 101. Guide to sampling and testing aggregates.
- Part 102. Methods for sampling
- Part 103. Testing aggregates
    - Part 103.1. Sieve tests
    - Part 103. 2. Sedimentation tests.
- Part 104. Recommended methods for determination of the nature and content of material finer than 75 micron.
- Part 105. Shape
    - Part 105.1. Flakiness index.
    - Part 105. 2. Elongation index
- Part 106. Methods for determining the shell content in coarse aggregate.
- Part 109. Moisture content.
- Part 110 Aggregate crushing value. (ACV)
- Part 111. Method for determination the 10% fines value. (TFV)
- Part 112. Method for determination of the aggregate impact value. (AIV)
- Part 113. Aggregate abrasion value (AAV)
- Part 114. Polished stone value. (PAV)
- Part 117. Water soluable chloride salts
- Part 118. Methods for determination of sulphate content.
- Part 119. Acid soluble material in fine aggregate.
- Part 120. Drying and shrinkage of aggregate.
- Part 121. Soundness of aggregate.

BS 882. Specifications Aggregates from natural sources for concrete.
BS 1199 and 1200. Specifications for Building sands from natural sources.
BS 1370. Specification for low heat Portland cement.
BS 1881. Testing concrete.

There are many others but these generally refer to specialised light aggregates. The general specifications are shown below;

1 Specification for Concrete Aggregate Materials as used in Large Civil Projects

100 kN. for 37.5 N/mm2
10% fines @ 50 kN.
ACV: not greater than 35
Los Angeles Abrasion (LAA): not greater than 50
Absorption: not greater than 2%
Sodium sulphate soundness: not greater than 12%
Magnesium sulphate soundness: not greater than 18%

2 Specifications for Bituminous mix and Surfacing Materials

Coarse Aggregate	Fine Aggregate
LAA 35 max - 40 max	LAA 50 max
ACV 28 max - 30 max	
PSV 50 min	

American Association State Highway and Transportation Officials Standard (AASHTO) T210 (durability) 25% min
Sodium Sulphate soundness 15 max - 12 max - 9 max
Crushing Index 100
Absorption not greater than 2.5%

3 Specifications for Road Base and Sub Base Aggregates

Base	Sub base
ACV not greater than 30%	
LAA not greater than 40%	LAA not greater than 45%

Physical properties

The essential physical properties of aggregates are;

- grading including silt, clay and dust
- strength, 10% fines, ACV, AIV
- polishing, degradation & soundness - PSV, LAA
- soundness
- shape and surface texture
- specific gravity and absorption

The assumption has been made that grading including silt, clay and dust content can be controlled by extraction and crushing techniques, similarly, the shape and surface texture under these conditions essentially vary with quarrying operations. The other physical property variables left to consider are now each discussed in relation to meeting the specifications.

Strength

This is not a stated requirement for material satisfying ASTM, C33 but it is for materials satisfying BS 882. The compressive strength of limestones varies from 10mN/m^2 to 200 mN/m^2. A dense limestone should have a strength of about 180 MN/m^2.

Los Angeles abrasion value

Concrete aggregates are required to have a maximum allowable abrasion value of 50% and good limestones should give about 25%.

Polished stone value

The polished stone value (PSV) is not a requirement of ASTM C33 and BS882 disregards polishing, a typical limestone will give 40 or more.

Aggregate abrasion value

For concrete aggregates an abrasion value of a maximum of 50% (ASTM) is permissible, good limestone should give, about 10.

By a similar review of specifications for the above materials defined under BS and ASTM standards the following conclusions can be made regarding specifications for aggregates for bituminous mixes, surfacing materials, road bases and sub bases.

Physical property requirements of for bituminous mixes and surface treatments(except grading and deleterious materials).

Coarse Aggregates, relevant codes ASTM D 692 and BS 594

Soundness	SSS	12% max
	MSS	18% max
Degradation	LAA	40% max for surface courses
	LAA	50% max for base courses

Fine Aggregates, relevant codes ASTM D 1073 BS 594

Soundness	SSS	15% max
	MSS	20% max
	P. I.	4% max

Bases and Sub-bases ASTM D 2940

Base LL 25	PI 4	sand equivalent 35 min
Sub base	PI 4	sand equivalent 30 min

Macadam Pavement ASTM D 693

Degradation	LAA	40 max surface
		50 max base
Soundness	SSS	20 max
	MSS	30 max
Fines	LL 30 PI 6	

Flakiness index

This expression describes a material of which the thickness is small relative to the other two dimensions. A high percentage of flaky material can adversely affect the durability of concrete as the material tends to orient in one plane, often with water and water voids forming between the layers. More than 15% flake by weight in coarse aggregate is considered to undesirable in concrete mixes and the BS 882 limit for flake is not more than 40% in crushed rock. Unless the bedding is very pronounced, flake is not commonly associated with limestone aggregates.

Elongation

Elongation is a product of both the nature of the rock and the type of crushers. Generally the more the rock is exposed to crushing, the less will be the elongation. Primary crushing only, will cause a high percentage of elongation. The BS 882 1983 limit for elongation is less than 15%.

Actual crushing value

Actual crushing value is an old style test of the strength of the rock, because the fines can become compacted in the test, weak aggregates tend to show a better result than is true. A figure of less than 12 is very good. For use in asphalt 28 is permissible and for concrete, 35 is permitted. A good limestone will give a value of 25 or less.

Hardness

Typically, limestones will fall between 2 and 4 on the Moh's scale of hardness.

10% Fines Value Test

This is a further and more accurate test of the strength of rock with an increase in the value of the number showing an improvement in the rock strength. The qualifying pressure to produce 10% fines for aggregates to be used in concrete with non wearing surfaces is 50 kN, this is a strange value as a rock giving 10% fines at 50 kN would hardly be classed as a rock. A good limestone should give a value of 180 kN or more.

Aggregate impact values

This test determines the toughness of the aggregate and consists of the impact of a standard hammer falling 15 times under its own weight upon a cylinder containing rock. The greater the value of the number, the tougher is the rock. The BS 882 1983 limits of aggregate impact values for use in concrete are up to 25% in normal concrete and 45% for use in concrete with wearing surfaces. Good limestones should give a value of 20% or less.

Absorption

The water absorption of aggregate is determined by measuring the increase in weight of an oven-dried sample when immersed in water for 24 hours (with the surface water removed). The ratio of the increase in weight to the weight of the dry sample, expressed as percentage is termed absorption. The BS 8002 recommends a maximum limit of absorption of 3%, and a result of less than 2% would be very good. High absorption figures can lead to problems with the water cement ratio, and therefore, the strength and workability of concrete. Absorption is often a product of the type of cementing medium in the rock, the degree of weathering and the amount of vesicles. Good limestone should have absorption values of less than 2%.

Soundness

This is the name given to the ability of aggregate to resist changes in volume as a result of changes to its physical conditions. The physical causes of large or permanent volume changes of aggregate are freezing and thawing, thermal changes at temperatures above freezing and alternating wetting and drying. A common characteristic of rocks with a poor record is their low specific gravity and their high absorption.

The tests involve immersing the sample in a solution of magnesium sulphate for seventeen hours and then oven drying. This cycle is carried out five times and the tested sample compared with the original. For use in concrete, the soundness index of a rock type should not be less than 88%. A good quality limestone should give a value of 99%. The British Highways gives soundness maxima of 65% for coarse and 75% for fine aggregates.

Apparent density

The apparent density will not define the quality of the rock. However, it is normally considered that the aggregate with the highest figure represents the best and most dense rock. A specific gravity of less than 2.2 indicates that the rock is not very satisfactory for use in concrete. Dense limestones dried at 110 degrees centigrade, should show figures of between 2,3 and 2.9 g/cm^3.

Alkali silica reaction

Certain types of rock have been found to have inclusions of minerals that cause a deleterious reaction between the silica, silicates, or carbonates in aggregates and the sodium and potassium hydroxides in the cement paste in concrete. Other component required are, oxygen and water. The minerals that are present in the rocks that may be deleteriously reactive include;

- fractured (granulated) quartz
- cristobalite
- Zeolite
- iron sulfides

The most common reaction being caused by the active silica constituents of the aggregates. In relation to bitumen-mineral reactions, the composition and texture of a material may suggest that its binding properties may not be optimal because of the amount and type of feldspars and ferro-magnesian minerals. Tests carried out for this are made to ASTM C586.

