Rock Mechanics and Rock Engineering (2023) 56:747–778
https://doi.org/10.1007/s00603-022-03068-3

ORIGINAL PAPER

Reduction of Fragment Size from Mining to Mineral Processing: A Review

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Received: 30 March 2022 / Accepted: 7 September 2022 / Published online: 4 October 2022

Abstract

The worldwide mining industry consumes a vast amount of energy in reduction of fragment size from mining to mineral processing with an extremely low-energy efficiency, particularly in ore crushing and grinding. Regarding such a situation, this article describes the effects of rock fragmentation by blasting on the energy consumption, productivity, minerals' recovery, operational costs in the whole size reduction chain from mining to mineral processing, and the sustainability of mining industry. The main factors that influence rock fragmentation are analysed such as explosive, initiator, rock, and energy distribution including blast design, and the models for predicting rock fragmentation are briefly introduced. In addition, two important issues—fines and ore blending—are shortly presented. Furthermore, the feasibility of achieving an optimum fragmentation (satisfied by a minimum cost from drilling-blasting to crushing-grinding, maximum ore recovery ratio, high productivity, and minimum negative impact on safety and environment) is analysed. The analysis indicates that this feasibility is high. Finally, the measures and challenges for achieving optimum fragmentation are discussed.

Highlights

- The effects of rock fragmentation on the whole size reduction chain from mining to mineral processing are described.
- The main factors influencing rock fragmentation by blasting are analysed.
- Main models for predicting rock fragmentation are briefly introduced and commented on.
- The feasibility, measures, and challenges of achieving optimum fragmentation are analysed.

Keywords Energy efficiency · Rock fragmentation · Blasting · Mine to mill · Mineral recovery · Crushing and grinding

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List of symbols

q	Specific charge or powder factor, kg/m^3 or kg/t
x_{50}	Median fragment size, mm
n	Uniformity index
B	Burden, m
S	Spacing, m
H	Bench height, m
d	Diameter of borehole, mm
L_h	Borehole length, m
L_c	Charge length, m
c_p	P-wave velocity of rock, m/s
D or VOD	Velocity of detonation, m/s
P	Percentage passing
x_P	Particle size at the percentage passing P, mm
η_{db}	Energy efficiency in drilling and blasting
η_{cg}	Energy efficiency in crushing and grinding

1 Introduction

The modern economy depends heavily on minerals production. To supply sufficient minerals and meet the demands of the modern economy, the worldwide mining industry consumes a vast amount of energy every year. For example, in USA, the mining industry consumes approximately 1.3×10^{18} J of energy per year (BCS 2007). Unfortunately, the energy efficiency is extremely low in operations, such as rock blasting, crushing, and grinding. For instance, the energy efficiency is about 10% in percussive rock drilling (Carrol 1985), 3–5% in rock crushing (Prasher 1987), approximately 1% in milling (Chi et al. 1996; Alvarado et al. 1998; Fuerstenau and Abouzeid 2002; Zhang and Ouchterlony 2022), and maximum 6% in rock blasting (Ouchterlony et al. 2003; Sanchidrián et al. 2007). These extremely low-energy efficiencies result in a considerable amount of energy wastage and make the mining industry worse than most other industrial sectors regarding energy utilization.

In hard rock mining, the ore mass is first broken down into various sized fragments by blasting, which is often called (blast) fragmentation. In the downstream operations such as crushing and grinding, ore fragments from blasting are further crushed and ground into smaller particles. In general, crushing and grinding are together called comminution in mineral processing and grinding is typically the highest energy consuming stage of the mineral processing. Consequently, to improve process economics, there is high demand for technologies capable of reducing particle size more cost-efficiently. Before the mine-to-mill concept was initiated (probably, much initial site-related mine-to-mill research was done in the Julius Kruttschnitt Mineral Research Centre (JKMRC) from the 1980s, see e.g. McKee et al. 1995; Ouchterlony 2003a; McKee 2013), mining operation and mineral processing had been separated into two independent units in management, especially regarding accounting. Thus, the mining unit focused on its internal costs, including drilling, blasting, loading, transportation, and hoisting, without considering whether drilling and blasting do or do not affect the downstream operations of crushing and grinding. From a mineral processing viewpoint, blasting plays an important role in improving the energy efficiency of comminution process (McKee et al. 1995; Kojovic et al. 1995; Michaux and Djordjevic 2005; McKee 2013; Napier-Munn 2015).

When hard ores are mined, crushed, ground, and processed (e.g., concentrated), a huge amount of energy must be spent and a considerable amount of CO_2 produced. According to Life Cycle Assessment (LCA), a relatively new methodology for assessing the environmental impact of various activities, the greenhouse gas emissions were 12 and 5 kg CO_2e (CO_2e means carbon dioxide equivalent) per ton iron ore and per ton bauxite, respectively, while they were 628 kg CO_2e per ton copper concentrate corresponding to 39 kg CO_2e per ton copper ore (Norgate and Haque 2010). In the case of copper ore, it is the crushing and grinding (particularly the latter) steps that make the largest contribution (approximately 47%) to the total greenhouse gas emissions for the production of copper concentrate. On the other hand, explosives made only a small (1–8%) contribution to the overall greenhouse gas emissions, amounting to 0.7 and 0.4 kg CO_2e/t for iron ore and bauxite respectively, and 9.1 kg CO_2 e/t concentrate (or 0.6 kg CO_2 e/t ore) for copper concentrate (Norgate and Haque 2010).

In the 1970s, it was recognized that rock fragmentation by blasting influenced other operations, such as loading, hauling, and crushing (Zeggeren and Chung 1975; McKee 2013). Then the concept of optimum fragmentation became an important research topic in mining engineering (e.g., Chiappetta and Borg 1983; Xu and Yu 1984; Nielsen 1984). Nielsen (1984) carried out one of the earliest experimental studies linking mining to processing, considering that blasting could precondition fragmented rock, so that the energy required in subsequent crushing and grinding operations would be reduced. Jaeger et al. (1986) found, by means of scanning electronic microscope (SEM), that blast-produced rock fragments contained a multitude of cracks, indicating that such cracks could be beneficial to crushing and grinding. Meanwhile, Chertkov (1986) mathematically modeled the correlation of preliminary and explosive-induced cracking with the comminution characteristics of brittle rock and concluded that the blasting process introduced cracks into the rock fragments. Two years later, Revnivtsev (1988) reported an experimental result that the energy required for crushing and grinding of an explosive-produced rock lump was lower than that for the rock prior to blasting, and then, McKee et al. (1995) demonstrated that fragmentation in blasting and comminution in mineral processing were correlated. Many small-scale laboratory tests indicated that rock blasting had a significant impact on crushing and grinding (Eloranta 1995; Chi et al. 1996; Tunstall and Bearman 1997; Nielsen and Lownds 1997; Mansouri et al. 2018), and influenced the Bond work index (Nielsen and Kristiansen 1996), the strength (Kemeny et al. 2003), the damage, and the P-wave velocity (Roblee and Stokoe 1989; Katsabanis et al. 2003) of rock fragments. Since then, a large number of the so-called mine-to-mill studies on optimum fragmentation have been initiated in Australia (McKee 2013) and then adopted in other countries (Ouchterlony 2003a). Such studies have been widely performed over the world for several reasons. One reason is the fact mentioned above that the worldwide comminution of rocks consumes a vast amount of energy and other resources. For example, comminution consumes 53% of the total energy in the whole process from mining to mineral processing, and the comminution cost is as high as 67% of the total cost in the process, whereas the cost of drilling and blasting is only 5% (Spathis 2015). Another reason is that the energy efficiency of comminution is extremely low, as mentioned earlier.

With an increasing population and GDP per capita, the global production and consumption of various minerals have increased for over one century, and at the same time, a vast amount of minerals has been lost in mining and mineral processing since 1920 (Zhang et al. 2021c). It is worth noting that the ore recovery ratio in mining can be increased by advanced blasting technology (Brunton et al. 2010; Zhang 2005a,b, 2014, 2016a,b; Zhang and Wimmer 2018), meaning that the sustainability of raw material production can be improved by optimizing fragmentation in blasting. To realize optimum fragmentation, previous studies focused on improvement of rock blasting, mainly due to the great discrepancy between energy efficiency in rock blasting and that in grinding or milling described above. By making use of this discrepancy, Zhang (2008, 2016a) mathematically demonstrated that energy efficiency in the chain from rock drilling and blasting to crushing and grinding could be increased when energy input in blasting was increased by a certain amount. In other words, theoretically, savings in the chain could be achieved by increasing the energy input in rock blasting. In practice, many mines gained more savings or higher mill throughput by employing a higher specific charge (or powder factor) in mining production blasting (Kojovic et al. 1995; Strelec et al. 2000; Karageorgos et al. 2001; Lam et al. 2001; Paley and Kojovic 2001; Kojovic 2005; Michaux and Djordjevic 2005; Adel et al. 2006; Bye 2006; Brent et al. 2013; McKee 2013; Ouchterlony et al. 2013). For example, many projects called mine to mill increased productivity in the range of 10–20% by means of a higher specific charge (McKee 2013). However, a higher specific charge does not necessarily yield better fragmentation, more savings, and higher mill throughput, if misfires occur or poor blast designs are used (Zhang 2016a).

Like high specific charge, the delay time between two adjacent blastholes was considered to be another possible key to optimum rock fragmentation by realizing efficient superposition of stress waves from the two neighbouring holes. In this spirit, Rossmanith (2002) and Rossmanith and Kouzniak (2004) described how a positive effect of stress wave interaction could be achieved between two blasting holes with a short inter-hole delay time. Their theory was tested by Vanbrabant and Espinosa (2006) in full-scale blasts. They found that the average fragmentation was improved by nearly 50%. However, many other small-scale and full-scale blasts did not produce better or much better fragmentation when very short delay times were used (e.g., Stagg and Nutting 1987; Katsabanis et al. 2006; Johansson and Ouchterlony 2013; Petropoulos et al. 2014). Regarding these two contradictory results, Blair (2009) argued that the probability of the positive stress wave interactions mentioned by Rossmanith (2002) was very limited. Zhang (2016a) further explained that considering only stress wave interaction without crack propagation and fragment movement was not sufficient to determine a best delay time for optimum fragmentation.

Grinding is the most expensive process in the mineral processing stage (Aldrich 2013; Napier-Munn 2015; Díaz et al. 2018). In addition to the direct energy consumption, grinding indirectly consumes energy through media and wear materials. One challenge in recent years has been to treat larger volumes of low grade and geographically disseminated ores, while energy and operating costs constantly increase. Disseminated complex ores of low grade require sufficient size reduction for liberating the valuable minerals, but at the same time, it is vital to prevent the formation of excessive quantities of fines, whose creation requires much energy. Consequently, there is a high demand for solutions for reducing the energy consumption and formation of fine particles in comminution. The technologies capable of producing primarily micro-cracks and selective fragmentation along grain boundaries are of interest, since they might create less fine particles.

After ore particles are ground in mills, they will be further processed by separation techniques, such as flotation. In this process, extra-fine mineral particles cannot be recovered by current processing technology (Wills and Napier-Munn 2006). Since such small particles are produced not only in crushing and milling but also in blasting, how to control or reduce them and what the smallest particles are that can be accepted by modern processing technology are two relevant questions.

To achieve optimum fragmentation, it is necessary to be able to predict rock fragmentation results. With this purpose, several models or functions, such as the Kuz–Ram fragmentation model (Cunningham 1983; 1987; 2005), the Swebrec function (Ouchterlony 2005a), the JK breakage appearance model (for DWT or drop weight testing, e.g., Napier-Munn et al. 1996; Shi 2016), and the fragmentation-energy fan (Ouchterlony et al. 2017; Sanchidrián and Ouchterlony 2017; Ouchterlony and Sanchidrián 2018; Segarra et al. 2018; Ouchterlony and Sanchidrian 2019) were developed. These models, most of which describe the outcome of bench blasting, can describe the relation between particle/fragment size and accumulated mass passing quite well and the Swebrec function was found to be the best-fitting function in general, in nearly all groups of data and across the whole passing range (Sanchidrián et al. 2012, 2014), compared with other functions. It is desirable that in the future, such models can be further developed to more accurately link the input parameters of a blast such as burden, spacing, rock mass properties, etc. with the output, the fragment size distribution.

The above description indicates that optimization of rock breakage from mining to mineral processing follows a clear trend leading to gains in mining productivity and savings. However, the successful applications of optimum fragmentation or mine-to-mill projects delivered different productivity gains or savings, e.g., as reported by McKee (2013), the mine-to-mill projects were not all successful in achieving either higher productivity or larger savings when a higher specific charge was used. The reason may be either the different blast techniques such as different specific charges used or incorrect blast designs giving rise to misfires, but this needs to be confirmed. Previous studies or applications have in general not explained sufficiently well what the optimum specific charge is and how to determine it. In addition, the contradictory results from short delay time tests have not been explained satisfactorily yet, resulting in that determination of optimum delay times in rock blasting lacks a reliable scientific description. Finally, it is necessary to consider the effect of fine particles on mineral processing. Based on the above background, this article describes the following topics: (1) the operations rock fragmentation may affect, (2) the factors influencing fragmentation, (3) fines, ore blending, and ore sorting, (4) prediction of rock fragmentation, (5) optimum fragmentation, and (6) measures and challenges of achieving optimization of rock fragmentation.

2 Effects of Fragmentation on Mining, Mineral Processing, Sustainability, and Environment

2.1 Energy

Rock drilling, blasting, and comminution (crushing and grinding) consume a vast amount of energy in hard rock mines. As mentioned in Sect. 1, statistics from hard rock mines indicates that drilling and blasting consumes 2% and comminution does 53% of the total energy input in the whole production chain from mining to mineral processing (Spathis 2015), see Fig. 1.

