Chapter 8

CONCLUSIONS AND RECOMMENDATIONS

8.1 CONCLUSIONS

8.1.1 The Guided Approach

A new approach called the 'Guided Approach' has been developed for blast design at surface mines. It consists of ten elements, namely formulation of the objectives, study of the existing practice, site characterisations, selection of explosives, environmental considerations, determination of blast design parameters, initial design, field trials, assessment of blast performance, and optimisation. The silent features of the guided approach are:

- It incorporates most simple, readily available site characteristics that have relevance to blast design. They include rock type, compressive strength, density, penetration rate of drilling, block size, blasthole logging and hydrogeology.

- It critically reviews the existing blasting practice to identify the shortcomings and to explore the possibility of improving the blast results by introducing new techniques or new products.

- It incorporates the methods to assess and control the environmental effects of blasting.

- It recognises that field trials cannot be eliminated but the number of field trials can be reduced by the judicious selection of explosives and blast design parameters. A new design has to be properly justified before it is tried in the field.

- It emphasises the need for blast monitoring at different stages of blast design process.

- It is an integrated approach where empirical, engineering judgement and field trials are combined together to produce the desired results.

The existing practice of selecting explosives in Indian mines gives undue importance to the cost of explosives alone which is against the basic definition of optimum blasting. The concept of equivalent products does not consider the energy per metre of blasthole and the way in which the energy is partitioned between shock and heave energies. Very little efforts are made to evaluate the performance of explosives for a given condition. Tailoring explosives...
is limited to changing the density of the bulk explosives. Cartridged explosives are still widely used in large operations where bulk explosives should be the choice.

The first step in selecting an explosive is to decide between cartridged and bulk systems. If the annual requirement of explosives in a mine is more than 1000 tonnes, bulk explosives should be selected. The presence of water in blastholes may require water resistant explosives. The next step deals with matching the rock and explosive characteristics. The energy partition and rock-explosive interaction are directly considered as the performance of explosives is evaluated for real blasting conditions. Two methods are proposed for the evaluation of explosives. In the first method, the performance is judged by yield and power consumption for digging and is recommended when previous blast records are not available or new explosives are tested. In the second method, the performance is evaluated from the data base of previous records and is recommended when previous blast records are available. The overall cost determines the ultimate selection of an explosive.

From the analysis of the monitored data, an equation (Equation 5.3) is derived to estimate the ground vibration. The available equation (Equation 5.2) underestimates the vibration levels for greater scaled distances and overestimates the same for lower scaled distances. Hence, Equation 5.3 should be used. If the predicted values are greater than 5 mm/s, ground vibration should be monitored.

An equation is also derived for estimation of air overpressure. The derived equation (Equation 5.6) and the available one (Equation 5.5) are identical and give nearly equal values for the entire range of scaled distances. If the predicted values are greater than 120 dB, air overpressure should be monitored.

Air overpressure levels from the detonating cord (covered or uncovered) are significantly greater than those from the bench blasting. Therefore, while blasting close to built-up areas, trunks of detonating cord should not be used. If the detonating cord is the only available device, the air overpressure level can be reduced by 15-16 dB by covering it with soil. The measured air overpressure levels from secondary blasting are also higher than those from the bench blasting. If the boulders can be reduced and thus the frequency of secondary blasts, complaints due to blasting can be minimised.

In addition to conventional method of controlling ground vibration and air overpressure by restricting the maximum charge per delay, there is great potential to control them by optimising the design parameters at which the dissipation of explosive energy in creating these hazards is minimum.
Instead of specifying a fixed danger zone of 500 m, the flyrock should be controlled within one-half the distance between the blast and the nearest structure/village. Among all blast design parameters, the most important to control flyrock are the ratio of the stemming to burden and the ratio of the powder factor used to optimum powder factor. The occurrence of flyrock, all other conditions remaining the same, would be more when the blastholes are watery.

A simple procedure has been established for the selection of blasthole diameter incorporating the concept of compatibility between blasthole diameter and bench height, environmental constraints, block size in the rock mass, and the overall cost. Compared with the recommended hole diameters, there are only a few cases in Indian mines when the hole diameters are smaller but several cases when they are larger. A survey of international practices indicated that a smaller hole diameter than the recommended one is used when the environmental issues are very sensitive and/or very small size of fragments are required. A larger diameter than the recommended one is used when very high capacity loading equipment is deployed and/or the rock is either soft or highly fractured.

As the drilling equipment is usually decided at the mine planning stage, it is necessary to follow the recommended procedure at this stage itself so that the selected drilling equipment will not pose a constraint in the development of optimum blast design. The possibility of changing bench height may also be explored in case the hole diameter is fixed.