Appendix G

Cement Manufacture

CEMENT [ref 68]

Definition

The word cement is often used in a general sense to define adhesive substances of all kinds, but in a narrower sense it refers to the binding materials used in building and civil engineering construction. Cements of this kind are finely ground powders that, when mixed with water, set to a hard mass. Setting and hardening result from hydration, which is a chemical combination of the cement compounds with water that yields submicroscopic crystals or a gel-like material with a high surface area. Because of their hydrating properties, constructional cements, which will even set and harden under water, are often called hydraulic cements. The most important of these is Portland Cement

Quarrying

The proving, assessment and winning of cement raw materials is an integral part of the cement manufacturing process. The operation of the quarries must never be considered as a separate operation to the manufacture of the cement. Problems associated with the finished cement, low strengths or air setting for example, are just as likely to be associated with the raw materials as to the manufacturing process. The raw materials also dictate the design of the cement plant itself.

The manufacture of cement can be divided into a number of stages :

> <u>Quarrying</u> The chemistry of the stone being quarried, together with the variation in that chemistry, will determine every aspect of the quarrying procedure. Stones with little variation can be worked using a quarry layout and plant which is designed for cost effectiveness, often with large plant which is moved only short distances between blasts. Highly variable stone may require extraction in the form of a series of small widely spaced panels, using highly mobile plant and, possibly, computer control of the dump-truck routing to ensure that the required ratio of stone from each panel is maintained. In certain instances it is necessary to extract the stone from underwater using floating dredges.
>
> The physical nature of the stone can dictate whether or not the stone is carried in dump-trucks or can be transported out of the quarry using conveyors. The moisture level of a rock, or its ability to absorb moisture in wet conditions, can dictate the amount of stone which can be extracted from a face by blasting or ripping, and left on the ground prior to loading and removal from the quarry.
>
> One important aspect of quarrying is the collection of chemical data as the development of the quarry progresses. Sampling of blast-

hole dust, face sampling, or even drilling holes to collect samples as the faces progress, will all enable the detailed variations in the deposit to be determined more accurately. This is very important in ensuring that development of the reserves allows the stone leaving the quarry to have the correct chemistry on a day by day basis. This is an aspect of raw materials assessment which cannot be undertaken during the exploration and initial assessment stages due to the very close sampling required.

Crushing The choice of crusher is dependant upon the nature of the stone. A jaw crusher may be the most suitable equipment for a hard limestone, whereas a clayey material which would build up on the jaws of such equipment may require a hammer mill. Very wet raw materials could require an air swept crusher, possibly requiring an oil-fired furnace to provide the heat.

Stone storage and blending The purpose of crushed stone storage on a cement works is to provide a buffer stockpile to accommodate any problems in production of the stone in the quarries, or in the transport system to the works. If the stone has little chemical variability, this storage facility could be simple and consist merely of a stockpile from which the stone can be reclaimed. If there is high variability in the quarry, or a number of grades of stone are being used, the stockpile could take the form of a blending bed, the complexity of which would again be determined by the variation of the materials.

Milling Where raw materials consist of a mixture of rock types, the components of which vary in specific gravity and grain size, the nature of the mill lining will take into account the composition of the materials. Should the nature of the stone change, it is possible that the efficiency of the mill would be affected.

Raw meal homogenisation Just as with milling, the effectiveness of a raw meal homogenisation system will be affected by the size range of the particles leaving the milling system. This size distribution will be largely controlled by the nature of the raw materials used.

Kiln system Although vertical shaft kilns are used for the manufacture of cement, the vast majority of cement works use rotary kilns. Most modern works use a dry process which consists of a rotary kiln preceded by heat exchangers, normally in the form of ducting and cyclones, which pre-heat the kiln feed before it enters the kiln. The main clinker forming reactions take place in the kiln. Many modern kilns have a secondary burner, or pre-calciner, between the heat exchangers and the kiln. All these systems are highly fuel efficient and the waste gases, which leave the system by means of the stack, are at a low temperature. This has the disadvantage of trapping all the volatile elements, from the raw materials and the fuel, within the kiln system. Here they either enter the clinker or form compounds which build up within the components of the kiln system, causing it to operate inefficiently, or even resulting in complete failure of the system. These volatile compounds include alkalis, sulphur and chlorine.

Where raw materials contain such elements in excessive amounts, it is often necessary to reduce the circulating load of volatiles in the system by removing some of the circulating gases. Unfortunately, when this is done heat is also lost. An increased fuel consumption is inevitably the result of such a process.

The actual rate of clinker formation, the temperature required in the kiln, and therefore the type of refractory lining required, is determined by the nature of the raw materials. As indicated previously, the amount of energy, in the form of heat, required to combine pure silica with pure lime, is rather more than if the silica was already in the form of a silicate in a clay mineral or a mica.

Cement milling The ease with which cement clinker can be milled will be a function of its mineralogy. Although this will largely be determined by the burning and cooling conditions of the clinker, the raw materials will have had an influence on the final product.

The gypsum which is added to the clinker at the milling stage, should be considered a raw material. As with all the other raw materials, uniformity is essential. Variable quality gypsum could result in an unacceptable variation in insoluble residue in the cement, or could result in the presence of a variable amount of anhydrite in the final cement.

The cement "Flash setting", "false set" and "air setting" are all caused by compounds such as alkalis and sulphates in the cement. All of these will have come from the raw materials or the fuel.

The chemistry of cement raw materials

Cement consists mainly of minerals composed of calcium, silicon, aluminium and iron. When quoted in terms of oxides, the quantities of each are typically :

SiO_2	18.0% to 24.0%
Al_2O_3	4.0% to 8.0%
Fe_2O_3	1.5% to 4.5%
CaO	62.0% to 66.0%

The quantity of cement which is required throughout the world dictates that, in general, the only source which could supply the quantities of raw materials required is the earth itself. Rocks have, therefore, to be located which can supply calcium, silicon and the other elements in the required proportions.

Of the materials necessary for cement production, a source of calcium is the most important. Limestone, composed mainly of the mineral calcite, $CaCO_3$, is the obvious choice for this primary material. Other sources of lime have been used, ranging from the calcium carbonate-rich igneous rock carbonatite, to sea-shells dredged from the floor of the ocean.

The second main requirement is silica. This could, of course, be supplied using silica sand. However, as both alumina and ferric oxide are also required, clay or shale are almost universally used since they can supply all these requirements, often in approximately the correct ratio. Despite this a range of rocks and other materials, including by-products of other processes have been utilised from time to time.

It has been shown in the section on cement chemistry that, in order to create the right minerals in the correct combination, the proportions of silica, alumina, ferric oxide and lime must be absolutely correct. It is unlikely that deposits of limestone and clay, found in close proximity to each other, would achieve this chemical requirement. It is not unusual, therefore, to require a small supply of other materials to adjust the final chemistry of the feed to the cement kiln.

The level of silica in the mix can be adjusted with pure silica sand. Correction of the iron content is often made with iron-rich laterite or waste iron-slimes from other processes. Alumina can be adjusted with pure bauxite.

It is pertinent to note here that, if the cement plant is coal fired, the ash from the fuel will be incorporated in the cement clinker. This ash is effectively another raw material, and due account must be taken account of it when designing the chemistry of the raw materials fed to the cement kiln. Rocks are natural materials, often formed in a complex manner. They can, therefore, vary in composition, often over a very small distance. On the other hand, Portland cement is a material with a fixed chemical and mineralogical composition. Moreover, the plant constructed to produce the cement clinker will be designed for one set of materials of a given chemical composition and, once constructed, is relatively inflexible. If a mixture of limestone, shale and other materials, even with the correct chemical composition, is fed into a cement plant designed for a different type of limestone and shale, the cement plant will fail to function correctly, the required output will probably not be achieved, and the quality of the product may well not be acceptable.

A potential source of raw materials for a cement works cannot, therefore, be considered to be a raw materials reserve, until it has been shown that all the stone can be worked in such a manner that a continuously uniform material can be supplied for the life of the cement works. The chemistry of all materials which were used to design the cement manufacturing process, *must be available and remain constant*, within relatively tight limits, throughout the life of the cement works. A *detailed knowledge, not* only of the chemistry of all the materials used in a cement works, but also of the variation in those materials, is an essential pre-requisite for the successful construction and operation of a modern cement plant. The planned development of the quarries will be controlled by the chemistry of the rocks. The aim will be to supply, on a continuous basis, stone whose chemical variation is well within the blending and homogenisation capabilities of the cement works.