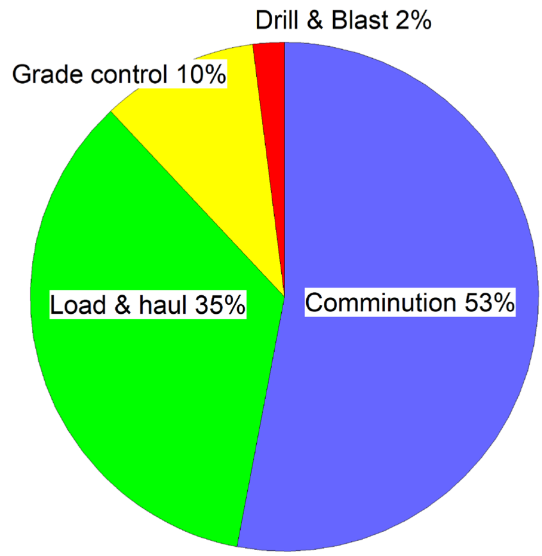


Fig. 1 Apportionment of energy consumption in hard rock mines (based on the data from Spathis 2015)

A notable fact is that the energy efficiency of conventional milling is only about 1%, as mentioned in Sect. 1. An earlier experimental study indicated that about 80–90% of the energy input of a ball mill was used in heating the material (Schellinger 1951, 1952), and a recent study showed that over 75% of the electrical energy was used to heat the slurry (Bouchard et al. 2016). Based on these studies, it can be concluded that most of the energy input in milling is consumed in heating the materials.

Better (or finer) fragmentation could normally save a vast amount of energy in the mining industry. For example, as shown in Fig. 2, the two pictures that were taken after blasting but before ore extraction started show the muckpiles of two similar blasts in a production drift in a sublevel caving mine (Zhang 2016a). Except for different detonator positions in the two blasts, the ore mass, the explosive, the sizes of blastholes, and the explosive charges were almost the same or at least very similar. That is to say, the different fragmentation results are with a large probability mainly due to the different detonator positions. Evidently, the muckpile shown in Fig. 2b will save much energy in the downstream crushing and grinding. In other words, a better blast design can most probably save energy in crushing and grinding. A similar result was achieved in a recent study, showing that the sizes of large fragments were markedly reduced as the detonator positions were changed to the middle of the explosive charge length from the position close to the bottom of blasthole in

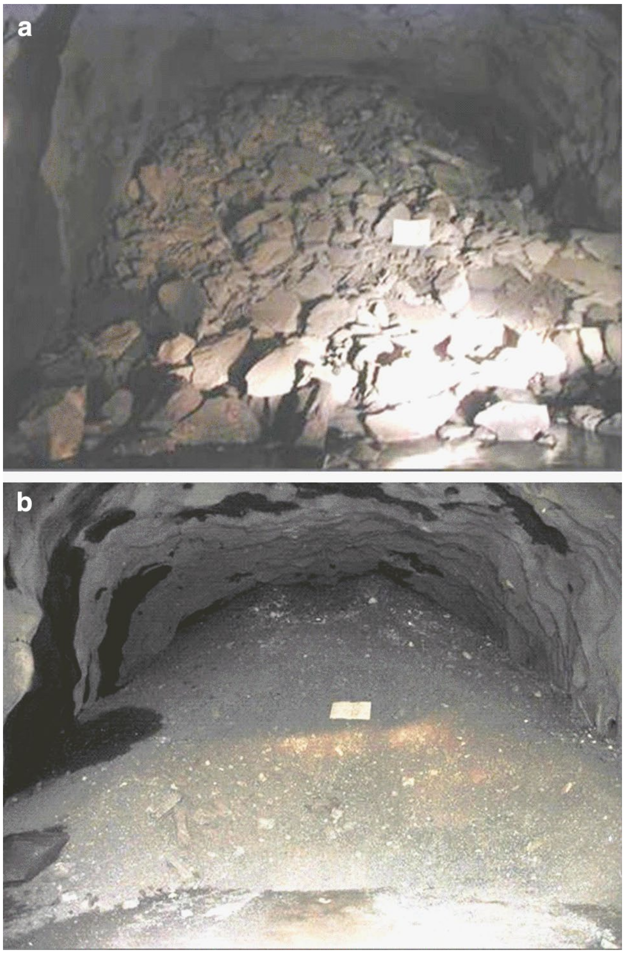


Fig. 2 Muckpiles of two ring blasts in sublevel caving (after Zhang 2016a). The pictures were taken after blasting but before extraction started, i.e., when the extraction is zero. The paper in the pictures has the same size of 40×50 cm and the drifts are 6.5 m wide and 5.5 m high. **a** Muckpile from the blast with a detonator position close to the collars of blastholes. **b** Muckpile from the blast with a detonator position at the midpoint of explosive charge length

the Boliden Kevitsa open pit mine (Ylitalo et al. 2021). For example, the number of boulders (> 1.0 m fragments) was reduced by more than 30% in five production blasts with the middle detonator position, compared with other production blasts with a close to bottom detonator position in the mine.

2.2 Ore Recovery

Rock fragmentation influences ore recovery through two mechanisms. (1) Better (finer) fragmentation can increase ore recovery ratio in some mining methods such as sublevel caving, since smaller ore fragments from blasting can more easily flow to draw points, resulting in a higher ore recovery ratio. (2) Better fragmentation may produce more inter-granular cracks and increase the ore recovery in mineral processing. Production blasts in sublevel caving in the LKAB Malmberget mine demonstrated that when detonators were placed at the midpoint of explosive charge length, the ore extraction ratio (the ratio of the extracted mass (volume) to the nominal ring mass (volume)) was increased by 110% in 93 production rings (Zhang 2005b), compared with that when the detonators placed at positions close to the collars of the blastholes (note that both ore recovery ratio and extraction ratio of a ring may be higher than 100% in sublevel caving). Similarly, midpoint detonator positions yielded a higher ore recovery ratio than the toe detonator positions in the Ridgeway gold and copper mine using sublevel caving (Brunton et al. 2010). In addition, when two detonators were placed at different locations in each hole and they were fired at the same time, the ore recovery ratio in the Malmberget mine increased, too (Zhang 2014). Even when one blast was split into two parts, the rock fragmentation was improved, the extraction ratio increased, and the final ore recovery increased in the Malmberget mine (Zhang and Wimmer 2018). Figure 3a shows the ore extraction ratio from 8 drifts (4 drifts with detonator position near the collar and 4 drifts with midpoint detonator position) at two production levels JH390 and JH437 of one ore body and 12 drifts in other ore bodies where two different detonator positions were tested in each drift. Clearly, the midpoint detonator position yielded a much higher ore extraction ratio than the near-collar detonator position. Moreover, the corresponding iron content from the midpoint detonator position was also increased; see Fig. 3b. Unlike the studies on ore recovery in mining, the number of studies on ore recovery in mineral processing is small.

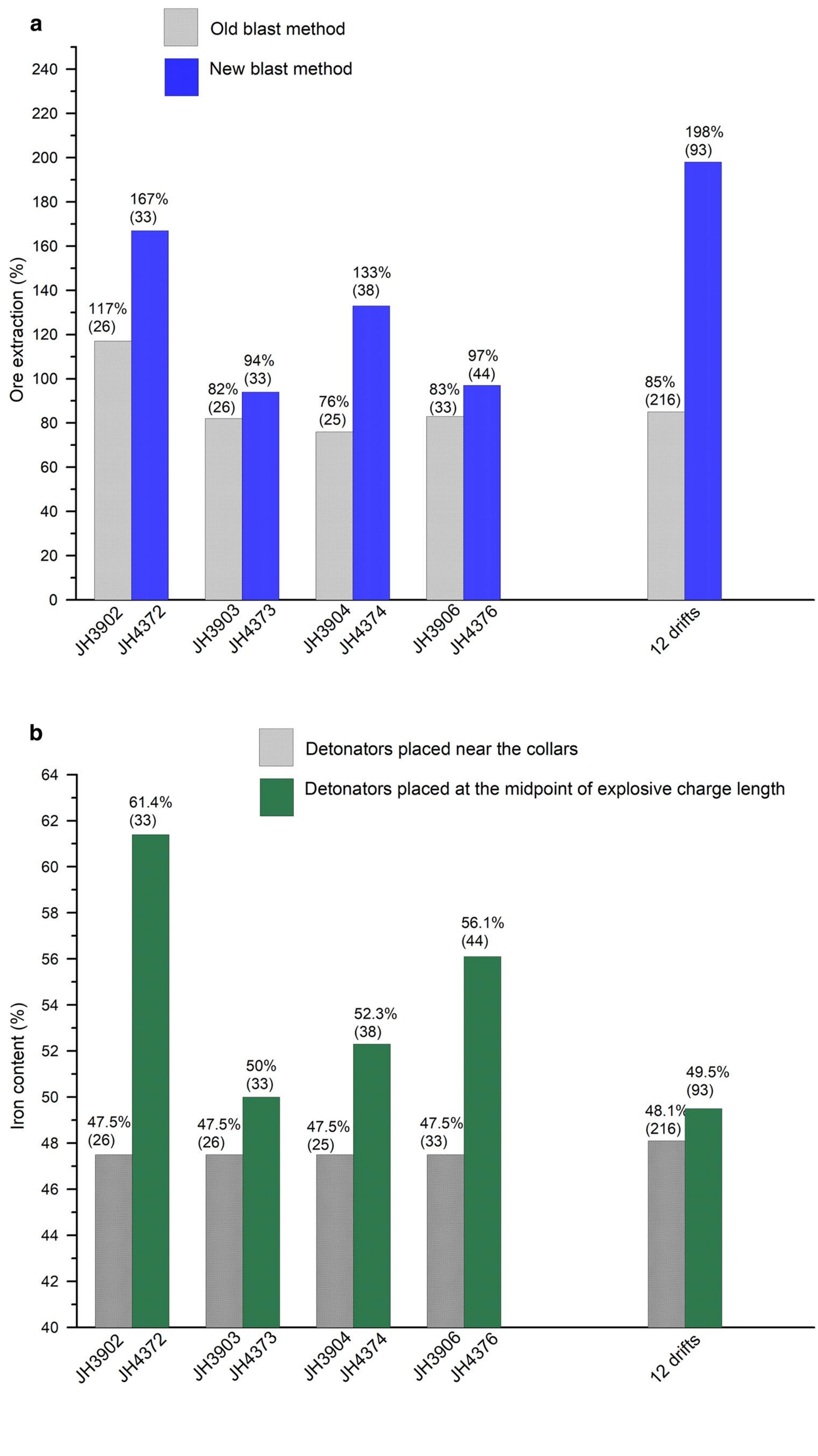
2.3 Productivity

Fragmentation has a strong impact on productivity such as extraction rate and mill throughput.

2.3.1 Ore Extraction Rate

Fragmentation and the associated number of boulders influence extraction rate (extracted tons per shift in Fig. 4) from different aspects such as bucket filling time, weight of loaded ore in a bucket, and so on. When fragments by blasting are small, the bucket of a loading machine or a shovel can be quickly filled up, and the number of boulders to be handled is small. Accordingly, ore extraction can be carried out efficiently. This case often appears in the beginning of the extraction in a sublevel caving ring, since the ore fragments mainly come from the lower part of the ring where the specific charge is higher than that in the upper part. This is why, one loading machine can extract more than 4000 tons of ore per shift, as shown in Fig. 4 (Zhang 2016a). When the extraction approaches completion, more and more boulders

Fig. 3 Average ore extraction ratio (**a**) and iron content (**b**) from ordinary blasts (grey bars) in a sublevel caving mine, with detonator positions close to the collars of the blastholes and blasts with detonator positions at the midpoint of explosive charge length (green bars, data from Zhang 2005b). The numbers in brackets are the number of the rings (blasts) behind each bar value. Since the iron content data of drifts JH3902, 3903, 3904, and 3906 were not available, the average of iron contents in the whole mine, 47.5%, was assumed to be the iron content in those four drifts. The four pairs of bars to the left are for one ore body, while the pair in the right gives average results for several orebodies



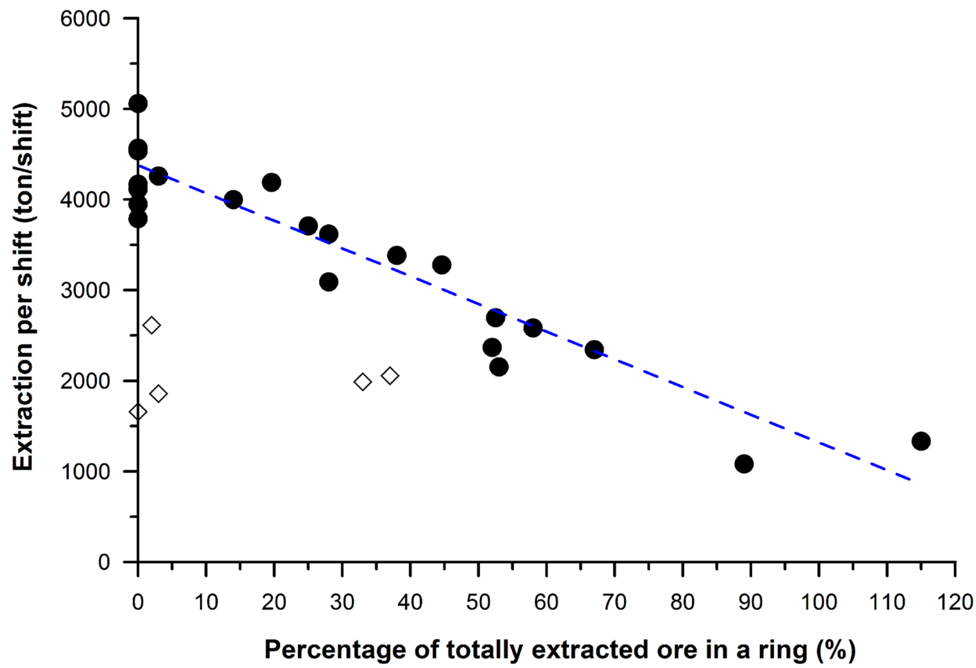


Fig. 4 Extraction rate (extracted tons per shift) vs percentage of totally extracted ore in sublevel caving rings according to production data (after Zhang 2016a). The empty diamonds represent five rings with special geological conditions. Note that the extraction ratio, which is defined in Sect. 2.2 as extracted mass (volume) over nominal ring mass (volume), is different from the extraction rate and can be more than 100% of the nominally blasted mass

come down, and such boulders must be broken by secondary fragmentation before they are loaded, resulting in lower extraction rates. Note that under some special geological conditions such as a high mixture of ore and waste rock in a ring, the extraction rate is not high even in the beginning of extraction, as shown by small squares in Fig. 4. Similarly, studies in open pit mining also show that better fragmentation (smaller fragments and fewer boulders) leads to better diggability (Ylitalo et al. 2021), reduced bucket-fill times, increased bucket-fill factors, and reduced shovel and truck maintenance costs (Orlandi and McKenzie 2006). Matters are not always as clear cut though. Quarry blasting results from Ouchterlony et al. (2010) showed that for pyrotechnic and electronic initiation at comparable specific charges in comparable blasting patterns, the electronic initiation gave 1) a coarser fragmentation, a smaller crusher flow and a smaller specific crushing energy, but 2) a significantly shorter bucket-fill time than the pyrotechnic initiation.

