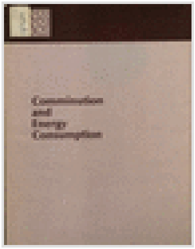

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Comminution is a single or multistage process by which mineral ores and other materials are reduced by crushing and grinding to those sizes required for beneficiation, in which the severed components are separated into concentrate and tailing products. On the order of a billion tons of rock ore alone is crushed and ground every year in the United States. Up to 99% of the energy consumed in grinding these ores may be expended in the movement of machinery, with noise and heat the undesirable by-products, leaving only 1% of the applied energy for size reduction. In addition there is evidence that many of today's crushing and grinding techniques not only are inefficient and antiquated but have little or no scientific base of understanding.

This report discusses the current state of the art in various topic areas of comminution, and presents a series of recommendations related to each topic. Areas for improvement are divided into six groups: (1) comminution device design; (2) classifier device design; (3) control; (4) grinding additions; (5) materials; (6) education.

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COMMINUTION
AND
ENERGY CONSUMPTION

Report of the

Committee on Comminution
and Energy Consumption

NATIONAL MATERIALS ADVISORY BOARD
Commission on Sociotechnical Systems
National Research Council

Publication NMAB-364
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1981

NOTICE: The project that is the subject of this report was approved by the Governing Board of the National Research Council, whose members are drawn from the Councils of the National Academy of Sciences, the National Academy of Engineering, and the Institute of Medicine. The members of the committee responsible for the report were chosen for their special competence and with regard for appropriate balance.

The report has been reviewed by a group other than the authors according to procedures approved by a Report Review Committee consisting of members of the National Academy of Sciences, the National Academy of Engineering, and the Institute of Medicine.

The National Research Council was established by the National Academy of Sciences in 1916 to associate the broad community of science and technology with the Academy's purposes of furthering knowledge and of advising the federal government. The Council operates in accordance with general policies determined by the Academy under the authority of its congressional charter of 1863, which established the Academy as a private, nonprofit, self-governing membership corporation. The Council has become the principal operating agency of both the National Academy of Sciences and the National Academy of Engineering in the conduct of their services to the government, the public, and the scientific and engineering communities. It is administered jointly by both Academies and the Institute of Medicine. The National Academy of Engineering and the Institute of Medicine were established in 1964 and 1970, respectively, under the charter of the National Academy of Sciences.

The Commission on Sociotechnical Systems is one of the major components of the National Research Council and has general responsibility for and cognizance over those program areas concerned with physical, technological, and industrial systems that are or may be deployed in the public or private sector to serve societal needs.

The National Materials Advisory Board is a unit of the Commission on Sociotechnical Systems of the National Research Council. Organized in 1951 as the Metallurgical Advisory Board, through a series of changes and expansion of scope, it became the National Materials Advisory Board in 1969. Its general purpose is the advancement of materials science and engineering in the national interest. It fulfills that purpose by providing advice and assistance to government agencies and private organizations on matters of materials science and technology affecting the national interest, by focusing attention on the materials aspects of national problems and opportunities, and by making appropriate recommendations for the solution of such problems and the exploitation of the opportunities.

This study was prepared under the joint sponsorship of the U.S. Bureau of Mines (Contract No. J0188128) and the National Science Foundation (Grant No. ENG-7910532).

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DEDICATION

The National Materials Advisory Board Committee on Comminution and Energy Consumption dedicates its report to a man whose interest in and service to the science of comminution will be sorely missed. Bert Bergstrom, who passed away at the age of 51, was an active member of the committee, and his countless contributions were invaluable in the preparation of this report. The members regret that he is not here to share in the product of this effort.

ABSTRACT

Comminution may be defined as a single or multistage process by which mineral ores and other materials are reduced from various sizes (e.g., run of the mine) by crushing and grinding to those sizes required for the beneficiation process, in which the severed components are separated into concentrate and tailing products.

Comminution is a huge consumer of energy, and is an appropriate target for significant savings. On the order of a billion tons of rock ore alone is crushed and ground every year in the United States. Up to 99% of the energy consumed in grinding these ores may be expended in the movement of machinery, with noise and heat the undesirable by-products, thus leaving only 1% of the applied energy available for size reduction. Coupled with this is the evidence that many of today's crushing and grinding techniques not only are inefficient and antiquated, but have little or no scientific base of understanding.

In this report the current state of the art in various topic areas of comminution has been discussed, and a series of recommendations related to each topic has been presented. The areas for improvement are divided into six groups: (1) Comminution Device Design; (2) Classifier Device Design; (3) Control; (4) Grinding Additions; (5) Materials; and (6) Education and Training. Within each of these six groups a set of project areas is identified involving specific efforts that are required to achieve or that will contribute to the achievement of the overall objective.

The fundamentals underlying various aspects of comminution (e.g., Fragmentation Physics, Transport, and Modeling and Simulation) are discussed at length. The committee feels that studies in the fundamental areas are extremely important and may in the long run be of predominant importance in preparing the way for future improvements.