As was indicated previously, ash from the coal when this is used as a fuel, is a component of the raw materials. It is important that the chemical nature of this ash, together with the percentage of ash in the coal, is constant. It would not be unusual for a cement works to specify a specific mine as its source of fuel, but also a specific seam within that mine.

As well as requiring a uniform chemistry, the raw materials fed to the cement kiln should also have a uniform mineralogy. Chemical analyses are notional and a stone composed of pure coarse-grained quartz, bauxite and iron oxide, could have exactly the same chemical analysis as a clay containing minute grains of silica. In the environment within a cement kiln it would be considerably easier to combine the latter with limestone to form cement clinker, than it would with the former. Under given kiln conditions the silty clay could produce an excellent clinker, the sandy ferroginous bauxite an extremely poor quality one. For efficient operation of a cement kiln it is essential that the operating parameters remain constant. If the mineralogy of the kiln feed varies, this can result in a very variable quality clinker.

The chemistry of cement

Portland cement consists of a mixture of minerals, all of which are, to a greater or lesser extent, hydraulic. This means that when mixed with water, they form new minerals. In the case of cement these new minerals form an interlocking mass which gives rigidity to the mixture. The most important minerals in cement are calcium silicates. The chemical analysis of cement is expressed in terms of oxides, for which the cement chemist has developed his own shorthand :

lime	CaO	is referred to as	C
silica	SiO_2	is referred to as	S
alumina	Al_2O_3	is referred to as	A
ferric oxide	Fe_2O_3	is referred to as	F

The most important silicate in cement is tri-calcium silicate, $3CaO.SiO_2$ or C_3S in the cement chemist's notation. The crystalline material as it occurs in cement clinker normally contains impurities and is called alite. C_3S forms about 60% of Portland cement clinker and is responsible for much of the strength of concrete up to 28 days. As this early strength of concrete is often the basis for the comparison of cements, it is important to maximise the alite content through the correct choice of raw materials and process conditions.

The second compound is di-calcium silicate, $2CaO.SiO_2$, known as belite and abbreviated to C_2S. It makes up about 15% of cement clinker. This mineral hydrates more slowly than alite and is an important contributor to the development of the strength of concrete after 28 days.

Less important from the point of view of strength development are tri-calcium aluminate and tetra-calcium alumino-ferrite. The former mineral, abbreviated to C_3A, may contribute to the early strength of concrete. Although only present in small amounts, about 10%, it is much more reactive than the calcium silicates and has an important influence on the early hydration reactions which determine the workability and setting behaviour of concrete. The ferrite, C_4AF, contributes little to the strength of concrete, but is responsible for the characteristic grey colour of Portland cement.

The minerals C_3S, C_2S, C_3A and C_4AF are the main components of the cement clinker manufactured in the cement kiln. The cement powder used to manufacture concrete and mortar is made by grinding together this clinker with the mineral gypsum, $CaSO_4.2H_2O$. This mineral is added to control rapid setting of the cement paste caused by the reactive C_3A. This it does by forming a coating of the mineral ettringite on the surface of the grains of tri-calcium aluminate, which slows down the hydration of this mineral.

Because C_3S and C_2S are the main strength producing minerals in cement, the cement chemist will normally aim to produce a clinker with the maximum practical amount of these minerals, and with as much as possible being alite. This must, however, be undertaken at realistic combination temperatures in the cement kiln. Chemical and practical considerations have allowed a number of chemical control factors to be developed which enable these conditions to be achieved. These factors are silica ratio, alumina ratio and lime saturation factor.

Silica ratio (SR) this is defined as :

$$\frac{SiO_2}{Al_2O_3 + Fe_2O_3} \quad \text{or} \quad \frac{S}{A + F} \qquad \text{[ref 68]}$$

Since calcium in the kiln feed will react with silica to form silicates and with alumina and iron to form aluminates and ferrite, this factor provides an indication of the total possible amount of calcium silicates which can be formed in the process. The higher the silica ratio, the higher the quantity of silicates. However, the chemical reaction between silica and lime to form silicates will only take place at very high temperature unless fluxes are present. Since alumina and iron can provide this flux, some of these materials must be present if the reactions are to take place at realistic temperatures. The result of this required balance between silica and the other oxides, is that the optimum silica ratio is about 2.5.

Alumina ratio (AR) this is defined as :

$$\frac{Al_2O_3}{Fe_2O_3} \quad \text{or} \quad \frac{A}{F}$$ [ref 68]

The relationship between alumina and iron gives an indication of the characteristics of the molten flux which facilitates the formation of the silicates. The optimum value of the alumina ratio is about 1.4. Above or below this figure the formation of silicates becomes progressively more difficult.

Lime saturation factor (LSF) this is defined as :

[ref 68]

$$\frac{CaO}{2.8\,SiO_2 + 1.2\,Al_2O_3 + .65\,Fe_2O_3} \quad \text{or} \quad \frac{C}{2.8\,S + 1.2\,A + .65\,F}$$

The aim of the cement chemist is to maximise the content of calcium silicates in the clinker, in particular the content of tri-calcium silicate. This situation is achieved when the LSF has a value of 1.0. The weightings applied to silica, alumina and iron are derived from phase equilibrium studies. For convenience, the LSF is sometimes multiplied by 100 and expressed as a percentage.

Since, in practice, complete and perfect combination never takes place in the kiln, there are a number of other factors, based upon the LSF, which can be used in the production of cement. For example, the lime combination factor (LCF) reduces the quantity of lime used in the equation by an amount equivalent to the free lime present in the clinker plus 0.7 times the sulphur content.

In addition to the main constituents of cement clinker, lime, silica, alumina and ferric oxide, there are a number of other elements which can occur in cement raw materials and can have a significant effect on the cement or its production.

Magnesium oxide (MgO) This may react in a similar manner to lime and can, therefore, augment the silicate content. Uncombined magnesium oxide, or periclase, can cause unsoundness in concrete due to its high expansion when it eventually carbonates in the set cement. For this reason the content of MgO in cement is limited by national standards. In British Standard 12, for example, the limit is 4%.

Alkali oxides (Na_2O and K_2O) In clinker they prefer to be present as sulphates (Na_2SO_4 and K_2SO_4). Not only does this effect the cement by increasing early strength at the expense of later strength, it can also result in build-ups in the kiln system which can cause restrictions to air-flow and even blockages in the system. Where insufficient sulphur is present to form sulphates, alkali oxides can enter into the main components of the clinker. If the alkalis enter the C_3A, they can increase early reactivity and cause workability problems in the cement paste.

Alkalis in cement can also react with reactive silica or silicates in aggregate. This causes the production of a complex alkali silica gel which can expand and crack the concrete.

Mangannic oxide (Mn_2O_3) This compound acts like ferric oxide and can lower the effective alumina ratio if present in sufficient quantity.

Titanium dioxide (TiO2) Although apparently having no effect on the properties of cement, this compound, when present in sufficient quantity, can act like both silica and alumina and may, therefore, alter the effective AR and LSF.

Phosphorous pentoxide (P_2O_5) The effect of this material is complex. It can be both slightly beneficial or harmful depending upon the concentration. This is due to effect on the formation of the correct phase of the hydraulic minerals, some of the phases being less hydraulic than others.

Sulphur trioxide (SO_3) This has already been mentioned in connection with the alkali oxides. Due to the fact that sulphates can be harmful to hardened concrete, as well as being related to "flash setting" and workability problems, maximum levels are specified in national standards. Since C_3A is related to "flash setting", sulphur limits are related to the C_3A content. It should be remembered that the fuel used in the cement kiln can be a major source of the sulphur in the clinker.

Insoluble residue (IR) Although the material in the cement which is insoluble in dilute acid comes mainly from impurities in the added gypsum, it can also arise from uncombined silica in the clinker. It is limited by national standards to ensure the purity of the cement.

Loss on ignition (LOI) This is mainly water and carbon dioxide which are derived from the gypsum, the cement milling process and from the atmosphere during storage. It is limited by national standards in order to protect the quality of the cement.

The strength development of concrete is not only dependant upon the chemistry of the clinker and the formation of the correct proportions of the relevant minerals, but also on the size and distribution of those minerals. In order to set in a controlled and uniform manner, the different forms of silicate and aluminate should be in the form of individual grains in the cement. In order to carry out the milling of the clinker to form these grains in a cost effective manner, the crystals in the clinker must not be too small. If this was the case the cement would have to be ground to a very small grain size to liberate individual crystals of silicate. This in turn would probably enhance the reactivity of the cement. Control of the crystallography of the clinker is, therefore, an important aspect of cement manufacture. This is facilitated by the use of the correct raw materials, as well as by careful control of the burning and cooling processes.