The calculation of burden is very crucial in the guided approach because spacing, stemming, and subgrade drilling are derived from the burden. The burden is calculated from the blasthole diameter, bench height, and rock mass properties. The burden to hole diameter ratio in Indian mines usually varies from 15 to 40 which is consistent with the published literature. This ratio was found to be less than 15 when the hole diameter was relatively large and greater than 40 when the hole diameter was small. Compared with the recommended burden to bench height ratio of 0.25 to 0.5, this ratio was found to be greater than 0.5 when the hole diameter was larger and less than 0.25 when the hole diameter was smaller.

An equation is also derived for the calculation of a new burden from the optimum one, when there is a change in the hole diameter or in the bench height or in both.

The calculated burden and spacing can be varied, provided that the area (burden x spacing) is kept constant and the spacing to burden (S/B) ratio is maintained between 1 and 2. S/B ratio was found to decrease with increasing hole diameter due to the greater risk of coarser
fragmentation with larger hole diameter. The observed S/B ratio deviates a little from the recommended limits. The use of S/B greater than 2.0 may be explained from the wide spacing technique but the ratios less than 1.0 are difficult to justify.

The recommended stemming length of 0.7 to 1.3 times the burden is consistent with the published literature. In Indian mines, some consideration is given to stemming length but no regard is given to the stemming material and the particle size of the stemming material. A significant portion of gaseous energy released by the explosive is lost due to stemming ejection. There is great potential in these mines for improving blast performance by using suitable stemming material or by using blast control plugs.

Empirical relations are given to calculate powder factor from rock type and the density of rock. The density of rock alone is not sufficient because rocks like granite with a density of 2.5 to 2.7 gm/cc would require twice the powder factor than that of limestone with the same density. A correction factor for block size is also incorporated. If the blasthole diameter is larger for the given bench height, an increased powder factor is unavoidable.

Practical problems related to delay sequence and delay timing while using a Sequential Blasting Machine were identified and solved. Based on this, principal diagrams were prepared and recommended for use under stringent site conditions. The suggested diagrams can be extended to other initiation systems also. In the line of international practices, the conventional use of detonating cord downline initiation should be replaced by better initiation systems such as the shock tube system. The advantages of better initiation system are well established abroad and can significantly improve the blast performance in Indian mines.

The Indian mining industry follows a cost analysis system which gives item-wise break-up such as wages, stores, power, government levies, general administration, miscellaneous. The existing costing system does not give operation-wise break-up and hence it considers the direct cost of explosives and accessories. It is essential to calculate operation-wise costs to appreciate the benefits of optimum fragmentation.

8.1.2 Applications of the Guided Approach

The guided approach was applied at three surface mines, namely Malanjkhand Copper Project, Rampura Agucha Project and Lambidhar Limestone Quarry.
8.1.2.1 Malanjkhand Copper Project

Malanjkhand Copper Project (MCP) was facing the problems of boulder formation and hard digging. The guided approach was applied for blast design in granite and quartz which are the principal rock types at MCP.

Cartridge explosives were used at MCP whereas bulk explosives could be ideal. On contrary to the apprehension of mine management about the suitability of blasthole diameter at MCP, the existing hole diameter of 165 mm for 12 m bench height was found to be suitable. The burden and spacing were optimised to 3.8 m and 4.7 m. The first row of holes was drilled at a distance of 2.0 to 2.5 m from the crest to minimise the problems of toe and flyrock. A laser profiler was used to determine the precise burden against hole depth and to mark the front row of holes at the proper distance. The staggered pattern was replaced with a rectangular pattern and was initiated with a diagonal or a V-cut. The stemming ejection was controlled by minimising the toe and using a shock tube initiation system. The video observation of the blasts with the shock tube system showed that the burden rock moved prior to the stemming ejection with controlled throw and negligible flyrock. It was conclusively demonstrated that closer drilling and excessive use of explosives do not always contribute to overall fragmentation but result in excessive flyrock, throw and backbreak.

The recommended design significantly improved the fragmentation, reduced secondary blasting, minimised the toe and flyrock. The powder factor could be reduced and the yield could be increased as a result of better utilisation of explosive energy in rock fragmentation. There was a direct saving on explosive and drilling cost with better fragmentation. The reduction in explosives consumption was about 16% and 25% in granite and quartz, respectively. The increase in yield was about 8.9% in granite and 29% in quartz. The savings per year were substantial in view of the magnitude of the operation at MCP. In addition, there were indirect savings in the downstream operations.

8.1.2.2 Rampura Agucha Project

The guided approach was also applied at Rampura Agucha Project (RAP) to assess the problem of ground vibration and to critically review the existing blast design parameters.

Based on the analysis of the previous records, it was found that the mine had taken 110 primary blasts with the frequency of two or three blasts in a week. The total charge varied mostly between 5 and 15 tonnes and the monthly consumption of explosives between 65 and 122 tonnes. The blasts were conducted with the maximum charge per delay up to 1000 kg.
The estimated peak particle velocity at the nearest structure of the village for this charge per delay was 2.59 mm/s and the estimated air overpressure was 117.59 dB. These levels were within the permissible limits. Actual monitoring of 10 primary blasts confirmed that the peak particle velocity at the village did not exceed 3 mm/s but the air overpressure levels were greater than 120 dB. The air overpressure was the possible cause for annoyance.