PREFACE

Comminution is a process whereby particulate materials are reduced from various feed sizes by crushing and grinding to the product sizes required for downstream processing or end use. Comminution operations are found in a variety of process industries. In many industries --e.g., cement, coal (utilities), pulp and paper, agricultural products, fertilizer, pharmaceuticals, and paint and pigment--comminution steps are used to provide materials in the proper size range to give the final product the required properties. In other cases--e.g., minerals and coal--size reduction of raw material is required to ensure that valuable constituents are physically liberated from waste constituents before physical or chemical separations are attempted.

Current comminution technology is both energy-intensive and inefficient. U.S. industries use approximately 29 billion kWh of electrical energy per annum for size reduction and an additional 3.7 billion kWh per annum in contained energy in consumables such as grinding media and liners. This amount of energy approaches 2% of our total electric power production nationally. From an industrial point of view, size-reduction devices are costly to operate (frequently 25% or more of total raw materials processing costs) and are notoriously inefficient (less than 1% of the energy input reports as new surface area in the product). These facts should provide the incentive for industry to conduct research to improve the efficiency of comminution operations. From a national point of view, such research is justified because the tonnages involved in these operations are large enough so that even small improvements in the efficiency of comminution operations would provide considerable savings of our energy and mineral resources. Regrettably, the level of research in the United States designed to achieve these improvements has not been commensurate with the potential returns. In fact a recent survey has shown that at present there are 16 active, federally funded research projects in the area of comminution having a total dollar commitment of only $1.7 million. This situation exists in part because comminution science and technology is not a recognized entity within the funding agencies.

Current pressures for improving the efficiency of energy-intensive operations such as comminution, combined with anticipated demands for finer products in the minerals and coal industries, suggest that expanded research is a must. Energy conservation (minimization of the energy required [kWh/ton] to produce a given product size) will undoubtedly be the single most important goal of this research.

To identify those research areas that promise the greatest return in energy conservation, the U.S. Department of Energy, the U.S. Bureau of Mines, and the National Science Foundation, in a jointly funded endeavor, asked that the National Academy of Sciences-National Research Council convene a committee to study the problem. It was thought that such a study would be especially helpful in organizing the available information in the field, in assessing its importance to physics and materials research especially, and in providing perspective for future investigation.

In pursuing this assignment, which was accepted in 1978, the National Materials Advisory Board (NMAB), a unit of the National Research Council, was asked to "assist in the identification of practical ways to improve the energy-intensive comminution processes for coal, ores, cement, and other materials, both in the short- and long-term, by reduction of the energy consumption of the current processes, extension of the useful life of existing equipment, and development and application of new comminution methods. Both U.S. and foreign technical efforts and accomplishments [would] be reviewed in the study...."

To meet this request, the NMAB convened the Committee on Comminution and Energy Consumption. This committee was asked to document in its final report promising areas of research appropriate to the opportunities and problems and to suggest in this report the kind and extent of research and development necessary to advance comminution science and technology.

In the execution of its work, during the period December 4, 1978, through May 2, 1980, the committee conducted seven formal two-day meetings, plus a two-week visit to six European research and development facilities concerned with comminution. Additionally, the committee invited several expert witnesses to various committee meetings. This report is intended to provide an overview of the field, its present standing, and its promise. The report represents a unified point of view shared by the committee members and invited guests, rather than a set of compromises among widely divergent opinions.

John A. Herbst, Chairman

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Chapter I

SUMMARY

1.1 Introduction

This study has shown that U.S. industries use approximately 32 billion kWh of electrical energy per annum in size reduction operations. More than half of this energy is consumed in the crushing and grinding of minerals, one quarter in the production of cement, one eighth in the preparation and utilization of coal, and one eighth in the processing of agricultural products. However, current comminution technology is both energy-intensive and inefficient.

Size reduction steps consume about 70% of the total processing energy required to make grain products, on the order of 25% of the total processing energy required to make copper and iron, and approximately 12% of the energy required to produce cement. The efficiency of energy utilization in the size reduction steps involved in each case is less than 5%, indicating an enormous potential for improvement.

Current levels of research and development in comminution science and technology in U.S. industry, universities, and government laboratories are clearly not commensurate with the potential for return. Total government funded research in this area for 1979 is estimated at $1.7 million, while a decrease of only 10% in energy consumed in comminution in the United States would mean an annual savings (at current prices) of approximately $160 million in imported oil.

1.2 General Summary and Conclusions

The committee considered more than 15 topical areas involved in fundamental and practical aspects of comminution. Five specific areas (classification device design, comminution device design, control, grinding additives, and materials) have been identified as areas to focus on to achieve large, short-term (less than five years) savings in energy. Further, it has been estimated that the implementation of recommendations in only two of the areas, classification and automatic control, could result in short-term savings of 6 billion kWh per annum. In the longer term (five to ten years), additional savings on the order of 15 billion kWh seems feasible with significant improvements in comminution device design, grinding additives, and materials for liners and media.