Source and many thanks to Dr David Jefferson [ref 68]

Appendix H

Glossary

GLOSSARY

Absorption Spectrometry or mass spectrometry

This is the analytical method in which ions or ionic fragments of an analyte are separated based on mass-to-charge ratios. It is used to determine the metallic elements in solution. It operates by the analyte solution being aspirated into a flame which enables the atoms of the elements to absorb radiation from a cathode lamp which is unique to each element measured.

Alluvial

Alluvials, refers to unconsolidated detritus such as clay, silt, sand and gravel deposited by streams and rivers as sorted or semi-sorted sediment in channels and over flood plains and deltas. An alluvial deposit is a layer of broken rocky matter, or sediment, formed from material that has been carried in suspension by a river or stream and dropped as the velocity of the current changes. Alluvial deposits can consist of a whole range of particle sizes, from boulders down through cobbles, pebbles, gravel, sand, silt, and clay. The raw materials are the rocks and soils of upland areas that are loosened by erosion and washed away by mountain streams. River plains and deltas are made entirely of alluvial deposits, but smaller pockets can be found in the beds of upland torrents. River currents produce a sorting action, with particles of heavy material deposited first while lighter materials are washed downstream, this separation of products is often used by prospectors to locate heavy minerals.

Anticline

An anticline is an arch-like fold in stratified rocks in which the layers or beds bulge upwards to form an arch (seldom preserved intact) and where the two limbs (or walls of the arch) dip away from each other. The fold of an anticline may be undulating or steeply curved. A steplike bend in otherwise gently dipping or horizontal beds is a monocline.

Apatite

The word comes from the Greek apate, meaning to deceive, probably because the mineral can be found in almost any colour. It refers to any member of a series of phosphate minerals and is the world's major source of phosphorus. The chemical formula is $Ca_5(PO_4)_3,(F,Cl,OH)$ this mineral often occurs in igneous rocks and pegmatites, also in hydrothermal veins and as detrital deposits with copper sulphide. It has a hardness of 5 on Moh's scale of hardness As much of the material found is clear, apatite could be cut into an inexpensive gemstone, however, it is fragile and difficult to cut and polish.

Arenites

Arenaceous is Latin for sand like or sandy. The term is used to describe a sedimentary rock that consists of cemented sand or sand-sized particles (0.06-2 millimetres in diameter), irrespective of composition. More formal *nomenclature of such rocks* is based on composition, particle size, and mode of origin, *e.g.*, sandstone, arkose, *quartzite*.

Argillites

Argillaceous is French for clay like and Latin "lutaceous" for silt like. The term is used to describe silty clay deposits or shales of sedimentary rock. A compact rock composed largely of clay or clay minerals

Austenitic steel

Austenitic steels contain about 1.2 percent carbon and 12 percent manganese. The latter element is a strong austenizer; that is, it keeps steel austenitic at room temperature. Wear resistance is brought about by the high work-hardening capabilities of these steels; this in turn is generated during the pounding (*i.e.*, deforming) of the surface, when a large number of disturbances are created in the lattices of their crystals that effectively block the flow of dislocations. In other words, the more pounding the steel takes, the stronger it becomes. Manganese steels are often called Hadfield steels, after their inventor, Robert Hadfield.

Biotite

The micaceous silicate biotite (mica), has a composition of, $K(Mg,Fe)_3(AL,Fe)Si_3O_{10}$ $(OH\ Fe)_2$, and often occurs in granites, gneisses, schists and other rocks. It has a short tabular form a green to brown to black colour, specific gravity of 2.5 to 3.0. It can vary from opaque to translucent. This material is often found in southern Zambia.

Breccia

Limestone breccia (pronounced brekkia) is the name given to material formed by the crushing of rocks along a fault zone. The fragments of rock have become re-cemented by mineral matter deposited from percolating solutions after movement from a fault has ceased. The infill between the rock fragments is sometimes pure calcite. Sometimes the material has been broken and "healed" many times.

Calcite

Calcite (calcium carbonate, $CaCO_3$) is a colourless, white, or light-coloured common rock-forming mineral that rates 3 on the Mohs' scale of hardness. After quartz, calcite is the most common mineral found in the earth's crust it occurs as massive or compact rocks (limestones). Calcite originates in many ways, it occurs in magmatic, sedimentary and metamorphic rocks and is also formed secondarily by the decomposition of minerals rich in calcium. Secondary limestone (calcite) has often been deposited on sea beds due to the activity of living organisms.

Calcium

This is a useful element, the human body is two percent calcium. With a symbol of (Ca), the chemical element calcium is one of the alkaline-earth metals of main Group IIa of the periodic table. Calcium does not occur naturally in the free state, but compounds of the element are widely distributed, constituting 3.64 percent of the Earth's crust. Calcium is found in many other minerals, such as fluorite, aragonite, and gypsum, and in many feldspars and zeolites. Calcium is a silver-white chemical element that is a metal of the alkaline-earth group, lightweight metal (slightly heavier than water) itself was first isolated in 1808 by Sir Humphry Davy after distilling mercury from an amalgam formed by electrolyzing a mixture of lime and mercuric oxide. His discovery showed lime to be an oxide of calcium.

Formerly produced by electrolysis of anhydrous calcium chloride, pure calcium metal is now made commercially by heating lime with aluminum. Naturally occurring calcium consists of a mixture of six isotopes: calcium-40 (96.94 percent), calcium-44 (2.09 percent), calcium-42 (0.65 percent), and smaller proportions of calcium-48, calcium-43, and calcium-46. The metal reacts slowly with oxygen, water vapour, and nitrogen of the air to form a yellow coating of the oxide, hydroxide, and nitride. It burns in air or pure oxygen with a brick red flame, producing a white smoke. The smoke consists of fine particles of calcium oxide CaO). It reacts rapidly with warm water and more slowly with cold water to produce hydrogen.

Its principal compounds are;

- calcium oxide, also known as lime, or quicklime, CaO, is a white or grayish white solid produced in large quantities by roasting calcium carbonate so as to drive off carbon dioxide. Lime, one of the oldest products of chemical reaction known, is used extensively as a building material and as a fertilizer. Large quantities of lime are utilized in various industrial neutralization reactions. When mixed with water, calcium oxide swells and crumbles into a fine flocculent powder which is calcium hydroxide or slaked lime.

- calcium sulfate, $CaSO_4$, is a naturally occurring calcium salt. It is commonly known in its dihydrate form, $CaSO_4 2H_2O$, a white or colourless powder called gypsum. When gypsum is gently heated to below 100 degrees C and loses three-quarters of its water, it becomes the hemi-hydrate $Ca\ SO_4 1/2H_2O$, Plaster of Paris. If mixed with water, plaster of Paris can be moulded into shapes before it hardens to a coherent solid by re-crystallizing to dihydrate form. Calcium sulfate may occur in groundwater, causing hardness that cannot be removed by boiling.

- calcium phosphate occurs abundantly in nature in several forms. For example, the tribasic variety (precipitated calcium phosphate), $Ca_3(PO_4)_2$, is the principal inorganic constituent of bones and bone ash. The acid salt $Ca(H_2PO_4)_2$, produced by treating mineral phosphates with sulfuric acid, is employed as a plant food and stabilizer for plastics.

- calcium hypochlorite, $Ca(ClO_2)$ is widely used as bleaching powder and is produced by the action of chlorine on calcium hydroxide. The hydride CaH_2, formed by the direct action of the elements, liberates hydrogen when treated with water.

- calcium nitrate $(Ca(NO_3)_2.H_2O$- a nitrogenous fertilizer that is often used to make illegal explosives. Calcium nitrate was once common as an efflorescence on the walls of stables; it is now manufactured from atmospheric nitrogen. Its chief applications are as a manure and in the nitric acid industry.

- calcium chloride, $(CaCl_2)$ a colourless or white solid produced in large quantities either as a by-product of the manufacture of sodium carbonate by the Solvay process or by the action of hydrochloric acid on calcium carbonate. The anhydrous solid is used as a drying agent.

- calcium carbide (CaC_2) - Also known as carbide, or calcium acetylide, this grayish black solid decomposes in water, forming flammable ethyne or acetylene gas and calcium hydroxide.

- calcium hydroxide, also called slaked lime $Ca(OH)_2$, is obtained by the action of water on calcium oxide. When mixed with water, a small proportion of it dissolves, forming a solution known as limewater, the rest remaining as a suspension called milk of lime. Calcium hydroxide is used as a cheap alkali in many industrial processes and as a constituent of mortars, plasters, and cement it is also used to reduce soil acidity.