By restricting the maximum charge per delay and by optimising the blast design parameters, ground vibration was controlled within 5 mm/s and air overpressure within 129 dB. As a result of this study, there were no complaints from the villagers about the blast vibration and no undue restriction to the blasting operations.

From the analysis of previous blast records, it was found that the frequency of secondary blasts as well as the explosives consumed for secondary blasts was more in ore than in waste. It showed that fragmentation was a dominant problem in the ore which was confirmed by the observations of blasts both in waste and ore.

Considering the site conditions, bulk explosives were rightly selected. In waste rock, the blast design parameters were found to be appropriate except for the delay interval. The delay interval of 25 ms used by the mine was insufficient to provide adequate relief. Hence a delay interval of 50 ms was recommended.

In ore benches, the existing hole diameter of 165 mm was incompatible with the bench height of 5 m. In this combination, there was excessive flyrock due to the short stemming length and the powder factor was also high. It was not advisable to increase the stemming length because of boulder problem from the stemming zone. In the trial and error method, attempts were made to optimise the pattern but the problems due to incompatibility between blasthole diameter and bench height were not realised. After making the overall assessment of blast results, it was recommended either to increase the bench height from 5 to 10 m by merging two benches or to use smaller diameter holes. The possibility of changing the hole diameter was ruled out by the mine but trials could be arranged in 10 m ore bench. Because of the compatible condition, blast results improved significantly. The drilling requirement for 10 m bench was reduced by 30%. In addition, there were advantages due to improved fragmentation, better control of flyrock, and reduced consumption of explosives.

**8.1.2.3 Lambidhar Limestone Quarry**

Lambidhar Limestone Quarry (LLQ) was facing a problem of fines (particles ≤ 25 mm). Although each of the mining operations contributed to the generation of fines, blasting was the...
The control of fines was necessary as the price of lumps (> 25 mm) was two to four times that of the fines. The guided approach was applied at LLQ to reduce the percentage of fines.

The principal rock to be blasted at LLQ is grey marble, which is hard with small blocks. The micro-joints present in the rock mass further reduce these blocks into smaller pieces. Considering the site conditions, ANFO was selected and used in the cartridge form to provide a decoupling ratio of 1.2 to 1.25. For the planned bench height of 6-7 m, the existing hole diameter (100 mm) was found suitable. The burden, spacing, subgrade drilling and powder factor were calculated and optimised.

By optimising the powder factor to a value of 0.3 kg/m³, the amount of fines was brought down to 36.5%. Further reduction of fines through blast design was however limited by the presence of fines (about 20-30%) in the rock mass itself.

Prior to the investigation, the production of fines at the screening plant was about 60-62%. After implementation of the optimised design, it was brought down to 50-52%. Thus, about 16% reduction in the lines was achieved.

8.2 RECOMMENDATIONS

On the basis of this work, the following recommendations are made for further study:

- Various initiation sequences are used for bench blasting, but none of them considers the joint orientation which is reported to have a significant influence on rock fragmentation (Burkle, 1979, Belland, 1966, Singh and Sastry, 1987). In Section 7.2.7.2, it was noted that fragmentation was always better when the free face was oriented in the east-west direction than in the north-south. It appears that there is an optimum firing direction with respect to the joint direction in which the explosive energy is better utilised in breaking rock. Therefore, it is recommended to determine the optimum firing direction for bench blasting.

- It is difficult to vary a design parameter in the field over a wide range and study its influence, keeping all others constant. The blast reflects the combined effect of several factors. For this type of problem, multivariate analysis techniques (Morrison, 1990) can be applied. These techniques are extremely powerful in considering the changes in several factors simultaneously and to determine the relative order of influence for the causative factors.
• The data base of ground vibration along with blast details created during the study was found very useful to predict ground vibration and air overpressure, to derive relationships among the various design parameters, and to evaluate the explosive performance. It is therefore recommended that every mine should maintain detailed records of the drilling and blasting operations. These records may be used to develop expert systems for blast designs.

• In this study, empirical relationships with powder factor could be established only for a limited number of rock types and for rocks having densities up to 3 gm/cc. This work may be extended.

• For short stemming length, technical and economical benefits of the different stemming materials and blast control plugs may be investigated.

• Matching rock and explosive properties is a very challenging and important area for further research.

• Available methods for estimation of flyrock are inadequate and an equation for estimation of flyrock may incorporate the ratio of stemming length to burden and the ratio of powder factor used to optimum powder factor.

• There is a need for development of inexpensive and simple tools for blast monitoring on routine basis.