The implementation of recommendations in both the short and long term requires the support of fundamental studies in a number of areas, including fragmentation science, particle-fluid and particle-particle dynamics, particle characterization, surface science, and materials science. It is apparent that a high level of interdisciplinary effort is necessary to carry out the required fundamental work and that such efforts must be strongly encouraged through workshops and joint research programs.

In addition to the fundamental research, the successful implementation of the recommendations presented here requires that risk assumption, a significant inhibitor for industrial acceptance of innovative technology, be limited in the case of comminution technology. To this end it is recommended that a series of local or regional pilot plants be established for scale-up testing and development of process improvement strategies. Such pilot plants would facilitate a wide range of training programs involving university and industrial people. In addition, a major national facility is being recommended for the testing of large-scale equipment. Such a facility would perform the final tests before incorporation of a new device or design into a plant or manufactured product. By this means the government could assure more rapid testing and transfer of new technology without industry's being expected to assume a disproportionate risk.

Some of the principal topics treated in the report and the associated recommendations are summarized below.

1.3 Specific Topic Areas and Recommendations

1.3.1 COMMINUTION DEVICES

The trend in mineral processing plants in the past two decades has been to install increasingly large individual grinding units, primarily to minimize capital costs. With the larger units, specific grinding energy consumption does not decrease, and for ball mills beyond a diameter of about 15 ft, it has been observed that specific energy consumption increases measurably.

Design engineers have taken advantage of the known attributes of standard processing equipment to provide effective comminution circuits, within the pertinent capital and operating cost constraints.

Changes that have appeared in conventional grinding circuits have centered on larger rod and ball mills. Limitations in the size of both rod mills (length) and ball mills (diameter) dictate the size and number of units for a given operation. Nonconventional developments have focussed on autogenous or semiautogenous grinding circuits, usually because of lower capital costs. However, grinding energy consumption with semiautogenous and especially autogenous grinding can be as much as 25% more than with conventional grinding. Decisions on which type of grinding circuit to use are influenced by the nature of the material to be processed, the energy aspects of steel (lines and media) consumption, and the energy consumed by the comminution devices.

Other novel features of comminution devices are the use of rubber or other polymeric materials as liners/lifters in place of conventionally used ferrous materials. A novel "square" and spiraled liner-lifter configuration is currently being tried in the Southwest on copper ore with reported reductions in specific grinding energy consumption. The application of gearless wraparound electric drives to large wet grinding mills is currently under consideration. The wraparound drive eliminates the gear reducers and large bull gears, which have been troublesome maintenance areas on large grinding mills.

Recommendations:

The potential energy savings associated with improvements in comminution device design are seen to be significant (on the order of 3%) in the short term, while rather large long-term savings (on the order of 20%) may be possible if the appropriate new technology is developed. Specific recommendations are as follows:

Short Term:

- The effect of liner configuration on grinding efficiency should be evaluated in both large and small mills.
- Studies of optimal equipment size, with particular concern for large diameter mills, should be carried out.
- Computer-aided design procedures should be developed to put more detailed and accurate information into the hands of the mill designer.

Long Term:

- o Autogenous or semiautogenous mills that would accept crude ore directly from a mine should be developed.
- o The influence of design variables on material transport through mills should be evaluated.
- o Zones of maximum fragmentation and wear in standard devices should be identified.

1.3.2 CLASSIFICATION

A comparative examination of closed circuit and open circuit grinding indicates that the specific energy required in closed circuit grinding will be 85% or less of that in open circuit. However, the minimum energy consumption for closed circuit grinding can be achieved only by employing a perfect classifier. Unfortunately, there are no perfect classifiers. Industrial classifiers suffer from apparent bypassing of particles and misplacement of both coarse and fine particles. The impact of these imperfections can be reduced in two ways: the development of better performing classifiers, or the staging of several (usually two) classifiers. The former is the more desirable solution; the latter is probably the near-term solution.

Recommendations:

Improving the performance of existing classifiers can lead promptly to very large energy savings, about 10%. Research in this field may be expected to more than double this percentage.

- o The poor classification performance of industrial classifiers because of apparent bypass and misplacement of particles should be reduced.
- o More information on the energy consumption of various industrial classifiers should be obtained.
- o A closed grinding circuit evaluation procedure should be developed. This procedure could be distributed to users of size reduction systems so that they could determine the performance of their grinding circuits.
- o Two-stage classification systems should be analyzed to determine potential benefits and actual benefits, difficulty of operation, and energy consumption.

1.3.3 INSTRUMENTATION AND CONTROL

Instrumentation for use in the harsh environment of industrial comminution plants must have a robustness not found in commercial units intended for other, less aggressive locations. Not all of the required instruments have been developed.

Considerable work has been done on the development of both analog and digital control schemes, and several systems are available. Optimization concepts are an integral part of modern systems using computer control, and several techniques for optimal control are available. However, in spite of two decades of research and development, there is no clearly defined strategy for gaining maximum benefit from the existing knowledge. Furthermore, remarkably few industrial comminution plants have installed or are using the available instrumentation and control system technology.