- calcium carbonate $(CaCO_3)$ - Calcium carbonate (calcite) obtained from its natural sources is used as a filler in a variety of products, such as ceramics and glass, and as a starting material for the production of calcium oxide. Synthetic calcium carbonate, called "precipitated" calcium carbonate, is employed when high purity is required, as in medicine (antacid and dietary calcium supplement), in food (baking powder), and for laboratory purposes. It occurs in limestone, chalk, marble, dolomite, eggshells, pearls, coral, stalactites, stalagmites, and the shells of many marine animals.

- calcium cyanamide ($CaCN_2$), made from calcium carbide, it is the basis of many pharmaceuticals, fertilizers, and plastics, including melamine.

- calcium cyanide $Ca(CN)_2$

The chemical symbol of calcium is Ca and its chemical details are;

atomic number 20, atomic weight 40.08 , melting point 842 C, boiling point 1,494 C specific gravity 1.55 (at 20 C), valence 2, electronic configuration or structure. 2-8-8-2 or $1s^2 2s^2 2p^6 3s^2 3p^6 4s^2$. Upon giving up its outer electrons the calcium atom changes into the dipositive ion Ca+++ and calcium is dipositive in all its compounds.

Carbonatites

These are igneous rocks that consist largely of the carbonate minerals calcite and dolomite; they sometimes also contain the rare-earth ore minerals bastnaesite, parisite, and monazite, the niobium ore mineral pyrochlore, and copper sulfide ore minerals. The origin of carbonatite magma is obscure. Most carbonatites occur close to intrusions of alkaline igneous rocks (those rich in potassium or sodium relative to their silica contents) or to the ultramafic igneous rocks (rocks with silica contents below approximately 50 percent by weight) known as kimberlites and lamproites.

Carboniferous

The Carboniferous Period of geologic time, fifth of six Paleozoic periods, spanned an interval of about 60 million years, succeeding the Devonian, which ended about 340 million years ago, and preceding the Permian, which began about 280 million years ago.

The period was first defined by stratigraphers W. D. Conybeare and William Phillips in 1822 in a report on the geology of England and Wales. The term Carboniferous was coined in the early years of the Industrial Revolution, when the increasing use of the carbon mineral coal prompted greater interest in coal mining and coal resources found in the rocks of this period (see coal and coal mining).

These rocks are also an important source of ceramic clay minerals, natural gas, ore deposits, petroleum and rock products. The lower division of the Carboniferous, generally consisting of marine sedimentary rocks such as limestone and shale, contains more oil and less coal, whereas the Upper Carboniferous has long been recognized as the Earth's main coal-bearing rock sequence. Industrialization first successfully proceeded in regions near ready supplies of high-rank Carboniferous coal.

Chalcopyrite

Chalcopyrite, is a brassy-yellow mineral consisting of copper-iron sulphide, $CuFeS_2$, the most common ore of copper. It usually has a bright iridescent surface tarnish. It occurs in many different types of mineral vein, in rocks ranging from basalt to limestone and is commonly found in the quarries described in the main text.

Chlorites

From the Latin chloritis meaning green stone. Chlorites are aluminium silicates with ferrous iron, magnesium and water, it is any of a group of usually green minerals associated with and resembling mica.

Conductivity

Conductivity is used by hydrologists and others to measure the purity of water. High purity, di-ionized water will have a conductance of 1.0 µS or less at room temperature. It is a direct measurement of the concentration of soluble salts in solution. Pollution will cause the conductivity to rise, common readings for drinking water are between 50 to 1,500 µS (microSiemens).

Density

Density varies significantly among different rock types because of differences in mineralogy and porosity. Knowledge of the distribution of underground rock densities can assist in interpreting subsurface geologic structure and rock type.

In strict usage, density is defined as the mass of a substance per unit volume; however, in common usage, it is taken to be the weight in air of a unit volume of a sample at a specific temperature. Weight is the force that gravitation exerts on a body (and thus varies with location), whereas mass (a measure of the matter in a body) is a fundamental property and is constant regardless of location. In routine density measurements of rocks, the sample weights are considered to be equivalent to their masses, because the discrepancy between weight and mass would result in less error on the computed density than the experimental errors introduced in the measurement of volume. Thus, density is often determined using weight rather than mass. Density should properly be reported in kilograms per cubic metre (kg/m^3), but is still often given in grams per cubic centimetre (g/cm^3). Another property closely related to density is specific gravity. It is defined, as noted above, as the ratio of the weight or mass in air of a unit volume of material at a stated temperature to the weight or mass in air of a unit volume of distilled water at the same temperature. Specific gravity is dimensionless (*i.e.*, has no units). The bulk density of a rock is where the weight of grains (sedimentary rocks) or crystals (igneous and metamorphic rocks) and natural cements, if any, and is the total volume of the grains or crystals plus the void (pore) space. The density can be dry if the pore space is empty, or it can be saturated if the pores are filled with fluid (*e.g.*, water), which is more typical of the subsurface (in situ) situation. Density measurements for a given specimen involve the determination of any two of the following quantities: pore volume, bulk volume, or grain volume, along with the weight.

Diagenesis

The processes, by which changes in a sediment are brought about after its deposition but before its final conversion to rock. Because most sediments contain mineral mixtures in which not all the minerals are in chemical equilibrium with each other, changes in interstitial water composition or changes in temperature or both will usually lead to chemical alteration of one or more of the minerals present. Diagenesis is considered a relatively low-pressure, low-temperature alteration process.

Dip

Dip refers to; the inclination of the stratum to the horizontal. Scarp and dip, the two slopes formed when a sedimentary bed outcrops as a landscape feature. The scarp is the slope that cuts across the bedding plane; the dip is the opposite slope which follows the bedding plane. The scarp is usually steep, while the dip is a gentle slope.

Dolomite

The name originates from the French geologist Deodat de Dolomieu born in 1801. The composition of dolomite is the double carbonate $MgCO_3.CaCO_3$, it is a type of limestone rock where the calcite content is either partly or wholly replaced by the white mineral dolomite, which has a rhombohedral structure and consists of a calcium magnesium carbonate. Dolomite rock may be white, grey, brown, or reddish in colour, it is commonly crystalline. Dolomite is a very common mineral and rock. It is very similar to calcite. It occurs as a mineral in hydrothermal ore veins and originates by the deposition of shells of tiny sea animals in oceans, and particularly by the metasomatic replacement of non cemented limestones affected by the action of sea water. The region of the Alps known as the Dolomites is an example of dolomite formation.

Dyke

Dykes are usually minor intrusions of igneous rocks, that is, small bodies of igneous rock created by the intrusion of magma (molten rock) across layers of pre-existing rock. They are often wall-like masses, steep or vertical, with approximately parallel sides and commonly have a fine to medium-grained texture. A fine-grained chilled margin is often formed by the rapid cooling of the igneous body at its contact with the country rock. Many of these are found intruding into the limestone at Changalume.

Evaporation

Evaporation is influenced by air temperature, windspeed and relative humidity, In general, evaporation increases with raising temperatures and windspeeds and decreases with high humidity. The rate of evaporation, is proportional to the difference between the pressure of the water vapour in the free air and the vapour that is next to, and saturated by, the evaporating liquid. If the liquid and air have the same temperature, evaporation is proportional to the saturation deficit. It is also proportional to the conductivity of the medium between the evaporator and the free air.

Evaporites

Evaporites are layered crystalline sedimentary rocks that form from brines generated in areas where the amount of water lost by evaporation exceeds the total amount of water from rainfall and influx via rivers and streams. The mineralogy of evaporite rocks is complex, with almost 100 varieties possible, but less than a dozen species are volumetrically important. Minerals in evaporite rocks include carbonates (especially calcite, dolomite, magnesite, and aragonite), sulfates (anhydrite and gypsum), and chlorides (particularly halite, sylvite, and carnallite), as well as various borates, silicates, nitrates, and sulfocarbonates. Evaporite deposits occur in both marine and nonmarine sedimentary successions.

Though restricted in area, modern evaporites contribute to genetic models for explaining ancient evaporite deposits. Modern evaporites are limited to arid regions (those of high temperature and low rates of precipitation), for example, on the floors of semidry ephemeral playa lakes in the Great Basin of Nevada and California, across the coastal salt flats (sabkhas) of the Middle East, and in salt pans, estuaries, and lagoons around the Gulf of Suez.