2.3.2 Mill Throughput

Mill throughput can be increased by improving fragmentation via blasting. This has been proven by mining production (e.g., Bergman 2005; Kojovic 2005; McKee 2013). In theory, fragmentation can be improved if the specific charge (when same explosive is used) is increased and the number of misfires is negligible. Highland Valley Copper experienced a decline in mill throughput after introducing larger blastholes for blasting, which resulted in coarser fragmentation and a coarser product from the primary crushers (Simkus and Dance 1998). Practices in many mines have shown that mill throughput increases with an increasing specific charge (e.g., Karageorgos et al. 2001; Lam et al. 2001; Paley and Kojovic 2001; Bye 2006; Paley 2010, 2012). Figure 5a shows the results from three mines—the Porgera mine in Papua New Guinea (Lam et al. 2001), the KCGM mine in Australia (Karageorgos et al. 2001), and the Red Dog mine in USA (Paley and Kojovic 2001). Figure 5b presents the results from the Sandsloot mine in South Africa (Bye 2006) and the Red Dog mine in USA (Ouchterlony and Paley 2013). Note that Fig. 5a uses mass specific charge and Fig. 5b uses volume specific charge to describe the 'intensity' of the blasts.

The mines in Fig. 5 used different explosives. For example, two different emulsions were used by Karageorgos et al. (2001), and one ANFO by Paley and Kojovic (2001). In addition, the rock masses and the mills were different in those mines, so that the relations between specific charge and mill throughput are different. However, in all five cases, a relation between mill throughput P_{mt} and specific charge q_w can be expressed by the following formulae:

$$P_{mt} = aq_w + b \tag{1}$$

where a and b are coefficients dependent on the properties of rock and explosive as well as other factors such as operation scale and blast design. One could argue that a non-linear dependence $P_{mt}(q_w)$ with a positive intercept at a critical specific charge and horizontal asymptote corresponding to free flow is physically more appropriate, but our data are too limited to go into such detail.

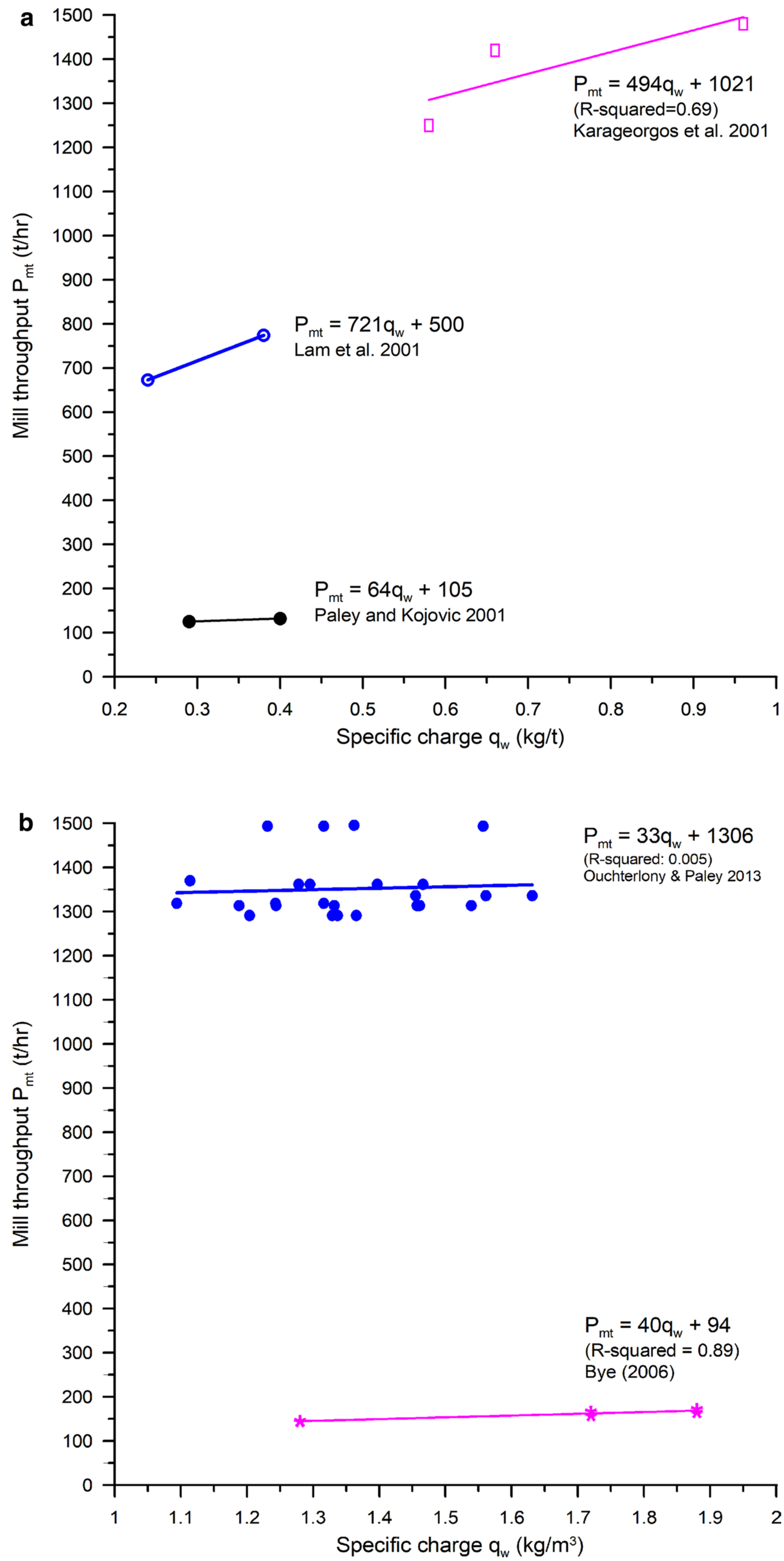


Fig. 5 Mill throughput vs specific charge. **a** Three mines—Porgera, KCGM, and Red Dog mines; the data come from Lam et al. (2001), Karageorgos et al. (2001) and Paley and Kojovic (2001). **b** Two mines—Sandsloot mine (Bye 2006) and Red Dog mine (Ouchterlony and Paley 2013)

2.4 Total Cost Covering Mining and Mineral Processing

The total cost from mining to mineral processing can be reduced by improving rock fragmentation. First, finer fragmentation may result in better crushing and grinding performance. For example, in the Luck Stone Bealon Quarry, USA, when specific charge was increased, the crushing energy decreased by 11% (Kojovic et al. 1995; Adel et al. 2006; McKee 2013). A similar result was found by Ouchterlony et al. (2010, 2015) using non-electric initiation in the Långåsen quarry. A higher specific charge, 0.99 vs. 0.72 kg/m^3, (+38%) gave both a markedly finer median fragmentation, 120 vs. 160 mm (-25%), a faster bucket filling time, 25 vs. 35 s (-39%), a higher crusher flow, 400 vs. 380 ton/hr (+5%) and a lower specific crushing energy, 0.25 vs. 0.30 kWh/ton (-17%). In an open pit mine in South America, when specific charge was increased by 40%, the mill throughput was increased; see Table 1. Second, a finer fragmentation leads to a higher ore extraction rate in mining and a larger mill throughput in mineral processing, as mentioned above (Sect. 2.3).

There is more evidence that blasting affects crushing and grinding results, and that large cost savings can be accrued (Eloranta 1995; Paley and Kojovic 2001). In the quarry Vrsi, when the drilling pattern decreased from 3.0 m × 4.5 m to 2.9 m × 3.0 m, while other parameters, such as borehole sizes, were constant, a significant saving of 14% was achieved for the quarry (Strelec et al. 2000). Due to a mine-to-mill implementation at the Red Dog Mine, the mine achieved savings exceeding $30 million per year (Paley and Kojovic 2001). The same project identified a further benefit, a marked reduction in SAG feed size, and throughput variability (Kojovic 2005). A second but important benefit was the reduced wear in the gyratory crusher, resulting in a significantly longer period between relines.

2.5 Mining Sustainability and Environment

Since better fragmentation can increase the ore recovery ratio, the ore losses will decrease and more mineral resources can be 'saved' and mining sustainability be improved, as descried in Sect. 2.2.

If rock fragmentation is improved, more of the energy fed into mining and mineral processing can be gainfully utilized instead of wasted as discussed in Sect. 2.1. According to the study by Norgate and Haque (2010) mentioned earlier, crushing and grinding made a contribution of about 21% and 47%, respectively, to the total greenhouse gas emissions for the mining and processing of iron ore and for the production of copper concentrate. Thus, to achieve mining sustainability and reduce the negative impact of mining on the environment, it is of importance to increase ore recovery and decrease energy consumption in ore crushing and grinding by improving rock fragmentation through blasting.

3 Factors Influencing Rock Fragmentation

Rock fragmentation by blasting takes place in a system that applies disruptive loads (including explosive and initiator, normally a detonator), to an object (rock), and the surrounding conditions, as shown in Fig. 6a. The production blasting takes place either on the surface (usually through bench blasting) or underground (through bench, ring, or drift blasting in some form). The rock fragmentation depends not only on the energy input to the system, but also on the energy distribution or effective energy used in rock fragmentation. In this sense, most factors have been discussed elsewhere (e.g., Ouchterlony 2003a, 2010; Zhang 2016a) and can be divided into three groups: (1) explosive and initiator, (2) rock, and (3) energy distribution and energy efficiency, as shown in Fig. 6b.

There are a number of fragmentation models for bench blasting. Such models (see details in Sect. 5) describe the compound effect of a limited number of the above-mentioned

Table 1 Crushing energy (kwh/t) or mill throughput (tph) versus specific charge

Mine	Specific charge (standard)	Crushing energy or mill throughput (standard)	Specific charge (mine to mill)	Crushing energy or mill throughput (mine to mill)	Decrease in crushing energy or increase in mill throughput by mine to mill (%)	Reference
Luck stone Bealon quarry	0.26 kg/t	1.77 kwh/t	0.47 kg/t	1.57 kwh/t	− 11	McKee (2013)[a]
Open pit mine in South America	1.15 kg/m^3	3500–4000 tph	1.62 kg/m^3	5000 tph	25–40	Dance et al. (2007)

[a]Original data from Kojovic et al. (1995) and Adel et al. (2006)

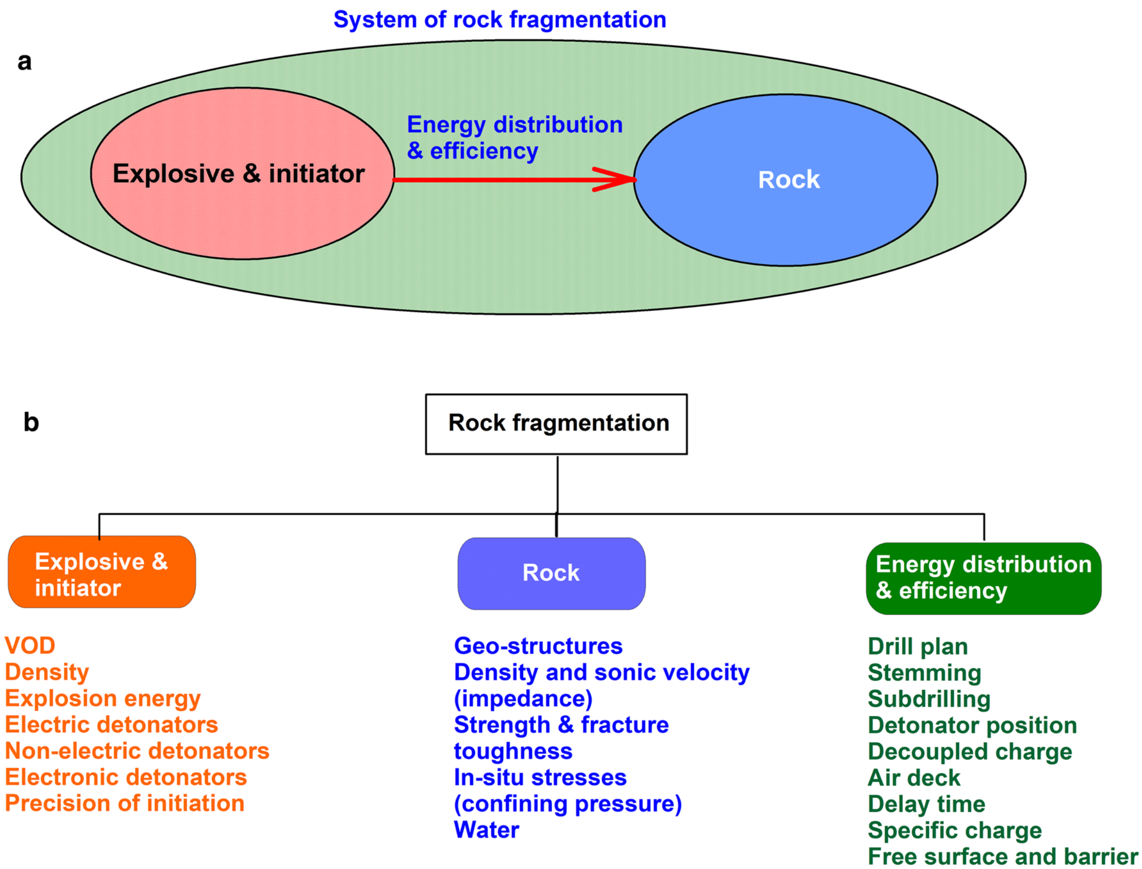


Fig. 6 Main factors influencing rock fragmentation by blasting

factors on the fragmentation, indicating that those models are far from complete. Ring or fan blasting models are rudimentary in comparison. Most of the factors influencing fragmentation are listed in Fig. 6b and they are discussed in the following.