The proper education of the people involved with control systems provides one of the largest single areas for potential energy savings and these savings could be achieved in a relatively short time.

Existing data indicate that throughput increases or energy savings of 10% can be expected of improved control. If this gain were achieved in only half of the nation's comminution plants, the annual saving in energy or increase in productivity would be equivalent to fossil fuel energy of 5.2×10^9 kWh.

Because of the individual nature of comminution systems and mineral deposits, there is no universally best comminution control strategy, and successful control of a given plant can be achieved only through a well defined control development project. To achieve the potential benefits, appropriate education and training programs need to be developed and made available to senior management, plant engineers, and operators.

<u>Recommendations</u>

Control systems and devices are being introduced too slowly. Large savings, about 10% of the energy, are possible when adequate controls are introduced. Research into better control systems and instruments can lead to an additional 10% saving.

- o Active cooperation between industry and academic institutions with experience in automatic control should be fostered.

- o Fully instrumented pilot plant facilities should be established at or near universities for (a) improvement of normal student training, (b) development of short intensive courses for industrial personnel, and (c) on-line research for the development and evaluation of comminution control systems.

- A set of standard process computer simulation packages should be developed for improved analysis of control system performance.
- New on-stream instrumentation, particularly a robust and economical on-stream particle size analyzer and new devices for monitoring mineral liberation, should be developed.

1.3.4 GRINDING AIDS

A considerable incentive has always existed to develop additives for grinding which will substantially improve the efficiency of the process. A large amount of work appears to have been done in the development of grinding aids for dry grinding systems. Ever since the first attempts to develop grinding aids in the cement industry, nearly 50 years ago, most of the grinding aids development has been oriented toward cement clinker grinding. Examples of grinding aids that have been shown to be effective in cement clinker grinding include amines, organosilicones, glycols, carbon blacks, asphaltenes, calcium sulfate, urea, etc. Reported advantages of using these grinding aids typically range from a 10 to 50% increase in production rate at the same energy consumption. In addition, there are many instances demonstrating the effectiveness of silicones, wool grease, acetones, and a variety of surfactants in the dry grinding of materials like quartz, limestone, gypsum, etc.

In wet grinding systems, water itself may be considered a grinding aid because of the higher efficiency and mill capacity obtained in wet grinding relative to dry grinding. Along these lines, some investigators have examined the process of grinding in various organic liquids, such as n-hexane, oils, sorghum, etc. The effect of surface tension and viscosity on the breakage rate of particles has been studied, with indications of more efficient grinding in liquids of high surface tensions at an optimum viscosity. A fair amount of laboratory evidence points to improvements in grinding efficiency that might be obtained by grinding in various organic liquids. Utilization of such results, unfortunately, is economically impractical in the mineral processing industry.

More recently, results of a fairly extensive effort to develop polymeric grinding aids for wet grinding systems have been reported. These results indicate that significant advantages may be realized by using certain additives if the mill is operated at a solid percentage high enough so that viscosity effects become important. This work represents a fairly complete analysis of the possible mechanism of grinding aids and has considerable potential for further developmental work on grinding aids.

Recommendations

The significance of developing industrially applicable grinding aids, particularly for the mineral processing industry, where wet grinding is frequently employed, cannot be overemphasized. Grinding aids for wet grinding have not enjoyed widespread commercial use despite the voluminous reports on their potential. The short-term energy savings associated with the use of known technology in this area is estimated at 1 billion kWh per annum. In the longer term, new developments should result in additional savings on the order of 2 billion kWh per annum. Specific recommendations are as follows:

Short Term:

- Standard laboratory and pilot scale procedures should be developed for the evaluation of grinding aids.
- In wet grinding, the effects of the state of particle aggregation and pulp rheological characteristics should be quantified.

Long Term:

- Fundamental studies into the detailed mechanisms of grinding aids should be made.
- Molecular design to achieve various types of grinding-aid action should be pursued.

1.3.5 CORROSION AND WEAR

The metal wear in crushing and grinding systems is an expensive item in the mineral industry because of the cost of grinding media and worn equipment parts, the decrease in production, and downtime for maintenance. In addition, metal wear is a source of pollution and contamination. The total annual steel consumption is on the order of 1 million tons, corresponding to a fossil fuel energy content of 12 billion kWh. This amount represents 12% of the total energy consumed in comminution.

Mill and crusher liner consumption represents about 100,000 tons or 10% of the total steel consumption. Rubber liners are now replacing the steel liners successfully and economically in a wide range of fine grinding operations in ball mills and rod mills. The annual steel consumption in rods and balls used for comminution comprises 90% of the total. Because of corrosive conditions in the mill slurry in wet grinding, steel consumption can be up to 10 times that of dry grinding. In such cases, some milling operations use grinding balls of nonferrous materials, such as ceramics, to reduce costs. Ball hardness and composition and the grinding environment are important parameters of the consumption rate. In rods, high impact strength and high toughness are not always desirable because rods with these properties will coil up in the mill and decrease grinding efficiency. Rod diameter, length, hardness, composition, and straightness tolerance and the grinding environment are the main factors affecting rod consumption.