Ancient evaporates occur widely in the Phanerozoic geologic record, particularly in those of Cambrian (from 570 to 505 million years ago), Permian (from 286 to 245 million years ago), and Triassic (from 245 to 208 million years ago) age, but are rare in sedimentary sequences of Precambrian age. They tend to be closely associated with shallow marine shelf carbonates and fine (typically rich in iron oxide) mudrocks. Because evaporite sedimentation requires a specific climate and basin setting, their presence in time and space clearly constrains inferences of paleoclimatology and paleogeography. Evaporite beds tend to concentrate and facilitate major thrust fault horizons, so their presence is of particular interest to structural geologists. Evaporites also have economic significance as a source of salts and fertilizer.

All evaporite deposits result from the precipitation of brines generated by evaporation. Laboratory experiments can accurately trace the evolution of brines as various evaporite minerals crystallize. Normal seawater has a salinity of 3.5 percent (or 35,000 parts per million), with the most important dissolved constituents being sodium and chlorine. When seawater volume is reduced to one-fifth of the original, evaporite precipitation commences in an orderly fashion, with the more insoluble components (gypsum and anhydrite) forming first. When the solution reaches one-tenth the volume of the original, more soluble minerals like sylvite and halite form.

Natural evaporite sequences show vertical changes in mineralogy that crudely correspond to the orderly appearance of mineralogy as a function of solubility but are less systematic.
Source Encyclopaedia Britannica

Facies

The concept of Facies was developed by Eskola to assist in linking metamorphic rocks to their origins. He proposed eight facies which were originally based on fieldwork conducted in an area of metamorphosed igneous rocks, hence the following names;

Greenschist, amphibolite, albite-epidote, pyroxene hornfels, sanidinite, granulite, glaucophane, eclogite. At any given time different kinds of deposits may be formed in different environments. Examination of a particular stratum or sediment yields information about the materials composing it, their texture and other characters. These features are known as its facies. A metamorphic facies, is an assemblage of rocks formed under similar conditions of temperature and pressure. It is assumed that mineralogical variations within a facies are due to variations in bulk chemical variations of the parent.

Footwall

In a normal fault the term footwall is used to describe the lower underlying wall of a mineral vein, ore deposit, or coal seam in a mine, in a reverse fault, the footwall will be lower than the side lifted.

Gas Chromatography

Gas chromatography allows the separation, identification and determination of a number of volatile organic and other materials.

Geological Timescale

GEOLOGICAL TIMESCALE			
ERA	PERIOD	EPOCH	Millions of years
CENOZOIC	QUATERNARY	Holocene	0.01
Means age of recent life		Pleistocene	1.6
		Pliocene	5.3
	TERTIARY	Miocene	7.0
		Oligocene	57.8
		Eocene	66.4
		Paleocene	75.0
MESOZOIC		Cretaceous	144.0
Means age of middle life		Jurassic	208.0
		Triassic	245.0
PALEOZOIC		Permian	286.0
Means age of ancient life	Carboniferous	upper	325.0
	Carboniferous	lower	360.0
		Devonian	408.0
		Silurian	440.0
		Ordovician	505.0
		Cambrian	590.0
PRECAMBRIAN			4,000.000

Gneiss

Gneiss is a coarse grained metamorphic rock with constituents similar to granite but having a foliated, laminated or banded structure consisting of bands of micas and amphiboles with granular bands of quartz and feldspar laying parallel to each other. It is formed under conditions of increasing temperature and pressure, and often occurring in association with schists and granites. Gneisses are formed during regional metamorphism; paragneisses are derived from sedimentary rocks and orthogneisses from igneous rocks.

Graben

Is German for ditch and is the segment of land that has dropped down to form the floor of a rift valley.

Granite

Formed from slowly cooling, silica rich magma, granite is a generic term given to a family of igneous rocks. A hard coarse grained intrusive rock consisting of quartz, alkali feldspar, plagioclase and mica. Colours may range from red to pink, green to blue and black to grey, depending on the composition of the feldspar. Average density is 2.5 to 2.8.

Graphite

Graphite is a pure carbon (C) and is a very common mineral, most graphite originates by the metamorphism of carbonaceous material of sedimentary origin. Graphite is commonly found in the limestone of Chilanga RP3 quarry.

Greywacke

Is a dark coloured, poorly graded, argillaceous sandstone composed of sand-sized grains (0.063-2 mm) with a fine-grained clay matrix. The sand-sized grains are frequently composed of rock fragments of wide-ranging mineralogies (*e.g.*, those consisting of pyroxenes, amphiboles, feldspars, and quartz). The grains are angular and poorly sorted with many minerals retaining growth forms that resulted from low abrasion. The matrix, which contains appreciable amounts of clay minerals, may constitute up to 50 percent of the volume. Of the clay minerals, chlorite and biotite are more abundant than muscovite and illite; kaolinite is absent. The abundant matrix tends to bind the grains strongly and form a relatively hard rock

Gritstone

Gritstone is a rock composed of quartz grains cemented together by silica.

Gypsum

Gypsum is the most common sulfate mineral. A hydrated calcium sulfate, it serves as a raw material in plaster of Paris and is also used as fertilizer. Alabaster, a massive, fine-grained form, and satin spar, a fibrous form with a silky luster, are employed as ornamental stone. A transparent, variety called selenite is used as optical material. Gypsum occurs with halite and other evaporite minerals in thick, extensive beds that were deposited from seas or lakes. The mineral is soft (hardness 2), clear, white, or tinted. It has a relative density of 2.3 and perfect cleavage in one direction.

Hanging wall

This term is used to describe the side of a normal fault which has dropped relative to the footwall. In a reverse fault the term is used to describe the side that has risen relative to the footwall.

Haematite

Ferric oxide is a red or black chemical compound, Fe_2O_3 , composed of iron and oxygen that is found in nature as haematite. It is the most important ore of iron. It occurs as an accessory mineral in igneous rocks and hydrothermal veins, and in sedimentary rocks where it may be of primary origin. It often occurs as a secondary mineral, being precipitated from iron bearing water.

Horneblende

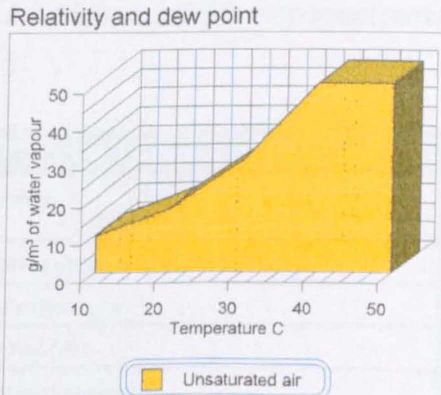
Hornblende $Ca_2(Mg,Fe)_4Al(Si_7Al)O_{22}(OH,F)_2$ is a member of the silicates group, it is found in igneous *rocks and the* metamorphic rock amphibolite. It has a hardness of 5 to 6 on Moh's scale of hardness, has a specific gravity of approximately 3.5, is found from green to brown to black.

Horste

From the German language and describes a block of the earth's crust on either side of a rift valley or between two parallel faults that have caused it to be separated from and raised above the adjacent land. The area each side of a Graben.

Humidity

Absolute humidity is the weight of water in grammes that is present in a cubic metre of air at a given temperature. The higher the temperature, the greater the amount of water which can be held in the air. However, for any temperature there is a maximum amount of absorption. Relative humidity is the ratio of absolute humidity to the saturation factor at the ambient temperature. The interface in the graph line also represents the dew point, whereby if a body of saturated air is cooled the water will condense, this is the dew point.



Karst Landscape

Karst (from German language) landscape is characterized by often remarkable surface and underground forms, created as a result of the action of water on permeable limestone. The feature takes its name from the Karst region on the Adriatic coast in Slovenia and Croatia and has given the name of *Karst topography* to landforms which are characteristic of chemically weathered limestone, but the name is applied to landscapes throughout the world.

Lamprophyre

Lamprophyre is a medium to coarse grained rock, homogeneous with a green, blue or black colour.

Lateritic soil

The term laterite is from the Latin "later" meaning brick. Lateritic soils develop by weathering and being leached in a tropical climate. They are often rich in oxides of aluminium and iron. Originally the term was used to describe a material which upon drying would irreversibly harden, the term is now used to describe many other types of soil that contain hard bands and nodules.

Limonite

Limonite is a generic term for mixed oxides or hydroxides of iron, these are usually weathered rock and of a yellow colour.

Lithology

Lithology refers to the character of rock expressed in terms of its mineral composition, structure, grain size and arrangement of its component parts.