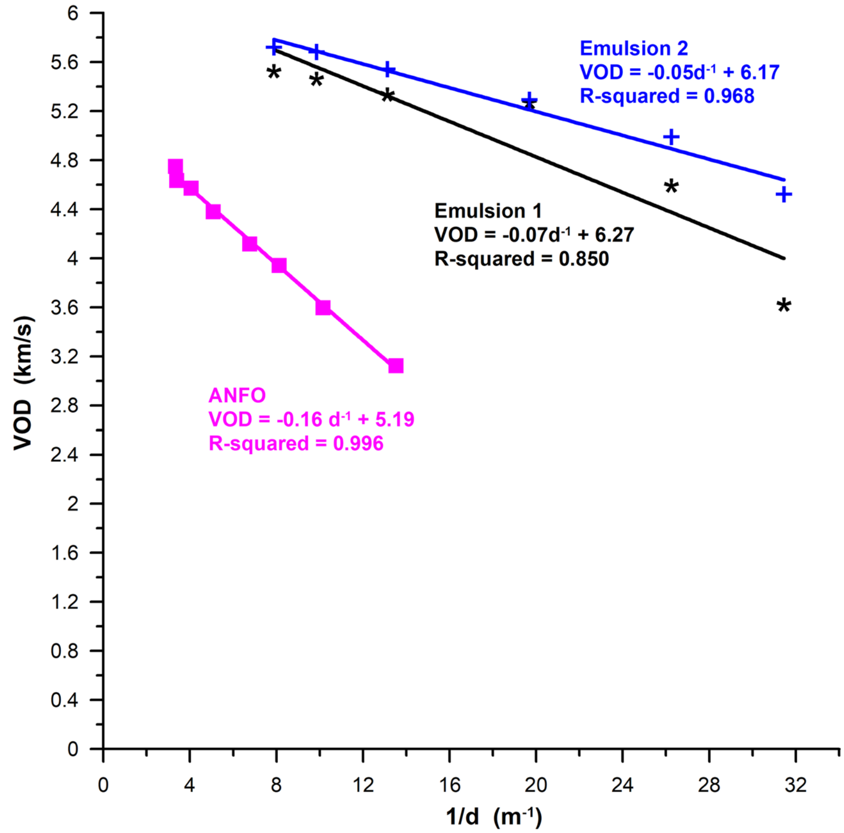
3.1 Explosive and Initiator

3.1.1 Explosive

The explosion (or heat) energy, the VOD (velocity of detonation), and the density of an explosive have a varying degree of impact on rock fragmentation. The explosion energy can be determined by different methods, such as thermodynamic codes and experimental measurements (Sanchidrian et al. 2007). The latter include the underwater test (Bjarnholt and Holmberg 1976; Mohanty 1999) and the cylinder test (Nyberg et al. 2003). Since there is no universal test method for determining the explosive energy, thermodynamic calculations are often used to assess the energy of explosives.

Velocity of detonation (VOD) is one of most important properties of an explosive, because the peak pressure of the detonation wave increases with an increasing VOD according to one-dimensional (1D) detonation theory (e.g., Cooper 1996; Zukas and Walters 1997; Fickett and Davis 2000; Zhang 2016a). The 1D theory indicates that $p_{CJ} = \rho \bullet \frac{VOD^2}{\gamma+1} \approx \rho \bullet \frac{VOD^2}{4}$ where p_{CJ} is the CJ (Chapman-Jouguet) detonation pressure, ρ the initial density, and γ the adiabatic gamma of the explosive, respectively. The CJ detonation pressure is valid for a radially stiffly confined cylindrical charge, or for a charge of infinite diameter d, inside which the reaction zone is negligibly thin and the chemical reaction occurs instantaneously. For finite charges, the VOD increases with an increasing diameter of the explosive charge, gradually approaching a limit value at large diameters (e.g., Sun et al. 2001). When plotting VOD data vs $1/d$, the data are, in many cases, well represented by a straight line, as shown in Fig. 7. The corresponding behaviour in linear VOD vs d space is shown in Sun et al. (2001) or in Fig. 8.7 of Zhang (2016a).

Fig. 7 VOD vs inverse of diameter of blasthole (full charge) (based on the measurement data of Sun et al. 2001)



VOD has an impact on rock fragmentation (e.g., Ouchterlony 2003a; Zhang 2016a). For example, blasting experiments by Bergmann et al. (1973), Nielsen and Kristiansen (1996), and Michaux and Djordjevic (2005) showed that the explosive with higher VOD yielded finer fragmentation, even if Ouchterlony (2003a) questioned Bergmann's et al. conclusion. Similarly, experiments by Kurokawa et al. (1993) indicated that fracture surface area slightly increased with an increasing VOD (see Fig. 5 in their article).

Rock fragmentation is not always improved by increasing VOD, for example, when the VOD of explosive is smaller than the P-wave velocity of the rock mass to be blasted, misfires in the explosive or in the detonator could occur (Zhang 2016a). The reason is that the P-wave can, via the rock, run ahead of the detonation and propagate into the undetonated explosive and make it fail through so-called dead pressing. If further, there is a detonator in the undetonated explosive, the detonator might be damaged before firing. Such damaged detonators were found in multi-decked blastholes (e.g., Farnfield and Williams 2011; Mencacci and Farnfield 2003). Thus, the rock with high P-wave velocity should be blasted using high-VOD explosives, while the rock with low P-wave velocity be blasted using low-VOD explosives according to the analysis by Zhang (2016a). For example, the emulsions can be used in relatively small holes, but the ANFOs need be used in large ones to have a sufficiently high VOD (see Fig. 7). However, this conjecture still needs more production blasts to be verified. Furthermore, emulsions are water resistant but ANFOs not, so in practice mixtures or blends of ANFO and emulsion are widely used, so that they can be loaded in wet holes and relatively hard rock (Olofsson 1999; Zhang 2016a).

The above description indicates that change in VOD sometimes does influence rock fragmentation but not always. Similarly, Ouchterlony (1997), using blast results from Jokinen and Ylätalo (1995, 1996), showed that for low-VOD explosives, the crack length increased with increasing VOD, but for high-VOD explosives, the crack length did not depend on the VOD. Therefore, it is necessary to perform more studies on the relation between VOD and fragmentation as well as crack length, taking into account that different kinds of blasting and different rocks may give different results.

3.1.2 Initiator

The initiator of the blasthole charge can be a detonator or a detonating cord. Of three common types of detonators, electronic delay detonators have the most accurate initiation time, and some of them have an initiation scatter of less than 1 ms. Electric and non-electric detonators with a pyrotechnic delay element have a much larger initiation error, in mining practice often several or tens of milliseconds, meaning that they may fire considerably earlier or later than their nominal initiation times. This error may bring about poor fragmentation in mining production or deeper blast damage in contour blasting. Electronic detonators bring an enormous flexibility in time selection allowing a very different neighbouring holes interaction and rock movement patterns, besides being very accurate, and they can be used to improve contour blasting and vibration control where an accurate initiation is required. However, there is no guarantee that better fragmentation can be achieved using electronic detonators. Only accurate initiation time, without correct delay time and other parameters, is seldom enough to yield better fragmentation. In spite of the fact that electronic detonators have many advantages over pyrotechnic detonators such as Nonel, the cost of electronic detonators is still much higher than that of pyrotechnic detonators. In many cases, if the delay times are carefully designed, pyrotechnic detonators can yield similar results as electronic detonators.

3.2 Rock

Rock fragmentation by blasting is potentially influenced by many rock-related factors. A number of them are shown in Fig. 6. They may be divided into groups as follows: (1) geo-structures such as faults, joints, bedding planes, and other discontinuities; (2) density, sonic velocities, strengths, fracture toughness, etc.; (3) in-situ stresses; (4) other environmental conditions such as water, etc. The data in group (1) are normally called rock mass properties and those in (2) rock material properties.

Geo-structural conditions, such as faults, joints, bedding, and intrinsic cracks, in rock mass usually influence stress wave and shock wave propagation (Zhao and Cai 2001; Li and Ma 2009). As a result, they affect rock fragmentation if the wave amplitude exceeds the strength of rock. In mining production, a slipping fault in an ore body may cut (shear) a blasthole and finally leave it uncharged, resulting in poor fragmentation, as shown in Fig. 8a. Notice that if a discontinuity like a joint or crack is open or close to being a completely or partially free surface, the crack will have a large impact on the fragmentation due to wave reflections and crack stopping effects (Zhang 2016a). If a crack is closed and it is located within a rock mass, these effects will be much smaller.

High in-situ stresses affect rock fragmentation. When in-situ stresses are high, in-situ rock strengths will be high too, due to a large effect of confining pressure on rock strengths and fracture toughness (e.g., Goodman 1989; Jaeger et al. 1969; Zhang 2016a). For instance, the high in-situ stresses may cause borehole breakage and make it impossible to load explosive into the borehole, as shown in Fig. 8b. This phenomenon is common in deep mining where in-situ stresses are often high (Ghosh et al. 2015). The above-mentioned shear or breakage in boreholes is one of reasons for the misfires occurring in the underground mines.

A rock with greater specific fracture energy requires more energy to break (Zhang and Ouchterlony 2022). Fragmentation is also affected by water and temperature in the rock. Under static loads or low loading rate conditions, wet rock is easier to fracture, since its strength and fracture toughness values are often smaller than those for dry rock (Feng et al. 2001; Yilmaz 2010; Willard and Hjelmstad 1971; Swolfs 1972; Singh and Sun 1990; Haberfield and Johnston 1990; Lim et al. 1994). Under dynamic loads, wet rock with water filled pores is a better wave transmitter than dry rock, see,

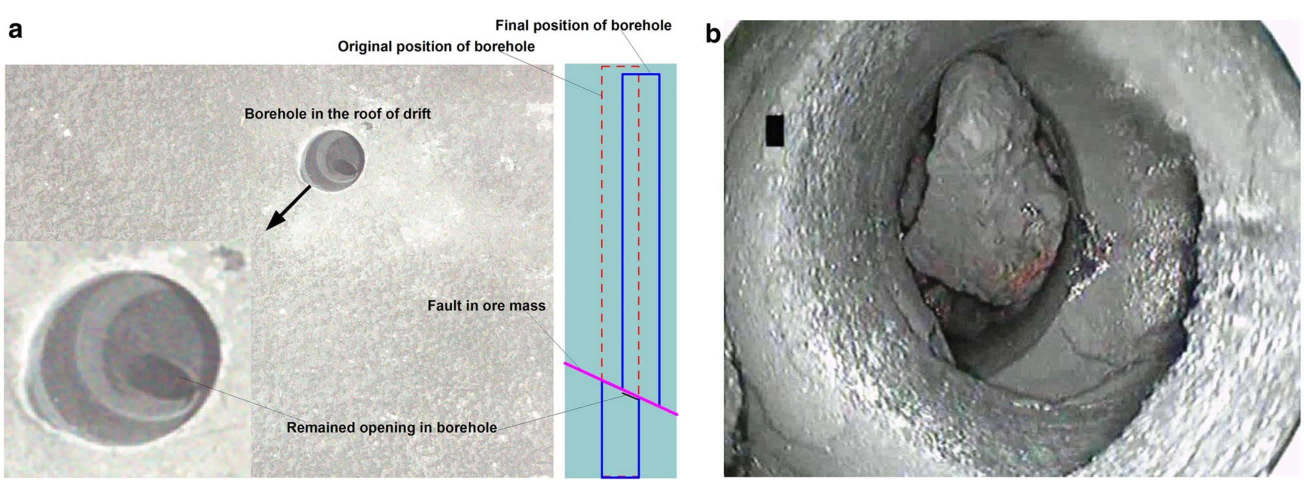


Fig. 8 Borehole sheared by fault (**a**) and borehole broken due to high in-situ stresses (**b**) (after Zhang 2016a)

e.g., Tilert et al. (2007), who found that a soaked specimen of fine-grained granite had a P-wave velocity of 4840 m/s, 8% higher than that for dry specimens. Accordingly, rock fracture and fragmentation have been found to be better in wet rock (Fourney et al. 1981; Tilert et al. 2007; Zhang 2016a).

Although rock fragmentation is affected by geo-structures and other rock mass properties, it is still difficult or expensive to detect them in mining engineering. To overcome this, a physical property of the rock mass—characteristic impedance or acoustic impedance (the product of the density and the sonic velocity of the rock mass)—may play an important role, since it describes the wave reflectivity at discontinuities like joints and cracks in the rock mass (Zhang 2016a; Zhang et al. 2020c, d). In particular, since the sonic velocities of rock masses can be measured in the field by a seismic system (or vibration monitors) and the densities of rock masses may be determined by muography (Zhang et al. 2020c; Holma et al. 2022) or a geophysical method, the impedances of rock masses can be determined and the rock masses may be evaluated. In this way, a blast design can be made using more detailed rock mass information.

3.3 Energy Distribution and Efficiency

Energy distribution and energy efficiency in rock blasting are to a great extent dependent on the blast design. Accordingly, the following description deals mainly with that topic.

3.3.1 Drill Plan

A drill plan involves diameter, length, burden, spacing, subdrilling, number, and distribution of boreholes in a blast. The diameter of boreholes can be chosen empirically (e.g., Adhikari 1999). The diameter must be larger than the critical diameter of the explosive to avoid malfunction of the explosive (Zhang 2016a). The length of boreholes is mainly dependent on the mining plan and the deviation. Since the bottom deviation of a borehole may be up to 10% of the nominal burden (Quinteiro and Fjellborg 2008; Sellers et al. 2013), the length of boreholes for mining production should not be too long to avoid either an increased or a decreased actual burden/spacing due to the deviation. Regarding the length of boreholes, subdrilling is another parameter to consider. Subdrilling is often used in open pit mining and its length is dependent on the blasting technique, such as primer placement and explosive charging plan. For example, when air decks are placed at the bottoms of boreholes in open pit blasting, subdrilling may be reduced, compared with similar blasting without air decks. In multi-hole and multi-row blasting, spacing and burden have an impact on rock fragmentation. Similar to other parameters in a drill plan, spacing and burden are often determined by empirical methods (e.g., Langefors and Kihlström 1967; Kou and Rustan 1992; Zhang 2016a).

3.3.2 Stemming

In civil and mining engineering, stemming is widely used in surface blasting or blasting in downward-drilled-holes, but not always in underground blasting with upward-drilled-holes, even though there are good examples of using stemming in underground mines (Oates and Spiteri 2021). In tunnelling, stemming is not always employed either. Measurements in an underground mine indicated that as much as 50% of the explosive energy escaped from blast holes in the form of gases when the holes were not stemmed (Brinkmann 1990). Model blasts (Fourney et al. 1988) indicated that the peak pressure in a stemmed hole was approximately two-to-three times higher than that in an unstemmed hole. Small-scale blasts (Zhang et al. 2020a) showed that at least 25% of the explosive energy was wasted when no stemming was used, i.e., to get the same median fragmentation for a specimen with an open collar as for a stemmed collar required 25% more explosives. Mining production blasts demonstrated that when drill cuttings were replaced by aggregates as stemming, rock fragmentation and mill throughput were improved (Kojovic 2005). Quarry production blasts indicated that longer (4.5 m) stemming yielded more boulders than shorter (3 m) stemming (Cevizci and Ozkahraman 2012). Cevizci (2012) found that a plaster stemming method yielded better fragmentation and lower cost than the drill cuttings stemming method in the blast tests of three quarries. Small-scale blasts (Zhang et al. 2021b) showed that different types of stemming affected rock fragmentation, e.g., full stemming with sand yielded better (finer) fragmentation than partial steel stemming. Theoretically, both the material and the length of the stemming influence rock fragmentation, and the length of the stemming can be determined by some formulas based on the VOD of explosive, the P-wave velocity of rock, the charge length, and one coefficient (Zhang 2016a). However, those formulas need to be validated by field trials.