Recommendations:

The potential short-term savings achievable by using existing technology to improve grinding media are about 3% of the total comminution energy. Increased research can lead to additional savings of about 6% of the energy.

Short Term:

- o Rapid methods of confirming projections of the results of laboratory media tests to plant scale should be developed.

Long Term:

- o The role of each of the basic mechanisms of wear, namely, abrasion and corrosion, should be identified.
- o The influence of composition, structure, and heat treatment on steel consumption in comminution should be studied in depth.

1.3.6 NOVEL TECHNIQUES

Recent years have seen an increase in the use of roller mills, largely because of scale-up in design, which has permitted increased throughputs. Roller mills have several advantages over tumbling mills, including less power consumption and the ability to process materials containing up to 20% moisture.

Tumbling mills depend on the acceleration of gravity to impart energy to the balls. A number of designs are being tried to increase the relative acceleration between the material and the media.

Laboratory studies have shown that the grinding energy for cement clinker could be reduced by 30-50% of present levels by prestressing in a press before tumbling in a conventional mill. Investigations at larger tonnages are under way in the use of rolls as a means of prestressing the clinker.

In the USSR a wet grinding process is used in cement production. Energy consumption is reduced considerably by wet grinding. Vibrational energy is applied to the slurry, reducing the requirement of high water content for transport to the furnace. This lowered water content of the slurry in turn reduces the energy of firing.

A method of applying ultrasonic energy to the rollers in the nip configuration while crushing coal has led to an increased rate of size reduction in the laboratory. 1500-micron coal particles have been readily reduced to the ultrafine 2-7 micron range of interest of coal-oil mixtures and sulfur removal. The process has been demonstrated only in the laboratory, and the energy requirements are not yet known.

Cryopulverizing--comminution at low temperature--is being investigated both in the U.S. and in the USSR. It has been demonstrated to be useful in grinding plastics, elastomers, and some pharmaceuticals, although the energy saving is small with respect to total national consumption. In the USSR, studies are being made of cryogenic embrittlement of sulfur inclusions in minerals.

The differential expansion of rock caused by local heating initiates microcracks at grain boundaries, which greatly weakens the rock. A variety of techniques, such as lasers, electrical resistance, and microwave heating, have been used to heat rocks before mechanical excavation. Most of this work has been applied to tunneling in which bore holes or kerfs are created. Some suggestions have been made for the application of these techniques to mining, and some field tests have been made. Failure of engineering design rather than concept has so far prevented application in significant volume.

Recommendations

No areas could be identified as having particular potential for short-term energy savings, but some may have some potential for longer-term benefits.

1.3.7. BLASTING

Considerable research has been done in an effort to make blasting a science, including fabrication of simulated rocks both with internal flaws and with layers, high speed photoelastic measurements, and field tests in granite blocks. The field tests were large enough for simulation of a mine or quarry blast, but small enough so that all the fragments could be collected and screened. The results are in good agreement with laboratory observations.

High speed photography of full-scale bench blasting with a double row of blasting holes and the usual electronic firing device has shown that commercial blast initiators have statistical variations in their supposed accuracy which are unacceptable for efficient blasting.

Studies of blasting in a mine shaft in oil shale have led to the development of a computer program that can simulate the blast and predict the degree of fragmentation. Experiments have shown that fragmentation can be predicted and optimized. In some cases the techniques are applicable to other rock formations.

Recommendations:

- o Computer simulation techniques should be applied to blasting in additional types of mine and bench blasting operations.

- o A standards laboratory in the Bureau of Mines or the National Bureau of Standards should develop techniques for evaluating blast initiators and assist manufacturers in developing acceptable quality control.

1.3.8 FUNDAMENTALS

Any practical grinding operation can be considered to involve two fundamental processes:

- o Breakage--particles are broken from the application of mechanical (or other) stress

- o Transport--particles must be placed in the appropriate location for the application of stress, and the products must be removed.

Over the years, considerable effort has gone into fragmentation science, i.e., the study of the conditions required for fracture of individual particles and the characterization of the fragments produced. The crushing event has been characterized by a stress-strain curve carried to the point of rupture. Energy dissipation from a specimen loaded to fracture has been investigated in detail. Although only about 1% of the input energy can be accounted for by new surface energy, there is often a good correlation between specific surface and specific energy.

Breakage tests have been carried out on irregular single particles as well as on special regular shapes. These tests generally involve compression or impact. The prediction of the size distribution of fragments resulting from comminution has not been very successful. It is known that the greater the energy input to single particles, the greater the extent of comminution. The required average force, and energy, is a function of specimen size. The probability of being crushed by compression is lower for particles in beds than for those in single layers.

Continuous grinding systems involve additional transport processes for moving material into, through, and out of the grinding machine. Clearly, the breakage process is of prime concern, but in practice, grinding systems will often be limited by transport phenomena. Low efficiency in grinding is frequently the result not of inability to break particles effectively, but of the application of stress where there are no particles.