Mercalli Scale

MERCALLI SCALE			
Scale	Intensity	Acceleration cm/sec/sec	Effects
I	None	Less than one cm	Measured on instruments only.
II	Very feeble	Over one cm	Can just be felt.
III	Feeble	Over 2.5 cm	Often felt.
IV	Moderate	Over 5.0 cm	Rattles windows.
V	Fairly strong	Over 10.0 cm	Can crack plaster.
VI	Strong	Over 25 cm	Slight damage to buildings.
VII	Very strong	Over 50 cm	Damage to weak buildings.
VII	Destructive	Over 100 cm	Damage to average buildings.
IX	Ruinous	Over 250 cm	Damage to strong buildings.
X	Disastrous	Over 500 cm	Buildings destroyed and ground cracked.
XI	Very disastrous	Over 750 cm	Few structures remain standing.
XII	Catastrophic	Over 980 cm	Total destruction and waves in ground surface.

The Mercalli scale is named after the Italian seismologist Giuseppe Mercalli (1850-1914), it is used to measure the subjective value of the intensity of an earthquake.

Metamorphism

The term was coined in 1833 by Scottish geologist Charles Lyell (1797-1875). The term is derived from the Greek "meta" signifying a change and "morph" meaning shape. Metamorphism is the term used to denote rocks altered in structure and composition by pressure, heat, or chemically active fluids after original formation and the transformation of rocks into new types by the re-crystallization of their constituents. The original rocks may have been sedimentary, igneous or ones that have already been metamorphosed. Changes in the rock occur from heat and pressure. These two influences need to express sufficient energy to mobilise the constituent minerals and reassemble them as new minerals. There are two main types of metamorphism.

- Thermal metamorphism, or contact metamorphism, is brought about by the baking of solid rocks in the vicinity of an igneous intrusion (molten rock, or magma, in a crack in the Earth's crust). It is responsible, for example, for the conversion of limestone to marble.

- Regional metamorphism results from the heat and intense pressures associated with the movements and collision of tectonic plates.

Metasediment

Metasediment refers to the changing of the structure of sediments by heat and pressure.

Monzonite

Monzonite is an intrusive igneous rock that contains abundant and approximately equal amounts of plagioclase and potash feldspar; it also contains subordinate amounts of biotite and hornblende, and sometimes minor quantities of ortho-pyroxene. Quartz, nepheline, and olivine, which are occasionally present, produce quartz, nepheline, and olivene monzonites. Monzonite is not a rare rock type, but it generally occurs in rather small, heterogeneous masses associated with (and perhaps gradational to) diorites, pyroxenites, or gabbros.

Muscovite

Muscovite is one of the commonest of the micas. It was named after the city of Moscow. The composition of this micaceous mineral is, $KA_{l3},Si_3,O_{10},(OH_2)$. It occurs as the main constituent of many rocks and pegmatites, arises by the metamorphosis of mica rocks into mica schists and gneisses, and is a secondary mineral resulting from the alteration of many silicates.

Nobel

Alfred Bernhard Nobel was born on . Oct. 21, 1833, in Stockholm, Sweden and lived until Dec. 10, 1896, in San Remo, Italy),a talented chemist and engineer who has been identified as the true "father" of high explosives, for without his work, the world would have had to wait many years before explosives became safe to use. Nobel invented dynamite in 1867, gelignite in 1875, and ballistite, a smokeless gunpowder, in 1887. In 1863 he developed a mercury fulminate detonator for use with his explosives. He founded the Nobel Prizes.

PH

pH is a negative logarithmic representation of the concentration of hydrogen ions in solution. For example, a solution of pH 5 is ten times more acidic than one at pH 6. A solution at pH 9 is ten times more alkaline than a solution at pH 8. The scale of measurement is from 1 to 14, with values of 1 to 7 being acidic and 7 to 14 being alkaline,7 being neutral.

Phearitic line

Taken to mean ground water or the ground water zone, the phearitic contour is the sub surface level of the ground water zone. The contact between the groundwater zone (phreatic zone) and the overlying unsaturated zone (vadose zone) is called the ground water table.

Phlogopite

The mica group silicate phlogopite, has a composition of, $KMg_3,AlSi_3,O_{10},(OH)_2$. This mineral occurs in contact metamorphic limestones and ultra-basic igneous rocks.

Plagioclase

Plagioclase is a group of aluminium silicates with calcium and sodium. It is a member of the feldspar family of rocks.

Pyroxene

The silicate minerals of the pyroxene group form many of the earths rock. They are found in basalts and metamorphic rocks. They form a large percentage of the earths mantle.

Quartz

Quartz consists of silica and oxygen, it is the crystalline form of silica (SiO_2). Silicon constitutes 27.7% of the earth's crust and is the second most abundant element after oxygen. Quartz is an important rock-forming mineral and is found in many forms and types of rock, including sandstone and granite, it ranks 7 on the Mohs' scale of hardness and is resistant to chemical or mechanical breakdown. Crystals of pure quartz are coarse, colourless, transparent, show no cleavage, and fracture unevenly; this form is usually called rock crystal. Impure coloured varieties, often used as gemstones, include agate, citrine quartz, and amethyst . Quartz is also used as a general name for the cryptocrystalline and noncrystalline varieties of silica, such as chalcedony, chert, and opal.

Quartzite

Quartzite is converted from the siliceous rock sandstone by metamorphism, by which the original quartz grains are re-crystallised as an interlocking mosaic of quartz crystals cemented together by silica.

Richter Scale

The Richter scale is named after US seismologist Charles Richter, it is based on the measurement of seismic waves and used to determine the magnitude of an earthquake at its epicentre. The scale is logarithmic, hence an rise by one unit corresponds to a ten-fold increase in the size of the wave and a thirty-fold increase in energy over the previous point. An earthquake's magnitude is a function of the total amount of energy released,

RICHTER SCALE	
Magnitude	Effects
2.5	Recorded but generally not felt.
4.5	Local damage
6.0	Destructive
7.0	Serious damage
8.0	Great destruction

Rutile

Rutile, TiO_2, is also known as titanium dioxide, it is found as an accessory mineral in igneous rocks and in gneiss, mica schist, granular limestone and dolomite.

Schist

Schists are crystalline metamorphic rock of medium texture. Different varieties of schists have been formed from sedimentary or igneous rocks by metamorphism. The name schist is derived from the Greek schistos meaning divided. This refers to the plate like quality of the rock.

Seismology

Seismology is the study of earthquakes and how their shock waves travel through the Earth. On a smaller scale artificially induced, generated by explosions or mechanical vibrators, can be used to search for subsurface features.

Sphene

A black or brown mineral, $CaTiSiO_5$, that is a silicate of calcium and titanium, it often contains varying amounts of other chemical elements (e.g. manganese and iron). This mineral often occurs as an accessory mineral in igneous rocks; in schists, gneisses and iron ore; and as a detrital mineral in sedimentary deposits.

Strike

Striking refers to; the direction of a horizontal line in the plane of an inclined stratum, joint or other structural plane

Stoichiometry

In chemistry stoichiometry is the determination of the proportions in which elements or compounds react with one another. The rules followed in the determination of stoichiometric relationships are based on the laws of conservation of mass and energy and the law of combining weights or volumes.

Strontianite

Strontianite is a carbonate mineral, strontium carbonate ($SrCO_3$), that is the original and principal source of strontium. It occurs in white masses of radiating fibres, although pale green, yellow, or gray colours are also known. Strontianite forms soft, brittle crystals that are commonly associated with barite, celestite, and calcite in low-temperature veins.

Syenite

Syenite is a coarse-grained igneous rock that is composed chiefly of the mineral feldspar. It is named after Syene an ancient city in Egypt.

Syncline

Geological term for a fold in the rocks of the Earth's crust in which the layers or beds dip inwards, thus forming a trough or inverted arch of stratified or layered rock in which the beds dip towards each other from either side. Vening Meinesz (1887 to 1966) proposed that synclines were the result of a downward buckling of the crust causing light sediments to fill the resulting depressions. This is the origin of the concept of the syncline.

Synform

A synform, is a fold in which the limbs (or walls of the arch) dip towards one another.

Tectonic

The expression tectonic originates from the Greek language tector meaning builder. It relates to earth plate movement. Tectonics embraces as its chief working principle the concept of plate tectonics, a theory that was formulated in the late 1960s by American, Canadian, and British geophysicists to broaden and synthesize the notion of continental drift and the sea floor spreading hypothesis.