3.3.3 Coupled and Decoupled Charges

A radially decoupled explosive charge usually gives rise to shorter radial cracks in the remaining rock mass than a full explosive charge. For example, when a Gurit explosive with a 22 mm diameter was charged in a dry borehole with a 64 mm diameter, the length of the radial cracks induced was about 15 cm behind the half-casts. However, the same explosive in a borehole with 24 mm diameter resulted in that the lengths of the radial cracks became up to 1 m (Olsson and Bergqvist 1997; Ouchterlony et al. 2002). In practice, decoupled explosive charges are often used either to reduce vibrations from blasting or to produce smooth surfaces in presplitting and smooth (cautious) blasting, while fully explosive charged holes are normally used in mining production blasting to maximize breakage and produce an acceptably fine fragmentation (Zhang 2016a).

In a special case where the amount of extra-fine particles must be reduced to increase ore recovery in mineral processing, it may be a good option to use decoupled charges instead of fully coupled ones. This is supported by two groups of measurement results. One group of results indicates that the crushed zone is an important source of fine particles in blasting and around 50% weight of the fine particles (smaller than 1 mm) is generated in the crushed zone of a fully charged (i.e., decoupled ratio is zero) blasthole (Svahn 2002; Reichholf 2003). The other group shows that a decoupled charge produces a smaller ratio of crushed zone diameter to borehole diameter than a full charge, and this ratio decreases with increasing decoupling ratio according to Chi et al. (2019c) who summarized their own results and the ones from Iverson et al. (2009) and Sun (2013). Note that if the air gap between charge and blasthole wall is too large, the so-called channel effect will often cause dead pressing and detonation cut off. This puts a higher limit on the decoupled ratio.

3.3.4 Specific Charge

The specific charge or powder factor represents the average explosive weight per volume or per weight of rock to be blasted, e.g., kg/m^3 or kg/ton. If misfires or malfunctions do not occur in the explosive and detonators, a higher specific charge means more explosion energy supplied per volume or per weight of rock, which should, in general, result in a finer fragmentation (Zhang 2016a). Laboratory experiments have indicated that a larger specific charge increased both crushability and grindability of a low-grade quartz banded iron ore (taconite), a nepheline syenite ore, and an ilmenite ore (Nielsen and Kristiansen 1996; Ouchterlony 2003b). Similar experiments showed that the oxidized copper ore blasted with higher specific charge yielded a larger surface area than that with lower specific charge per gram of ore (Fribla 2006). In addition, Fribla found that the Cu recovery increased with increasing explosive charge.

A higher specific charge does not necessarily result in finer fragmentation if misfires occur. Therefore, when a blasting plan is made, the possibility of misfires must be considered, especially when use of a high specific charge is planned. In addition, when different explosives are used in multiple blasts, one should be careful to compare blast results by specific charge value in kg/m^3 or kg/ton alone, since different explosives usually have different explosion/heat energies (per volume or per weight of explosive). Another issue is that a very high specific charge normally causes much more fines. If these fines consist of ores and they are too small to be recovered by current mineral processing technology, such a high specific charge should not be used.

3.3.5 Detonator Position

Stress wave analysis has demonstrated that if each hole has only one detonator, the best detonator position in sublevel caving rings and bench rounds is the midpoint of the explosive charge length (Zhang 2005a, 2016a; Ylitalo et al. 2021). Numerical simulation (Long et al. 2012; Menacer et al. 2015; Liu et al. 2015) and mathematical analysis (Gao et al. 2020) yield the same result as the stress wave analysis. Similarly, stress wave and shock wave analysis has shown that for long holes with two detonators, there is an optimum location for them (Zhang 2014). Both underground mining and open pit mining blasts have confirmed that detonators located at the optimum locations yield finer fragmentation (Zhang 2005a,b; Ylitalo et al. 2021) and higher ore recovery (Zhang 2005a,b, 2014; Brunton et al. 2010). Regarding detonator position, more trials are expected.

3.3.6 Delay Time

Early studies have shown that the delay time influences fragmentation (Langefors and Kihlström 1967; Winzer et al. 1983). If stresses from two (or more) delayed blastholes are effectively superimposed on each other, the final stresses at certain areas or points in the rock will increase considerably. As a result, the final energy utilization in blasting will markedly increase, at least locally. However, this would happen only when the delay time between the two holes is very short compared to normal delay times. In addition, detonators with an accurate initiation time, e.g., electronic detonators, are required. For other types of detonators such as pyrotechnic delay detonators, the aforementioned stress superposition is very limited or impossible, since their delay times usually are too imprecise. Even in the case that electronic detonators are used in multi-hole blasting, if a delay time of 30 ms or longer is employed, it is not possible to achieve stress superposition between two adjacent holes, since the total wave length of a current production blasthole less than 35 m is often shorter than 30 ms (measured at a distance of 60–90 m from the blasting holes) (Zhang 2016a). Rossmanith (2002) and Rossmanith and Kouzniak (2004) using stress wave theory described how a positive effect of stress wave interaction could be achieved between two blasting holes with short delay time between. Full-scale blasts with short delay times by Vanbrabant and Espinosa (2006) indicated that the fragmentation was improved, but other full-scale and small-scale blasts (e.g., Stagg and Nutting 1987; Katsabanis et al. 2006; Johansson and Ouchterlony 2013; Petropoulos et al. 2014) did not produce better or much better fragmentation when very short delay times were used. However, looking at the result of full-scale blasts by Petropoulos et al. (2014) (see the result also in Yi et al. 2017) in more detail, the production blast with 3 ms delay time yielded the best fragmentation, compared to the blasts with 1 ms, 6 ms or 42 ms delay times, meaning that it cannot be excluded that a proper short delay time may give rise to better fragmentation via stress wave interaction between two adjacent blastholes. Sanchidrián and Ouchterlony (2017) suggest a relatively long optimum delay time in their xp-frag model. Analysis of fragmentation through the size-energy fan on a set of blasts in a quarry leads to a much shorter optimum delay (Sanchidrián et al. 2022). The value of the optimum delay has been a topic of debate in the blasting community that remains a matter of discussion, and so is the amount of size reduction (e.g., from an instantaneous blast) with such an optimum delay.

3.3.7 Air Deck

The air deck technique, i.e., leaving one (such as the bottom) or several parts of the blasthole empty and uncharged, was originally developed by Melnikov and Marchenko in the 1950s (Melnikov et al. 1978). This technique has two important effects: (1) to reduce the amplitude of the initial shock waves propagating into the rock surrounding the borehole and (2) to increase the total length of the shock wave or stress wave traveling in the rock. These effects have been confirmed by laboratory experiments (Fourney et al. 1981, 2006; Marchenko 1982). In several mines, the air deck technique increased the excavator efficiency (Melnikov et al. 1978), reduced the explosive consumption (Melnikov et al. 1978; Mead et al. 1993; Correa 2003), improved rock fragmentation (Jhanwar et al. 2000), avoided subdrilling (Correa 2003), and decreased the blast-induced vibrations (Park and Jeon 2010). In addition, water decks were tested in a quarry and the result showed that this technique produced a flat floor and a satisfactory fragment distribution without boulders (Jang et al. 2018).

3.3.8 Free Surface and Barrier Nearby

A free surface near or close to and approximately parallel to a blasthole plays an extremely important role in rock fragmentation. Usually, this free surface exists, but sometimes not, like in crater blasting or in the later parts of large open pit rounds. Take another example in a hanging roof and cut blasts for tunnelling, such a free surface is either absent or of limited extent (Zhang 2016a, b). In production blasts of sublevel caving mining, such a free surface is generally absent, since the just created face is somewhat constrained by moveable waste rocks or ore materials (Janelid and Kvapil 1966; Johansson and Ouchterlony 2013; Zhang 2014; Zhang and Wimmer 2018). In addition, even though a free surface exists in a multi-hole blast, a barrier may stand in front of a free surface, e.g., in open cut blasting. Previous studies (Duvall and Atchison 1957; Hino 1959; Field and Ladegaard-Pedersen 1971; Fourney et al. 1981; Wilson and Holloway 1987; Rossmanith and Uenishi 2006; Fourney 2015; Zhang 2016a) demonstrated that stress waves play an important role in rock blasting. E.g., a free surface makes an impinging compressive stress wave reflected into a returning tensile wave that may result in tensile fracture—spalling. In addition, Fourney (2015) showed that the S-wave induced by the reflection of the compressive wave at the free surface might cause cracks in the radial direction of a blast model. Interestingly, in model blasting (Chi et al. 2019a; Zhang et al. 2021a), some radial cracks were discovered on the free surfaces of the rock specimens, but no gas was ejected out of such radial cracks, meaning that these radial cracks were initiated from the free surfaces rather than from the blastholes. This is consistent with the analysis by Fourney (2015) to a certain extent.

Cylindrical model blasts (Zhang et al. 2020b) found that when a cylindrical model was surrounded by a concentric cylindrical steel tube with air in the gap between the model and the tube a better fragmentation was achieved than when a model with a partially constrained free surface and a model with a complete free surface. The major reason is that the barrier is impacted by flying fragments that in turn fragment. The blasts with a partially free surface caused a coarser fragmentation than the two other test set-ups. The main reason was that there was no completely free surface, resulting in a smaller reflected tensile wave. In various types of open cut blasting and mining blasting below a hanging wall, a barrier exists in front of the free surface. This has been studied by Rustan (2013) and Zhang (2016a,b; 2017).

3.3.9 Confining Condition

Blasting against compacted rock fragments has been studied experimentally (Jarlenfors 1980; Wimmer and Ouchterlony 2011; Johansson and Ouchterlony 2013; Sun 2013; Rustan 2013; He et al. 2018; Chi et al. 2019b, c; Petropoulos et al. 2018; Zhang et al. 2020b). Blasts against compacted rock masses indicated that the finest fragmentation came from 100% swelling, while the coarsest fragmentation from 12.5% swelling (Jarlenfors 1980). Although the previous studies dealt with blasting against compacted materials, quantitative studies on rock fragmentation have been few so far (Johansson and Ouchterlony 2013; Chi et al. 2019c; Zhang et al. 2020b). The small-scale blasts using mortar and granite indicated that the confined blasting yielded coarser fragmentation than unconfined blasting (i.e., blasting with free surface) (Olsson 1987; Johansson and Ouchterlony 2013; Chi et al. 2019c).

4 Fines, Ore Blending, and Ore Sorting

4.1 Definition of Fines

Fines is a concept that depends a great deal on the type of operation and its economical parameters. In some aggregate quarries, the fragments (particles) smaller than 4 mm can be called fines, since such small particles often cannot be sold (Moser 2004). In mineral processing of hard ores, ore particles smaller than 50 microns are defined as fines in this article, because such small particles, especially smaller than 20 microns, are often difficult to concentrate using modern processing technology (Wills and Napier-Munn 2006).

4.2 Weight Percentage of Fines from Blasting

Model blasts using 9 granite cylinders show that the fine particles smaller than 50 microns are in a range of 0.07%—0.21% of the total weight of each granite cylinder (Zhang et al. 2021b). Using the higher ratio 0.21% and assuming that one iron ore mine produces 20 Mt crude ore per year, the total fines smaller than 50 microns will be up to 42,000 t per year. Since the specific charge in the model blasts is only 0.22 to 0.29 kg/m^3, which is significantly lower than the common specific charge used in real mines, the percentage of the fines smaller than 50 microns in a real mine could be significantly higher than 0.21%. Assuming that the fines fraction is just doubled, the annual production of fines from blasting would be 84,000 t. Furthermore, assuming that a half of such fines (<50 microns) is high-grade iron ore and the price is 200 USD/t, the value of the 84,000 t fines will be 16.8 MUSD. Thus, this value is not ignorable.

4.3 Source of Fines in Rock Blasting

The measurement (Svahn 2002; Reichholf 2013; Kukoli 2021) and numerical simulation (Iravani et al. 2018) showed that fines were produced in both inside and outside the crushed zone due to crack branching. For instance, for iron ore, magnesite, and limestone, around 50% weight of the material smaller than 1 mm was generated within the crushed zone of blasthole, meaning that other 50% fine particles were not from the crushed zone (Reichholf 2013).

Nielsen and Kristiansen (1996) did a series of full-scale blasts with different borehole diameters: 76, 89, 102, and 114 mm. After blasting, the muck was crushed to the particles smaller than 70 mm, and then screened at 32 mm. It was found that the proportion of particles smaller than 32 mm after crushing increased with increasing drillhole diameter at constant powder factor, as shown in Fig. 9. This result indicates that the larger holes will produce more particles smaller than 32 mm including the fines smaller than 50 microns. Notice that this result is a combined effect of the blasting and crushing and more detailed blast tests are needed to confirm if this is true also for the blast generated fines. On one hand, a larger crushed zone means that more small particles are produced by blasting and such small particles excluding the fines are beneficial for mining production and even for downstream crushing and grinding. On the other hand, the fines must be reduced in blasting. This is a dilemma and big challenge for the researchers and engineers in mining engineering.