An understanding of comminution fundamentals obviously must include consideration of transport as well as breakage. The principal weakness of the conventional, energy-size models of comminution (which still provide the basis for most mill design) is that they essentially ignore the transport problem. Their apparent success can probably be attributed to the relative constancy of transport behavior among similar kinds of grinding machines.

The more modern population-balance models account for transport through the use of a breakage kinetics parameter which amounts, in effect, to the probability that a particle will be stressed in some time interval and that the stress will lead to breakage. A second parameter is then used to describe the nature of the product size distribution, given that breakage has occurred. Much of the recent research into the modeling of grinding systems has emphasized the empirical evaluation of these fundametal parameters and the development of the mathematical formulations necessary to apply them to the description, prediction, and, ultimately, control of mill performance. For the most part, the basic link between these process parameters and machine mechanics has not been established.

In coarse crushing, which involves relatively small numbers of quite large particles, the transport process is reasonably straightforward and energy input to the machine can be concentrated on breakage. Consequently, these processes are quite energy-efficient. In fine grinding equipment, on the other hand, transport and breakage normally are closely interconnected and are not amenable to independent control. In tumbling ball mills, for example, both the positioning of particles for stress application and the stress application itself are provided by the random tumbling motion of the balls. The result is that much of the energy input to the mill is wasted in nonproductive, direct, ball-to-ball contacts, and the energy-efficiency of such processes is generally low.

Research Needs and Recommendations:

Long-range energy savings are most likely to come from improved control of the transport aspects of the process, i.e., by taking better aim at the particles. Continued research into single-particle fragmentation may yield a test more suitable than present tests for estimating specific energy requirements for crushing and grinding equipment. Although such continued research is unlikely to yield a dramatic decrease in the fragmentation energy requirement for single particles, unless outside the usual ambient conditions of temperature and pressure, it may uncover a means of directing the applied forces more effectively at each particle so that less energy is wasted. Research into the breakage of beds of materials similarly may disclose means by which the efficiency of single-particle fragmentation can be approached for beds. Research into the breakage of assemblies of particles of different characteristics, i.e., hard vs. soft, may yield methods by which the fracture of either type can be accentuated over the other. Similarly, research may uncover means by which the valuable mineral may be more readily liberated from its composite host.

Significant, short-range savings can probably be achieved by optimizing the design and control of existing types of mills. Research needs in this area include:

- Further development of models and evaluation procedures;
- Establishment of a data base;
- Application of new techniques to circuit design and control.

Chapter II

ENERGY REQUIREMENTS WITH CURRENT TECHNOLOGY

2.1 Introduction

This section of the report tabulates the amount of energy consumed in comminution processes using current technology. Energy requirements are classified according to the commodity being processed, product size, and reduction ratio as well as type of device.

Size reduction is used for a variety of purposes in the process industries. In some instances, it is necessary to produce a product of given size and/or reactivity. In other cases size reduction is used to free a valuable component from a waste component prior to physical separation. The major energy-consuming commodities and the purpose of size reduction for each commodity are given in Table 2-1.

In almost all industries the energy (kilowatt-hours per ton of product [kWh/ton]) required to achieve a specified objective (e.g., product fineness or degree of liberation) has been and will continue to be the accepted measure of efficiency for a given size-reduction task. This energy consumption is the directly measured electrical input, kWh_e, which can be converted to a fossil-fuel equivalent basis, kWh_f, using an electricity generation efficiency of 32%. Thus 1 kWh_f = 0.32 kWh_e. In addition, in some cases the media and liner material required to achieve the required objective are also considered (pounds of steel per ton of product [lbs/ton]). Considering the very large amount of energy required to make the steel used for media and liners--more than 5 kWh_f/lb (i.e., 1.7 kWh_e/lb)--this is an important quantity for the present study. Unless otherwise specified, all energies reported as kWh will be kWh_e.

TABLE 2-1: Major Energy-Consuming Commodities and the Purpose of Size Reduction for Each

Minerals

Ore:

Feed size - run-of-mine
Product size - 0.2 cm or finer
Objective - liberation of valuable minerals from gangue to permit physical separation

Stone and Aggregates:

Feed size - run-of-mine
Product size - 100% minus 10.0 cm to 100% minus 1.0 cm
Objective - produce (size) graded materials

Cement Raw Materials and Limestone for Desulfurization:

Feed size - run-of-mine
Product size - 75 to 90% minus 0.075 cm
Objective - produce materials with high surface area to facilitate chemical reactions

Cement

Feed size - minus 1.0 cm
Product - 0.003 cm or finer
Objective - produce finished material with size distribution that will yield desired strength characteristics

Coal

Feed size - minus 2.5 cm
Product - minus 0.075 cm
Objective - (a) produce feed for electric power plants with desired combustion characteristics
(b) liberate pyrite and ash for cleaning

Grain

Feed size - minus 1.5 cm
Product - minus 0.15 cm
Objective - separate the endosperm from the bran and germ

Section 2.2 gives energy consumed by comminution for a number of commodities. Wherever possible both comminution energy and equivalent energy content of the steel consumed are reported. This information is helpful in identifying the total energy consumed in this manner in the United States and in identifying particular commodities that are prime targets for improvement. In Sections 2.3 and 2.4, energy requirements are classified according to the size-reduction task (coarse, intermediate, fine, and ultrafine) and the devices used for each task. This classification points to certain types of devices that are especially efficient for a given task.