Tillite

Tillite is a term often used to describe rock deposited from contact with a glacier. It is usually from the Pleistocene era. Analysis would typically show clay, sand, gravel and boulders.

Toes

The rock left unbroken at the foot of a quarry face after a blast.

Trapezoidal rule

In modern terms, the method of exhaustion applied to surfaces, for example, may be stated as follows: if S is a surface of unknown area s, then one may choose another surface of known area s' contained in S, and yet another surface S'' of known area s'' containing S; thus $s' < s < s''$. The approximating surfaces S' and S'' are polygons or sums of slices, mainly trapezoidal or rectangular, selected by Eudoxus and by Archimedes according to the particular figure S. In fact, sums of rectangular slices (the areas of which are particularly simple to compute) were to become predominant from the 16th century AD on. To complete the proof, Archimedes used the following proposition: if a known area a is such that for every positive there exist surfaces S' and S'' with $s' < a < s''$ and $s'' - s'$, then $s = a$. Archimedes did not isolate this proposition. He proved it, in concrete geometric terms for every particular figure that he considered, by a double reductio ad absurdum (double, because negative numbers were not available to him): If $s < a$, then $< a - s$ is taken; but, if $s > a$, then the choice $< s - a$ is made. In either case, a contradiction is reached, so that $s = a$. The method does not specify how the value of a is to be found. (Source Encyclopaedia Britannica)

Tremolite

A member of the silicate amphiboles group, tremolite, has a composition of, Ca_2,Mg_5, Si_8,$O_{22}(OH)_2$. This material occurs in contact and regionally metamorphosed dolomites or low grade basic igneous rocks.

Turbidity

Turbidity refers to cloudiness caused by very small particles of silt, clay, or organic such as algae or micro-organisms and other substances suspended in water. Even a slight degree of turbidity in drinking water is objectionable to most people. Turbidity is also an evaluation of the optical clarity of water and is measured by light scatter (nephelometry) using the properties of the suspended matter to scatter light directed into the water. High levels of turbidity will cloud the water and the particles will provide sites for occupation by bacteria. Turbidity also interferes with disinfection by creating a possible shield for pathogenic organisms. Groundwater normally has very low turbidity owing to the natural filtration that occurs as it percolates through the soil. Surface waters, though, are often high in turbidity.

Unconformities

In many places, one series of rocks is seen to lie upon an older series with a surface of separation between them. Junctions of this kind are called unconformities. The older strata were originally deposited in horizontal layers but often they are now seen to be tilted and covered by beds that lie across them. The upper beds are said to be unconformable on the lower. There is often a discordance in dip between the younger and older strata. Unconformity represents an interval of time when deposition ceased and some denudation took place.

Velocity

- Velocity
  The speed of movement in a given direction, given as distance traversed per unit time, usually in metres per second.

- Velocity of detonation
  The speed at which the chemical reaction of an explosive travels through the explosive from the point of detonation.

- Velocity (speed) of sound
  Sound is transmitted through media with velocities depending on the density and elasticity of the substance. Sound has a velocity of about 0.33 km per second (0.2 mile per second) in air, 1.5 km per second in water, and 5 km per second in steel. All sound waves travel with the same speed in air regardless of their frequency.

- Seismic wave velocity (general)
  Primary seismic waves give rock particles a back-and-forth motion along the path of propagation, thus stretching or compressing the rock as the wave passes any one point. The velocity of a wave is equal to the product of its wavelength and frequency (number of vibrations per second) and is independent of its intensity.

Water

Contaminants and Conditions	
Alkalinity	Measure of level of dissolved alkaline substances eg; bicarbonates
Aluminium	Aluminium sulphate used in water treatment, salts occur naturally and high levels can be toxic.
Ammonia	Occurs when nitrogen products (including ANFO) breakdown. Toxic to all aquatic life.
Boron	Found in calcium or sodium borate, can be present in industrial effluent. Toxic to crops.
Bromine	A powerful disinfectant used in water treatment plants.
Chlorine	Chlorine is used for the disinfection of drinking water in water treatment.
Chlorine	High levels of chlorine are used to sterilise water in distribution systems.
Copper	Occurs naturally in water and may cause discolouration and an astringent taste. Also used as an algicide by boat owners.
Cyanuric acid	Used as chlorine stabiliser.
Fluoride	Naturally occurring, but more often introduced to water to prevent tooth decay.
Hydrogen peroxide	Used at low levels in water treatment processes. High levels are used in the paper making and bleaching industries.
Iron	Widely found in natural water. Affects the taste of water and causes staining. Often found contaminating water from waste and tailings dumps.
Magnesium	Salts contribute to the hardness of water.
Manganese	Found naturally and causes staining similar to iron.
Molybdate	Used as a corrosion inhibitor in industrial water treatment.
Nitrate	An intermediate product in the nitrogen cycle, mostly found from the breakdown of fertilizers vegetation and oxidation of nitrogen products. Reduces oxygen and harmful to fish. The World Health Organisation gives an acceptable limit of 50 milligrams per litre.
Ozone	Is used in water treatment plants.
Phosphate	Used in fertilizers, detergents and foods, high levels are used to treat water in industry. Not directly harmful but associated with eutrophication of lakes and rivers.
Potassium	High levels can be an indication of brackish water.
Silica	Colloidal and soluble silicates are common.
Sulphate	Found naturally especially in hot springs. Present in many effluents. Toxic to fish.
Zinc	Used to inhibit corrosion in industrial water systems.

No water is naturally pure since it contains a variety of materials, either dissolved or in solution, as well as micro-organisms. The chart below details some of the pollutants.

Waves

Poisson, Cauchy, and George G. Stokes showed that the equations of the general theory of elasticity predicted the existence of two types of elastic deformation waves which could propagate through isotropic elastic solids. These are called body waves. In the faster type, called longitudinal, dilational, or irrotational waves, the particle motion is in the same direction as that of wave propagation; in the slower type, called transverse, shear, or rotational waves, it is perpendicular to the propagation direction. No analogue of the shear wave exists for propagation through a fluid medium, and that fact led seismologists in the early 1900s to understand that the Earth has a liquid core (at the centre of which there is due to gravity, a solid inner core).

Lord Rayleigh showed in 1885 that there is a wave type that could propagate along surfaces, such that the motion associated with the wave decayed exponentially with distance into the material from the surface. This type of surface wave, now called a Rayleigh wave, propagates typically at slightly more than 90 percent of the shear wave speed and involves an elliptical path of particle motion that lies in planes parallel to that defined by the normal to the surface and the propagation direction.

Another type of surface wave, with motion transverse to the propagation direction and parallel to the surface, was found by Love for solids in which a surface layer of material sits atop an elastically stiffer bulk solid; this defines the situation for the Earth's crust. The shaking in an earthquake is communicated first to distant places by body waves, but these spread out in three dimensions and to conserve the energy propagated by the wave field must diminish in their displacement amplitudes as r^{1}, where r is the distance from the source. The surface waves spread out in only two dimensions and must, for the same reason, diminish only as fast as $r^{1/2}$. Thus, the shaking effect of the surface waves from a crustal earthquake is normally felt more strongly, and is potentially more damaging, at moderate to large distances. Indeed, well before the theory of waves in solids was in hand, Thomas Young had suggested in his 1807 lectures on natural philosophy that the shaking of an earthquake "is probably propagated through the earth in the same manner as noise is conveyed through air." (It had been suggested by the American mathematician and astronomer John Winthrop, following his experience of the "Boston" earthquake of 1755, that the ground shaking was due to a disturbance propagated like sound through the air.) With the development of ultrasonic transducers operated on piezoelectric principles, the measurement of the reflection and scattering of elastic waves has developed into an effective engineering technique for the nondestructive evaluation of materials for detection of such potentially dangerous defects as cracks. Also, very strong impacts, whether from meteorite collision, weaponry, or blasting and the like in technological endeavours, induce waves in which material response can be well outside the range of linear elasticity, involving any or all of finite elastic strain, plastic or viscoplastic response, and phase transformation. These are called shock waves; they can propagate much beyond the speed of linear elastic waves and are accompanied by significant heating. (Source Encyclopaedia Britannica)

Weathering of limestone

Limestone is soluble in the weak acid of rainwater. Erosion takes place most swiftly along cracks and joints in the limestone and these open up into gullies called grikes. The rounded blocks left upstanding between them are called clints, often holes will be bored into the limestone from the action of water, these infill to become plugs. Chemical weathering is not restricted to easily soluble rocks but attacks all rock types. The most easily weathered rocks are limestones.