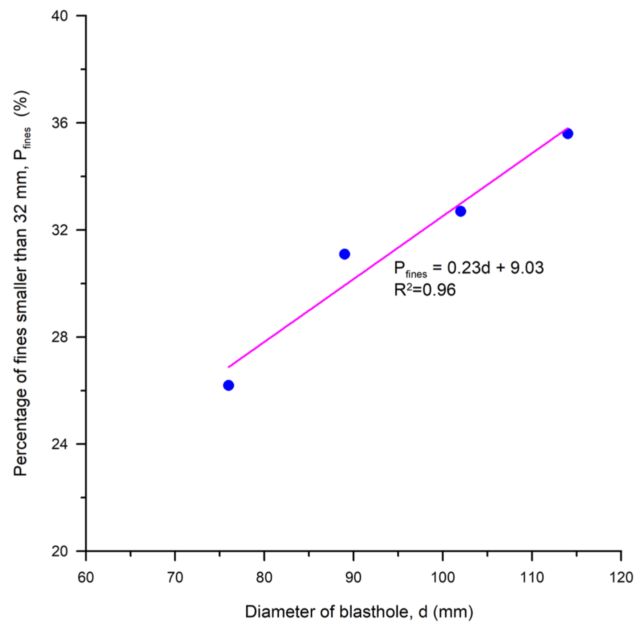


Fig. 9 Diameter of blasthole vs percentage of particles smaller than 32 mm (based on the data of Nielsen and Kristiansen 1996)

4.4 Ore Blending

In a metal mine, the grade and mineral composition of the ore in one part may be different from that in another. It is important to understand the ore variability through the deposit and through the life of the mine. To provide a steady and predictable feed to the mineral processing plant, different ores from different parts of the mine can be blended in a proper proportion. This method is called ore blending and it is used in many mines. For example, after ore blending in a mine-to-mill project, the Thalanga mine achieved higher recoveries in Zn, Cu, and Pb (McKee 2013). The purpose of ore blending is simply to provide uniform mill feed, so that the downstream processes can be fine-tuned, which aids in improving production efficiency and reducing production cost.

The process flow sheet of a concentrating plant is typically optimized for a certain defined head grade, and consequently, it is important to keep the feed quality constant to be able to optimize the quality of the final product. To make a rational use of resources and to increase the life of mines, operators need to balance the use of high-grade ore and low-grade ore as much as possible (Liu et al. 2021).

4.5 Ore Sorting

Ore heterogeneity could be a big problem in optimizing the mill feed and there is a great need to reduce the amount of waste materials before they enter the energy intensive grinding circuit. This requires not only efficient blending but also detection of the heterogeneity of the primary crushed ore. Ore sorting can be done by analysing the materials continuously, e.g., with an XRF analyser (Auranen et al. 2021). There are several pre-concentration technologies that can be applied at each stage of mineral processing and one of the most recent methods is Bulk Ore Sorting (BOS) which mechanically separates ore from waste rock before the materials enter the grinding and can potentially reduce processing costs and improve mine economics (Li et al. 2019).

5 Fragmentation Prediction Models and Formulae

A broad review of prediction equations for blast fragmentation has been done by Ouchterlony and Sanchidrián (2019). Most prediction equations apply to bench blasting; however, they can with caution probably be applied to other blasting geometries by adapting the different parameters (burden, spacing, etc.).

5.1 Size Distribution Functions for Rock Fragments: The Rosin–Rammler–Weibull

The classical approach to predict fragmentation is to assume a distribution function and use empirical formulae to calculate its parameters. To the question of what distribution should be used, there is not a simple answer and this is to be studied. Probably, the first contribution to particle size distributions was made by Paul Rosin and Erich Rammler, with final contributions from Karl Sperling and John Bennet (Rosin and Rammler 1933). Their interest was basically the particle size distribution of coal. They defined, out of experimental data, the following probability density function $f(x)$:

$$f(x) = n\left(\frac{x}{x_c}\right)^{n-1} \exp\left[-\left(\frac{x}{x_c}\right)^n\right], \tag{2}$$

where n is the uniformity index, x is the mesh size, and x_c is the characteristic size. The cumulative probability function $F(x)$ is the following:

$$F(x) = P(X \leq x) = 1 - \exp\left[-\left(\frac{x}{x_c}\right)^n\right], \tag{3}$$

where X is size as random variable and P denotes probability. In 1939, Waloddi Weibull used this distribution to represent the random strength of materials (Weibull 1939). He published a definitive paper in 1951 (Weibull 1951) where he showed many examples of properties following the distribution, including some concerning sizes (not of rock fragments). Weibull took all the fame amongst statisticians and engineers, so that the distribution is usually called today after his name only. The Rosin–Rammler–Weibull (RRW) cumulative distribution can also be written using as scale parameter the median fragment size, x_{50} instead of x_c

$$F(x) = P(X \leq x) = 1 - \exp\left[-\ln 2\left(\frac{x}{x_{50}}\right)^n\right]. \tag{4}$$

Note that the common way of representing the cumulative size distributions of rock fragments in mining is log–log, while in civil engineering, it is semi-log (linear P and log x).

The first use of the Rosin–Rammler–Weibull for a size distribution of rock fragments from blasting was made by Baron and Sirotyuk (1967) and Koshelev et al. (1971); then, the first "fragmentation formula" came in the wake of that work by Kuznetsov (1973).

5.2 The Kuz–Ram Model

The origin of the Kuz–Ram model is the Kuznetsov formula (Kuznetsov 1973). The data used for deriving his formula were some 11 blasting tests by Koshelev on largely undescribed, probably one-hole blocks. The explosive used was RDX. The fact that Kuznetsov writes his equation for the "mean" size of fragments (and not the median, x_{50}) has brought some controversy. In fact, it is not clear from Kuznetsov's paper if the formula even applies for x_c, since he mentions that the mean $< x > x_c$ for the usual n values. The formula is as follows:

$$x = Aq^{-4/5}Q^{1/6}\theta^{-2/3}, \tag{5}$$

where $<x>$ (cm) is the mean fragment size, A is the rock strength factor, q is the specific charge (kg/m^3), Q is the explosive mass per hole (kg), and θ is the TNT equivalent of the explosive. From the parameters that influence the fragmentation process, A comes from the rock side, q and θ come from the explosive side, and Q is a scale factor. For the large-scale formula, Kuznetsov (1973) removed the TNT equivalent in Eq. 5.

Cunningham (1983) picked up the Kuznetsov formula. He wrote it

$$\bar{x} = Aq^{-0.8}Q^{1/6}\left[\frac{E}{115}\right]^{-19/30}, \tag{6}$$

where E is here energy of the explosive relative to ANFO, or "Relative Weight Strength" (RWS in %); the 115 factor is the RWS of TNT. The equation can be readily inverted to obtain the necessary specific charge if a certain mean fragmentation is required. Cunningham wrote $\bar{x}$ as the "mean" fragment size, but in fact treated it as the median size.

The second part of the Kuz–Ram model is a prediction formula for the RRW exponent n (Cunningham 1983). However, no reference is given on the data set from which this expression is derived

$$n = \left(2.2 - 14\frac{B}{d}\right)\left(1 - \frac{W}{B}\right)\left(1 + \frac{S/B - 1}{2}\right)\frac{L_c}{H}, \tag{7}$$

where B is the burden (m), d the hole diameter (mm), S/B the spacing-to-burden ratio, L_c the charge length (m), and H the bench height (m). W is described by Cunningham as 'the standard deviation of drilling accuracy' though, this concept not having a clear meaning, the average drilling deviation in the bottom of the hole (m) is commonly used. For staggered drilling, the calculated n-value should be increased by 10%. Cunningham (1987) incorporated Lilly's blastability index (Lilly 1986) as proportional to Kuz-Ram's rock factor A

$$A = 0.06(RMD + RDI + HF), \tag{8}$$

where RMD is a rock mass descriptive term linked with the joints spacing and orientation with respect to the face, RDI is a function of density, and HF is a function of the strength. In the course of time, the Kuz–Ram model underwent some refinements and modifications (Cunningham 1987, 2005).

The Kuznetsov formula in Eq. 6 is physically sound, since it means that, for example, the higher the specific charge, the smaller the fragments. However, the splitting of the dependency between rock/explosive data for the "mean" size, and geometrical data for the shape index is not supported by any theory. There is a considerable lack of written specific information on the source data from which the RRW exponent was obtained.

5.3 The SveDeFo Model

The SveDeFo (Langefors and Kihlström 1963; Holmberg 1974; Larsson 1974) model uses an RRW distribution, with constant shape factor $n = 1.35$ and median fragment size as follows:

$$x_{50} = 0.143\left(B^2\sqrt{\frac{1.25}{S/B}}\right)^{0.29}\left(\frac{c}{sq}\right)^{1.35}\left[1 + 4.67\left(1 - \frac{L_c}{L_h}\right)^{2.5}\right]. \tag{9}$$

SI units apply. s is the strength relative to dynamite LFB ($s_{\mathrm{ANFO}} = 0.84$), c the rock constant, L_c the charge length, and L_h the hole length. Some of these data were classified for different rock types by Sanchidrián et al. (2002).

The SveDeFo model is contemporary to the Kuznetsov's (1973) formula (Eq. 6), and does not assess n either, but suggests the tentative value $n = 1.35$. It incorporates a rock term (c), an explosive term (the product sq), a scale term (burden B), and some layout information: the ratio of spacing to burden (S/B) and the ratio of charged length to hole length (L_c/L_h).

5.4 The Kou–Rustan Model

Kou and Rustan (1993), based on extensive model blast tests in magnetic mortar and literature data, and earlier formulae by Rustan and Nie (1987), derived the following expression:

$$x_{50} = 0.01\frac{(\rho c_p)^{0.6}(S/B)^{0.5}B^{0.2}}{(L_c/H)^{0.7}D^{0.4}q}, \tag{10}$$

where ρ is the rock density, c_p the rock P-wave velocity, and D the velocity of detonation.

SveDeFo's and Kou and Rustan's x_{50} formulae are contradictory in the influence of S/B (probably a result of generalization from limited and inaccurate data). It incorporates rock terms (ρ, c_p), explosive terms (q, D), a scale term (B), and some layout information (S/B and L_c/H).

5.5 The Chung–Katsabanis Model

Chung and Katsabanis (2000), using data from Otterness et al. (1991), 29 medium-scale blasts in dolomite, proposed the following equations for x_{50} (percentile 50 or median) and x_{80} (percentile 80):

$$x_{50} = AQ^{-1.193}B^{2.461}(S/B)^{1.254}H^{1.266} \tag{11}$$

$$x_{80} = 3AQ^{-1.073}B^{2.43}(S/B)^{1.013}H^{1.111}. \tag{12}$$

Interesting about this model is the calculation of different sizes from the blast data (the n-formula is directly derivable from x_{50} and x_{80}). After substitution of $q = Q/BSH$

into Eqs. 11 and 12, it can be shown that x_{50} is essentially a function of the specific charge only (the other exponents are very small), and no scale term. x_{80} shows a similar functional dependence as x_{50} plus a scale term (B). Chung and Katsabanis' model adds little to the Kuznetsov model and was derived from relatively limited data.

5.6 The Crush Zone Model (CZM)

In the late 1990s, it was found that the RRW distribution was predicting far less fines than real fragmentation produces. The crush-zone model (CZM) was developed by Kanchibotla et al. (1999) and Thornton et al. (2001). The fragment size distribution has two parts: coarse and fines. The coarse part results from tensile breakage and it can be predicted with the Kuz–Ram model. The fines are assumed to originate in the crush zone surrounding the borehole exclusively, from compressive/shear failure. The radius of the crush zone is calculated from a plane (2D) static elastic stress field. The maximum size of the crush zone particles is arbitrarily assumed to be 1 mm. In the crush zone, the compressive stress is higher than the compressive strength of the rock, i.e., a cylinder of radius

$$r_c = R\left(\frac{P_h}{\sigma_c}\right)^{1/2} \tag{13}$$

P_h was quoted in the authors' original work as the detonation pressure. In the CZM, two expressions are used for the size distribution, grafted at a certain change point. For the coarse part, the Kuz–Ram model predictions for x_{50} and n stand. For the fines part, two cases are presented depending on the rock strength. For strong rock (σ_c > 50 MPa), the grafting point is x_{50}: the coarse section, $x \geq x_{50}$ is calculated with the Kuz–Ram model. For $x < x_{50}$, a second RRW function is used passing through (x_{50}, $P_{50} = 0.5$) and with a fraction passing at 1 mm equal to F_c

$$P(x) = 1 - e^{-\ln 2\left(\frac{x}{x_{50}}\right)^{n}}, x \geq x_{50} \tag{14a}$$

$$P(x) = 1 - e^{-\ln 2\left(\frac{x}{x_{50}}\right)^{n_{fines}}}, x < x_{50} \tag{14b}$$

where the fines uniformity index, n_{fines}, is calculated by

$$n_{fines} = \frac{\ln\left[-\ln\left(1 - F_c\right)/\ln 2\right]}{\ln\left(0.001/x_{50}\right)} \tag{15}$$

Note that sizes must all have the same units so x_{50} in Eq. 15 must be written in m.

For weak rock ($\sigma_c < 10$ MPa) and intermediate rock ($10 < \sigma_c < 50$ MPa), there are formulas similar to those for strong rock (see details in Kanchibotla et al. 1999; Thornton et al. 2001). The crush zone model is largely arbitrary. In it, different fragmentation modes are represented by the same distribution (though with different parameters). There is evidence (Svahn 2002; Reichholf 2003) that fines are generated also outside the crush zone; this is further supported by recent numerical and experimental studies on blasted circular cylinders (Iravani 2020; Kukolj 2021). The first calculation of the extent of the crush zone is overly simplistic, and the effects of loading rate on rock strength and rock fracture toughness are not considered at all. Esen et al. (2003) and Onederra et al. (2004) provide improved formulae. The crush-zone model is one of the more commonly used fragmentation prediction models.

5.7 The Two-Component Model

This model was developed by Djordjevic (1999), as follows:

$$\begin{aligned} P(x) =& F_c\left\{1 - \exp\left[-\ln 2\left(\frac{x}{x_{50f}}\right)^{n_f}\right]\right\} \\ &+ \left(1 - F_c\right)\left\{1 - \exp\left[-\ln 2\left(\frac{x}{x_{50c}}\right)^{n_c}\right]\right\} \end{aligned} \tag{16}$$

F_c is the fraction of rock that fails under shear-compression (the 'crush zone') and $1 - F_c$ is the fraction of rock that fails under tension. Subscripts f and c apply to the fine and coarse components, respectively. The parameters of the function are obtained from blasting experiments on pieces of rock in a closed chamber. To the resulting fragments' size distribution, a bimodal RRW (Eq. 16) is fitted. The x_{50f} and x_{50c} values obtained in the fit are used to back-calculate rock factors for the coarse and fine fractions, from the Kuz–Ram (i.e., the Kuznetsov) equation. The fines exponent from the specimen blast is used for the large-scale blasts distribution, whereas Djordjevic uses the Kuz–Ram exponent for the coarse fraction (quite surprisingly as the experimental one could have been used).