2.2 Total U.S. Consumption of Energy in Comminution

This section examines the consumption of energy by comminution in the production of commodities involving four types of materials: minerals, cement, coal, and grain products.* Individual commodities whose production requires comminution are listed as completely as possible. This was done not only to arrive at a sound estimate of energy consumption, but also to show the pervasiveness of comminution in the production of materials.

It will be seen from the tabulation that the majority of the energy consumed by comminution in this country is concentrated in a few commodities. The total U.S. comminution energy used in the production of some commodities, such as boron, is smaller than the error in the estimated consumption for others, such as iron ore pellets. Commodities that consume less than 1 million kWh per annum (less than 10^{-3}% of total comminution energy in the United States) were omitted from the presentation.

The specific energy requirements for most commodities (kWh/ton) were obtained from a study of total energy consumed in mineral processing (Battelle Columbus Laboratories, 1975). From these data it was possible to obtain the percentage of energy used in the production of a commodity that is accounted for by comminution. The production figures were generally obtained from Mineral Commodity Summaries 1979 (Bureau of Mines, 1979). Where data sources other than the foregoing were used, the fact is indicated by footnotes to the tables that follow.

For steel consumed as grinding media and liners, specific consumption (lbs/ton of commodity) is given where possible. The electrical energy equivalent of the steel so consumed is quite significant and is given as well.

*Papermaking is not included in this survey. Although beating and mechanical pulping operations require 20 billion kWh to make 45 million tons of paper and 10 billion kWh to make 5 million tons of newsprint (Arthur D. Little, 1976), these processes are not traditionally considered comminution operations.

2.2.1 PRESENTATION OF FINDINGS

The energy used for comminution in the production of 43 commodities in 1977 and 1978 is tabulated in Table 2-2. For each commodity the table includes U.S. production, specific energy required for crushing and grinding, and total energy consumed by comminution in the production of the commodity.

The primary product listed for each commodity is the form that each takes after mineral and extractive processing. It may be the form that goes to a metal fabricator (e.g., aluminum ingot), a construction firm (e.g., cement), a road builder (e.g., gravel), or whatever other form a commodity takes en route to final use.

The specific energies given in Table 2-2 for crushing and grinding in the production of each commodity represent the electrical energy used per net short ton of primary product (kWh/ton). The values in some instances will be much greater than the more common values given in kilowatt-hours per ton of feed material. This occurs when the feed material has a low content of the constituent for which it is being processed. For example, the grinding specific energy of copper is given as 2013.9 kWh/ton of copper in refined copper. The average grade of copper ore in this country is 0.5% Cu (Bureau of Mines, 1976). Thus the equivalent energy consumption is 10.1 kWh/ton of crude ore feed.

The specific energies in the table are representative of the values usually encountered in the United States. The accuracy of the figures is not given here, but they are claimed to be the best available (Battelle, 1975).

The percentage of the total processing energy that is used for comminution is given for most of the commodities in Table 2-2. Some of the operations used in processing these commodities are given in Appendix A.

The energy used in the United States for comminution for each of the commodities listed was calculated as explained in Appendix A and is given in billions of kilowatt-hours (10^9 kWh). These are the most important values in the table: they determine the significance of the given commodity in terms of total energy consumed in the United States by comminution and make it possible to identify the major users. Note that each commodity is ranked according to the amount of energy it requires for comminution.

Each of the commodities potassium, manganese, titanium, uranium, and zinc can be processed by different means so that the grinding requirements in terms of such factors as product size depend on how the commodity is processed. Thus the energy requirement for comminution for a commodity changes according to the process, even though the primary product is essentially the same. In these instances the production figure for the commodity is broken down in Table 2-2 into those fractions produced by different processes. The comminution energy for each process is calculated separately.

There are several different forms of the commodity clay that depend on the source; these are given as bentonite, kaolin, fire, fullers, and common.