The two-component model uses a more powerful function: a bimodal, with 5 parameters, that can represent size distributions quite efficiently in a broader range of sizes. The problem is to feed the 5 parameters to the model, for which experimental data must be used. Although this increases the quality of the model, it is difficult to use the model on an a priori basis. In a way, it has some similarity with the crush zone model that also uses two RRW functions for the coarse and the fine zones of the distribution.

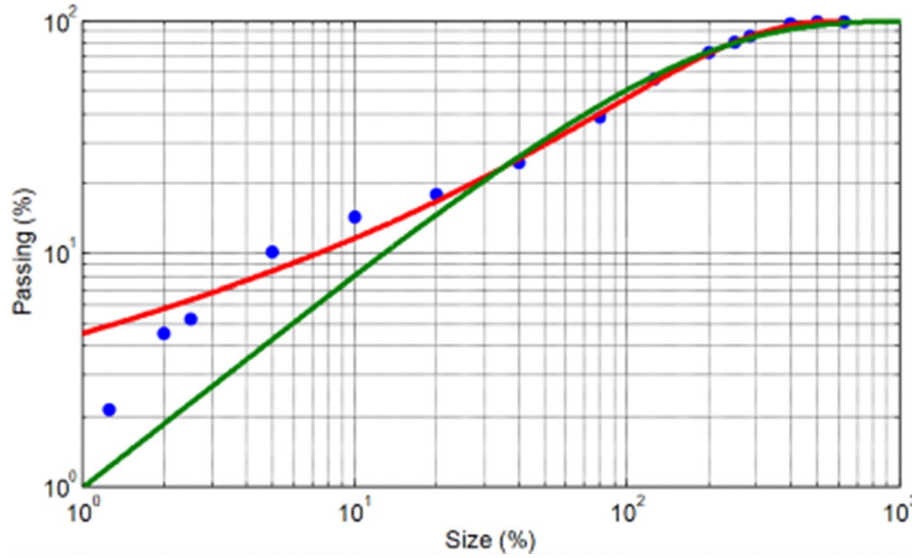


Fig. 10 The Swebrec (red line) and RRW (green) functions

5.8 The KCO Model

This model named after Kuznetsov–Cunningham–Ouchterlony was developed by Ouchterlony (2005b) in an adaptation of the Kuz–Ram model to make use of the Swebrec distribution (Ouchterlony 2003b, 2005a). This is a three-parameter function, x_{50}, $x_{\max}$ (the maximum size) and b, a shape parameter

$$P(x) = \frac{1}{1 + \left[\frac{\ln(x_{max}/x)}{\ln(x_{max}/x_{50})}\right]^b}. \tag{17}$$

Note that the Swebrec is not an infinite function as it is restricted to $x \leq x_{max}$. It typically bends up in the fines (Fig. 10). x_{50} is calculated with Kuz-Ram's median size formula and $x_{\max}$ is estimated as

$$x_{max} = \min\left(B, S, l_s, x_{IBSD}, \ldots\right), \tag{18}$$

where l_s is stemming length and x_{IBSD} median size of the in-situ block size distribution. Usually in blasting, $x_{\max}/x_{50}$ lies between 5 and 30. The parameter b may be estimated by the following formulae derived from relations of the parameters of actual fragment size distributions:

$$b \cong 0.5x_{50}^{0.25} \ln\left(x_{max}/x_{50}\right) \tag{19a}$$

$$b = 0.4 \ominus \left(\frac{B_{ref}}{B}\right)^{0.25} \ominus x_{50}^{0.25} \ominus \ln\left(x_{max}/x_{50}\right) \tag{19b}$$

$$b \cong 2 \cdot \left[\ln\left(x_{max}/x_{50}\right)\right]^{0.39} \tag{19c}$$

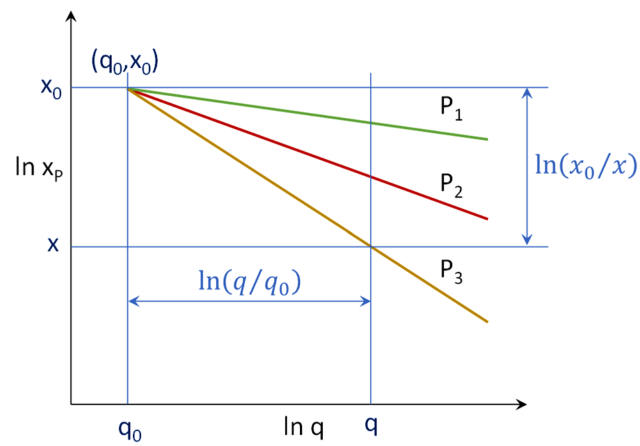


Fig. 11 Principle sketch of the fragmentation-energy fan

x_{50} must be in mm in the term $x_{50}^{0.25}$ of Eqs. 19a and 19b; B_{ref} = 4 m; see Fig. 10

The most relevant characteristic of the KCO model is that it uses the Swebrec function instead of the RRW. The Swebrec usually represents rock fragments much better than the RRW, especially in the fines. The KCO model is simple in its application as it does not use any arguable and complicated formula for the shape factor as the Kuz–Ram does. It requires an estimation of $x_{\max}$ though, and the estimation method suggested is, albeit reasonable, somewhat crude. Despite its obvious benefits, this model has not been used much.

5.9 The x_P-Frag Model

Ouchterlony (2009) made a blasting-related interpretation of the dimensional analysis by Holsapple and coworkers on asteroid collisions (Holsapple and Schmidt 1987; Housen and Holsapple 1990). Ouchterlony arrived at the following expression for the fraction P of fragments of mass less than m:

$$P = F_1\left(\frac{m}{M}, \Pi_s, \Pi_g\right), \tag{20}$$

where M is the total nominal mass fractured; Π_s and Π_g are strength- and gravity-related non-dimensional parameters. F_1 is an unspecified functional dependence. Based on this formulation, Sanchidrián and Ouchterlony (2017) developed a fragmentation prediction formula in the form

$$\frac{x_P}{L} = kk_2^h\left(\frac{\overline{\sigma}}{qe}\right)^{\kappa}\frac{1}{L^{\lambda\kappa}}, \tag{21}$$

where x_P is the size at the percentage passing P, $\tilde{\sigma}$ is the strength (with dimensions of stress) of the rock mass, e is the explosive energy per unit mass, and q is the specific charge. L is a characteristic length, related with the blast size. This model requires the calibration of the four functions $k(P)$, $h(P)$, $\kappa(P)$, and $\lambda(P)$ from experimental data. Some influential characteristics that are relevant to rock blasting, i.e., rock mass discontinuities, joint spacing, and delay time, were incorporated to the basic expression Eq. 21 in the form of multiplier factors; see Sanchidrián and Ouchterlony (2017) for details.

5.9.1 The Fragmentation-Energy Fan

This model was developed by Ouchterlony and coworkers (Ouchterlony et al. 2017, 2021; Ouchterlony and Sanchidrián 2018; Segarra et al. 2018; Sanchidrián et al. 2022). Sieving data obtained from tests with different values of specific charge q may be written

$$x_P/x_0 = \left(q_0/q\right)^{\alpha}, \tag{22}$$

where x_P is the size at the percentage passing P. The exponent $\alpha = \alpha(P)$ or α_P is, for a given blast geometry in a given material, a function of P only. Many of the fragmentation formulae reviewed in the preceding sections have in a way the form of Eq. 22, inasmuch as they make x_{50} (or other percentiles) a power law of the specific charge. Equation 22 generalizes this for any percentile. The function $x_P = f(q)$ in Eq. 22 is represented in log–log by a set of straight lines converging in the point (q_0, x_0), see Fig. 11.

The focal point usually lies outside the q- and x- intervals covered by the data, so a physical interpretation of the values q_0 and x_0 is not immediate. With (q_0, x_0) approaching infinity, the case of parallel x_P-lines is also covered.

After some manipulation of Eq. 22, it follows that the percentage passing P must be a function of the argument:

$$\frac{\ln\left(x_{max}/x\right)}{\ln\left(x_{max}/x_{50}\right)}. \tag{23}$$

This is the same logarithm ratio that one finds in the Swebrec distribution, see Eq. 17, and it can be further shown that a Swebrec-type function can be used for the $P(x, q)$ dependence

$$P(x) = \frac{100}{1 + \left[\frac{\frac{\ln(x_0/x)}{\ln(q/q_0)} - \alpha_{100}}{\alpha_{50} - \alpha_{100}}\right]^b}. \tag{24}$$

The constants x_0, q_0, α_{50} and α_{100} are directly obtained from the construction of the fan; the exponent b can be determined from three slopes, for example

$$b = \ln(4)/\ln\left[\frac{\ln\left(x_{50}/x_{20}\right)}{\ln\left(x_{80}/x_{50}\right)}\right] = \ln(4)/\ln\left(\frac{\alpha_{20} - \alpha_{50}}{\alpha_{50} - \alpha_{80}}\right). \tag{25}$$

Note that Eq. 24 can determine any size distribution at a given specific charge with only five constants. As of today, there are no formulae to predict the fan parameters, so the use of the fragmentation-energy fan model requires some experimental data. It offers, however, an excellent analytic frame for the analysis of fragmentation data as function of specific charge or, in general, energy input to the rock.

5.9.2 A Perspective on Rock Fragmentation Models

Many prediction formulae are available for fragmentation by blasting. Some estimate the median size, others attempt to obtain the whole size distribution. Many of the existing models assume a Rosin–Rammler–Weibull distribution for the rock fragments size. Generally, fragmentation models are derived from limited experimental data, at most combined with some simple Physics. Probably, the Kuz–Ram and its variety of the Crush-zone are the more widely used models for fragmentation by blasting.

Lilly's blastability index (Lilly 1986, 1992), or derivations from it, appears to include most of the rock mass features related with fragmentation. The explosive action is always modeled by the specific charge; the explosive specific energy is often considered as a correcting factor for more or less energetic explosives. Other properties such as density, VOD, etc. are seldom used. Scale factors are the explosive mass per hole, the hole diameter, the burden or spacing or combinations thereof. The initiation sequence is considered in the Kuz-Ram's 2005 update and the x_P-frag. Apart from these, long-standing ‘optimum delay’ figures of some ms per meter burden or spacing prevail.

Generally, model predictions are not accurate due to the following reasons: (1) The formulae have wide prediction bands due to data dispersion. (2) Fragmentation is strongly site dependent. (3) Too many rock mass variables are accounted for by only one parameter. (4) The explosive energy delivery rating is far from a standard calculation or measurement. Nevertheless, some of the trends that the models suggest can be useful and may guide the analysis of blast results and the tailoring of formulae for a given operation. Models are engineering tools. Some hints on their use are: (1) Always use more than one model and discuss the different results. (2) Always represent size distributions with the size axis in log scale (preferably both axes in log scale). (3) Bear in mind that size distribution functions have a limited range of validity; the RRW cannot usually be trusted below 10% passing, the Swebrec below 5%, bi-component distributions may function well down to about 1% (Sanchidrián 2015). (4) Do not extrapolate size distributions fitted to experimental data outside the range of the data. If your interest lies much on the fines then be aware that your calculations may have large errors; try to obtain experimental fragmentation information.

6 Optimum Fragmentation

6.1 Definition

Optimum rock fragmentation from mining to mineral processing must meet the following conditions (Zhang 2016a): (1) minimum cost in the size reduction chain: drilling–blasting–crushing–grinding, (2) maximum ore recovery ratio, (3) high productivity, and (4) minimum negative impact on safety and environment. The fragmentation which satisfies only three of the above conditions or fewer is not an optimum fragmentation. To achieve maximum ore recovery in mining, all the ore mass included in the mining plan should be completely blasted into the required sizes. Otherwise, more ore boulders may be produced, resulting in ore loss if such boulders are too large to be handled and loaded. To achieve high productivity, the sizes of fragments by blasting should be sufficiently small, so that loading can be carried out efficiently. To reach a minimum cost from drilling to grinding, blasting must be successful first, and the distribution of the energy consumption among the different operations should be optimized. In addition, the energy efficiency of each operation should be high enough. Since most of the energy is spent in grinding and that is the operation with a lower energetic efficiency, the energy efficiency of grinding must be increased, no matter if another operation (e.g., blasting) reduces its efficiency. To get minimum effect on safety and environment, the blast design must prevent high vibrations from blasting, fragment throw, and misfires and explosives leakage must be avoided or reduced.

6.2 Possibility of Optimum Fragmentation

To achieve optimum fragmentation, all of four conditions mentioned above must be met. However, optimum fragmentation mainly depends on whether a minimum cost from drilling to grinding can be achieved or not (Zhang 2016a). Furthermore, to achieve the minimum cost the total energy expenditure must be reduced from drilling to grinding without degrading fragmentation and the energy efficiency of at least some of the operations must be increased.

6.2.1 Disparity Between Energy Efficiencies in Drilling, Blasting, Crushing, and Grinding

In rock drilling, blasting, crushing, and grinding, the effective energy used in rock breakage is found to be quite small in comparison with the total energy input. As mentioned in Sect. 1 and the review by Zhang and Ouchterlony (2022), the energy efficiency, i.e., the ratio of the energy used in fracturing rock to the energy input, was only 10%, 6%, 4%, and 1% in rock drilling, blasting, crushing, and grinding, respectively. In short, the disparities between energy efficiencies of drilling, blasting, crushing, and grinding are large. Because of the large disparities between the energy efficiencies, a change in energy distribution between the different operations can be made to reduce the total energy expenditure.

6.2.2 Blast-Induced Microcracks

It was found that blasting created micro-cracks in the fragments (e.g., Jaeger et al. 1986; Nielsen and Kristiansen 1996). In rock fracture experiments under dynamic loads, branching cracks, in either macro-scale or micro-scale, were induced and most of them ended within the fragments as loading rate or impact speed was increased (Zhang et al. 2000).

McCarter and Kim (1993) found that after dynamic loading of quartz monzonite, diopside, wollastonite, and subarkosic siltstone specimens in a split Hopkinson pressure bar test set-up, their P-wave velocities were reduced on average by 31%, 29%, and 18%, respectively. Katsabanis et al. (2003) examined the damage development in small granodiorite blocks by measuring the P-wave velocity of each specimen before and after blasting it. Their results indicated that the average damage due to blasting was increased by 174%, compared with the initial damage in the specimens before blasting. Roblee and Stokoe (1989) measured a 10% reduction in P-wave velocity in sedimentary rock following blasting.

Kemeny et al. (2003) found that the uniaxial compressive strength, tensile strength, and point load index of the rocks were decreased by 10–40% due to blasting.

The P-wave velocity reduction, damage growth, and strength decrease mentioned above imply that more micro-cracks have been created within the specimens after blasting or impact loading. Such micro-cracks produced by blasting would be more or less favourable to the separation of different minerals along their boundaries in crushing and grinding.

6.2.3 Redistribution of Energy Input

The idea of redistribution of energy input from mining to mineral processing can be traced back many years. For instance, McCarter and Kim (1993) argued that it might be more advantageous to use chemical energy in explosives rather than electrical energy in comminution. Later, the redistribution of energy input was mathematically described by Zhang (2008; 2016a). Briefly, to make the redistribution of energy input in the whole size reduction system, the energy input in drilling and blasting will be increased by a specific amount Δ. Assume that the energy input in crushing and grinding can be reduced by the same amount Δ in the unit of J, and then, an additional amount of energy $(\eta_{db} - \eta_{cg})\Delta$ can be gained and utilized in the whole size reduction system after the redistribution of energy input, even though the total energy input has not changed. Here, η_{db} and η_{cg} are the energy efficiency in drilling and blasting and crushing and grinding in the unit of %, respectively. In other words, if the energy Δ is moved from crushing and grinding to drilling and blasting, an additional energy $(\eta_{db} - \eta_{cg})\Delta$ can be available in the whole size reduction system, for more comminution work. In practice, this can be realized, for example, by increasing the specific charge. How much additional energy input is needed in the blasting depends on the existing fragmentation level.

In brief, it is possible to realize optimum rock fragmentation by considering the blast-induced micro-cracks (or damage) inside rock fragments and the redistribution of energy input to the whole size reduction system from drilling–blasting to crushing–grinding.

7 Measures for Achieving Optimum Fragmentation

7.1 Increase of Specific Charge in Blasting

Section 6.2 indicates that it is possible to achieve optimum fragmentation by increasing energy input to blasting, for instance, by increasing specific charge in rock blasting. Section 2.3 presents several successful examples in which mill throughput has been increased using a higher specific charge. Two further examples are given. In the Aitik open pit copper mine, a specific charge increase from 0.9 to 1.3 kg/m^3 gave rise to an increase in the throughput by nearly 7% due to more fine materials produced and shorter grinding time achieved (Ouchterlony et al. 2013). Similarly, an increase in specific charge from 0.8 to 1.5 kg/m^3 resulted in an increase of 7% particles less than 25 mm in the SAG mill feed in Andina copper mine (Brent et al. 2013). All the examples using higher specific charge presented in this article are successful examples of the application of a higher specific charge in mining production blasting.

Notice that specific charge is not a parameter good enough to represent the actual stress and energy distribution in the rock to be fragmented, since the actual stress and energy distribution is far from even. This means that a constant specific charge may result in different fragmentation results if different explosives are used or if other parameters such as blasthole diameter, detonator placement, burden, spacing, stemming, etc. are different for the same rock. In addition, attention must be paid to possible misfires or malfunctions in detonators and explosives in multi-hole blasting, and inadvertent drillhole deviations. In brief, a higher specific charge may not always result in better fragmentation if the blast design is unreasonable, and misfires occur (Zhang 2016a).

A big challenge in employing a high specific charge is to find out a correct or suitable specific charge for optimum fragmentation in a specific blast. When a very high specific charge is used, the amount of fines increases and, if the holes spacing is reduced too much, the misfire rate may increase. In addition, blast-induced damage from flying rocks, etc. is another important issue to consider. The extra-fine mineral particles are difficult to recover by modern mineral processing technology (Wills and Napier-Munn 2006), i.e., they will become permanent ore loss.

7.2 Increase of Energy Efficiency in Blasting

In addition to directly increasing energy input, as described in Sect. 7.1, the energy used to fragment rock can be increased by increasing energy efficiency in blasting, without an increase in the specific charge of a blast. In practical blasts, this can be done by different ways such as reduction of gas ejection from blastholes and enhancement of stress distribution in the rock to be blasted.

To reduce high-pressure gas ejection from blastholes, correct stemming must be used. The correct stemming includes correct length and suitable material of the stemming, since both influence shock wave propagation in a blast hole (Zhang 2016a). How to determine the correct stemming parameters such as length is still an issue to study (e.g., Oates and Spiteri 2021).

To enhance the stress distribution in the rock, the following measures are recommended. (1) The best detonator position(s) should be employed in each blasthole. If only one detonator is placed in a blasthole, the best position is the midpoint of the explosive column length, but in the case that two detonators are placed in each hole, the best detonator positions should be based on the stress distribution predicted by numerical modeling. Some successful examples have been reported by Zhang (2005a, b) for underground mining and by Ylitalo et al. (2021) for open pit mining. (2) Shock wave collision can be tried by placing two detonators at two different positions in each hole. A successful example from underground mining is reported by Zhang (2014). (3) The air deck technique can be employed. (4) An optimum delay time between two adjacent holes should be used.

Shock wave collision can be realized by firing two detonators at different positions in a blasthole simultaneously. According to shock collision theory, as two identical shock waves meet each other, the final peak pressure will be locally greater than the sum of the two initial shock pressures (Cooper 1996; Zhang 2016a). Shock wave collision has been successfully used to reduce brow damage (Zhang 2014), to bring down hanging roofs (Zhang 2016b), and to improve fragmentation in sublevel caving (Zhang 2014; Zhang and Wimmer 2018). Since applications of shock wave collision are still few, more studies on this subject are needed.

As described in Sect. 3.10, delay time influences rock fragmentation. However, there are different experiences on the effect of short delay time on fragmentation. In addition to the studies cited in Sect. 3.10, analytical work on short delay blasting by Yi et al. (2016) and numerical simulation by Yi et al. (2017) concluded that the improvement of fragmentation by stress wave superposition was impossible. This analytic conclusion based on an idealized process is interesting but requires more full-scale blasts to verify. Bear in mind that rock fragmentation is dependent not only on stress magnitude and the stress distribution in the rock but also on crack propagation and fragment movement during blasting (Zhang 2016a). In brief, what is a correct delay time in multi-hole blasting and how to determine it is still a tough challenge in rock blasting.

7.3 Increase of Energy Efficiency in Crushing and Grinding

A recent review (Zhang and Ouchterlony 2022) confirms that only a very small fraction of energy input is used to create new rock surface area in the mills and single particle impact crushing, resulting in an extremely low-energy efficiency of about 1% in such conventional milling. For example, measurement results have demonstrated that about 75–90% of the energy input is finally degraded into thermal energy; see Sect. 2.1.

To increase the efficiencies of crushing and grinding, new crushers and mills should be developed. In this direction, high-pressure grinding roller (HPGR) mills, developed in the 1980s (e.g., Schönert 1979, 1988), represent a relatively new technique in mineral processing. The HPGR mills yield higher energy efficiency than ball mills, probably because the particles are confined to a certain extent, which can reduce both the energy used in the friction between particles and the kinetic energy of particle movement occurring in ball mills. Nevertheless, the development of new crushers and mills with higher energy efficiency could still be expected.

8 Challenges in Achieving Optimum Fragmentation

8.1 Stress Waves and Gases in Blasting

One challenge in the realization of optimum fragmentation is to fully understand the role of stress waves and gases in blasting, which has not been very clear so far. In the earliest studies, there were two viewpoints on this issue. One viewpoint considered that stress or shock waves played a predominant role in rock fragmentation (e.g., Hino 1959; Duvall and Atchison 1957), and the other thought that high-pressure gas played a dominant part in rock fragmentation (e.g., Langefors and Kihlström 1963; Clark and Saluja 1964). Since the 1970s, a combined viewpoint that has found support is that it is the combined effect of both stress wave and gas pressure that determines the rock fragmentation (e.g., Kutter and Fairhurst 1971; Field and Ladegaard-Pedersen 1971; Bhandari 1979; Dally et al. 1975; Fourney et al. 1993; Fourney 2015).

8.2 Blast Design

Another challenge in optimum fragmentation is that most blast parameters, such as burden, spacing, stemming (length and material), subdrilling, and delay time, have still been determined mainly by empirical methods rather than by physical and numerical models. A correct detonator or primer position can be determined by stress wave analysis, which has been proved by both underground and open pit blasts, as described in Sect. 3. However, the fact that rock fragmentation is about fracture mechanics, rather than continuum mechanics incorporates a severe difficulty in the process as compared with other engineering disciplines (e.g., mechanical and structural engineering, fluid mechanics, etc., which have their own difficulties). In addition, the description of the material medium, the rock mass, is far from being well established as discontinuities play a role in it. The strength of the rock under highly transient loading is one more feature that complicates the modeling so that reliable engineering models of the explosive/rock interaction directly based on physical principles are still needed.

8.3 Measurement of fragmentation

So far, it has still been a big challenge to measure the sizes and fracture surface areas of all fragments including fines produced by blasting in both model blasts and full-scale production blasts where image analysis is available, according to previous studies (e.g., Sanchidrián et al. 2006; 2009). Although digital image-based methods for determining the fragments size distribution are widely used in rock blasting, they have several drawbacks (Ouchterlony 2003b). For example, they define size differently than sieving (so far study only the surface) and they have not been calibrated, neither absolutely nor relatively to each other. They cannot be used as scientific instruments but offer advantages in field testing and production monitoring.

Fragmentation, except in some special cases, can only be measured by sieving if an absolute statement on sizes and percentage passing values is required. However, due to the complication of carrying out such an operation on the muckpile of a blast, image analysis systems are often employed. In this case, consideration should be given to the difficulties of measuring fragments from images, invariably leading to segmentation and sizing errors (see e.g., Koh et al. 2009; Rosato et al. 2002; Potts and Ouchterlony 2005; Wang 2008; Thurley 2011; Thurley and Ng 2008; Andersson and Thurley 2008). Added to these sizing errors is the fact that image analysis in any of its forms tries to determine the fragment size distribution of a pile of fragments (a three-dimensional structure) from measurements on the surface (a two-dimensional one). Stereological unfolding solutions, where random sampling in two-dimensional sections is used to derive quantitative information about a three-dimensional material based on statistics and geometrical principles, lead to generally unsatisfactory results in the case of muckpile imaging, as the section used when measuring rock fragmentation is the surface, not a random one, with a fragment size distribution intrinsically biased by the missing fines and the total or partial overlapping of particles in the second or third layer into the pile by particles in the first layer (Maerz 1996). Unfolding solutions heavily rely on calibration, which requires some a priori knowledge of the actual fragment size distribution that can only be achieved by sieving. Ultimately, proper sieving measures the 'waists' of a stream of oriented particles, but the particles in an image more likely have a different orientation, while image analysis does not have information on the depth. The two methods do not measure the same thing.

The above is all too often overlooked by image analysis studies, that most often lack a realistic error analysis or a statement on ranges of validity of the measurements, and that requires an estimation (often out of reach) of how much material is not present in the delineated areas or beneath the working images. Still, image analysis may help to roughly detect large changes of fragmentation and that is often all that is needed to flag out rock changes, or drill and blast problems, e.g., drilling errors or poor explosives functioning. However, when fragment sizes are sought in an absolute, quantitative or predictive fashion, image analysis alone cannot provide a solution. Combinations of image analysis with on-site sieving, or derivation of rock cuts weights from crushing plant mass flows, have been used with advantage (Cho et al. 2003; Ouchterlony et al. 2006, 2010; Segarra et al. 2018).

9 Concluding Remarks

1. Rock drilling, blasting, crushing, and grinding consume a vast amount of energy in hard rock mines, but most of the energy input to these operations is used in grinding which has the lowest energy efficiency, approximately 1%, compared with the other three operations, and about 75–90% of the energy input in grinding is wasted in heating the materials.
2. Better (finer) fragmentation by blasting has saved a substantial amount of energy in downstream operations, such as crushing and grinding, since fragment sizes have been largely reduced via blasting, according to production blast data in some mines. As a result, some mines have gained cost savings due to better fragmentation from mine to mill projects.
3. Ore recovery ratio and productivity (extraction rate and mill throughput) have been increased, and the total cost in mining and mineral processing reduced due to better fragmentation in several mines. Since ore recovery can be increased by better fragmentation, the sustainability in mining and mineral processing may be improved in many cases.
4. The main factors influencing rock fragmentation are represented by three groups of parameters: (i) explosive (including its density, VOD, and explosion

energy) and initiator (detonator type and initiation precision), (ii) rock (geo-structures, density and sonic velocity or impedance, strength and fracture energy, in-situ stresses including confining pressure, and water), and (iii) energy distribution and energy efficiency (drill plan, stemming, subdrilling, detonator position, decoupling ratio, air deck, delay time, specific charge, and free surface).

5. Optimum fragmentation is feasible for several reasons: (a) blasting induces micro-cracks within ore fragments and thereby weakening them; (b) a large disparity in energy efficiency exists among different operations, such as drilling, blasting, crushing and grinding where the energy efficiency of grinding is the lowest; (c) more energy could be input to blasting, for example, by increasing the specific charge.
6. It is important to predict rock fragmentation in mining production blasting. Correspondingly, several models have been developed. Generally, present model predictions are not accurate for several reasons. Thus, it is better to use more than one model and discuss the different results. In addition, bear in mind that size distribution functions have a limited range of validity and it is always very valuable to try to obtain fragmentation information from field and lab tests.
7. Several mines report that a high specific charge has resulted in higher productivity such as mill throughput. However, a higher specific charge does not necessarily result in higher productivity or/and finer fragmentation if misfires occur. Therefore, when a blasting plan is made, the possibility of misfires must be considered, especially when use of a high specific charge is planned.
8. Fines originate not only in the crushing zone surrounding the blastholes but also in the fractured zone due to crack branching. Either a larger blasthole or a higher specific charge may cause more fines. If these fines consist of ores and they are too small to be recovered by current mineral processing technology, a too large blasthole or a too high specific charge should not be used.
9. It is still a big challenge to measure the sizes and fracture surface areas of all fragments including fines produced by blasting in both model blasts and full-scale production blasts. On-site sieving is time- and labour-consuming in production blasts and present image analysis alone cannot provide a solution when fragment sizes are sought in an absolute, quantitative or predictive fashion.
10. The main measures for achieving an optimum fragmentation by blasting are to increase the specific charge in blasting while avoiding misfires and to improve the energy efficiencies in blasting, crushing, and grinding.

Acknowledgements This research was supported by University of Oulu in Finland, Universidad Politécnica de Madrid in Spain, and Montanuniversitaet Leoben in Austria.

Funding Open Access funding provided by University of Oulu including Oulu University Hospital.

Declarations

Conflict of Interest The authors declare that they have no conflict of interests.

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