TABLE 2-2: Energy Requirements and Production Figures for 43 Commodities

Commodity	Primary Product	Production 000 Short Tons 1977	Production 000 Short Tons 1978	Specific Energy kWh/ton Crushing	Specific Energy kWh/ton Grinding	Steel consumed lbs/ton	% of Total for Processing	Total Energy for Comminution 10^9 kWh* 1977	Total Energy for Comminution 10^9 kWh* 1978	Rank
Aluminum	Al in Aluminum Ingot	4,539	4,800	12.5	36.63[1]	NA	0.2	0.2230	0.2358	13
Arsenic	Arsenic Oxide	NA	NA		81.0	NA	8.6	0.0011[2]	0.0011[2]	41
Asbestos	Fiber	103	100	48.56	228.96	NA	43.0	0.0286	0.0278	26
Barite	Ground	2,449	2,800	0.73[3]	19.53[3]	0.15	27.7	0.0496	0.0567	20
Boron	Boron Oxide (B_2O_3)	749[4]	775[4]	9.59		NA	1.1	0.0072	0.0074	35
Calcium	Lime[5]	19,947	20,200	9.62		NA	1.0	0.1919	0.1943	14
Cement	Portland, Masonry	78,600	83,200	9.0	80.0	2.0	12.0	6.9954	7.4048	1
Ceramics	Common brick	17,600	NA	2.17		NA	1.0	0.0381[6]	0.0381[6]	23
Chromium	Cr in Chromite ore	291	218	7.8[7]		NA	0.15	0.0023	0.0017	40
Clays		53,468	55,767					0.4233	0.4466	9
	Bentonite	3,746	4,202	0.7	16.65	NA	9.2	0.0650	0.0729	
	Kaolin	6,489	6,821	3.37	28.52	NA	12.8	0.2069	0.2175	
	Fire Clay	2,966	3,147	0.42	3.28	NA	27.1	0.0110	0.0116	
	Fullers	1,428	1,561	NA	NA	NA	NA	NA	NA	
	Common Clay	37,945	39,078	0.42	3.28	NA	NA	0.1404	0.1446	
Coal	Bituminous & Lignite[8]	532,817	535,419	0.21	7.46			4.0819	4.1013	2
Copper	Cu in Refined Cu	1,496	1,500	396.3	2013.9	273.0	23.1	3.6057	3.6153	4
Diatomite	Calcined Diatomite	648	667	5.9		NA	7.0	0.0038	0.0039	37
Feldspar	Ground Feldspar	734	780	3.43	11.3	1.72	38.5	0.0108	0.0115	33
Fluorspar	Finished	169	130	11.28	87.74	2.48	20.2	0.0167	0.0129	31

*1 kWh is equivalent to 3.60×10^{-3} GJ

TABLE 2-2 (Continued)

Commodity	Primary Product	Production 000 Short Tons 1977	Production 000 Short Tons 1978	Specific Energy kWh/ton Crushing	Specific Energy kWh/ton Grinding	Steel consumed lbs/ton	% of Total for Processing	Total Energy for Comminution 10^9 kWh 1977	Total Energy for Comminution 10^9 kWh 1978	Rank
Gold[9]	*	0.0377	0.0333					0.0317	0.0280	25
	Au from precious metal ores	0.0230	0.0203	1,379,200		30.1	18.2	0.0317	0.0280	
	Au from base metal ores	0.0147	0.0130							
Grain	Milled Grains[10]						70.0	1.880[10]	1.880[10]	6
Gypsum	Calcined Gypsum	13,390	14,700	5.4	10.0	NA	10.9	0.2062	0.2264	12
Iron Ore	Pellets (10^3 lg tons)	49,100	70,800	12.8	34.3[24]	20.0	25.0	2.3126	3.3349	5
Lead[11]		537	531					0.1245	0.1232	15
	Pb from Lead Ores	456	451	28.6	244.5	23.4	12.0	0.1245	0.1232	
	Pb as a by and co-product	81	80							
Lithium[12]	Contained Li	4.9	4.3	1516.0	2589.0	537.1	5.0	0.0201	0.0177	29
Magnesium	Mg Metal	104.0	110.0	10.4[13]		NA		0.0011	0.0011	42
Manganese	Ferromanganese	409[14]	406[14]					0.0053	0.0053	36
	FeMn by blast furnace process	224	223		12.5	NA	0.3	0.0028	0.0028	
	FeMn by electric arc furnace	183	183		13.3	NA	0.3	0.0025	0.0024	
Mercury	Mercury Metal	1.1	1.0	3386		NA	9.0	0.0037	0.0034	38
Mica	Ground	176.0	201.0	100	200	NA	19.1	0.0528	0.0603	19
Molybdenum[15]		61.2	66.0					0.2130	0.2293	11
	Mo from Molybdenum Ores	40.4	43.6	725	4,547	378	25.5	0.2130	0.2293	
	Mo as by-product	20.8	22.4							
Nickel	Contained Ni	13.9	10.6	2,434.8		273	17.8	0.338	0.0258	27

*Gold production: 1,100,000 troy ounces, 1977; 970,000 troy ounces, 1978

TABLE 2-2 (Continued)

Commodity	Primary Product	Production 000 Short Tons 1977	Production 000 Short Tons 1978	Specific Energy kWh/ton Crushing	Specific Energy kWh/ton Grinding	Steel consumed lbs/ton	% of Total for Processing	Total Energy for Comminution, 10^9 kWh 1977	Total Energy for Comminution, 10^9 kWh 1978	Rank
Perlite	Expanded	871	890	8.7		NA	1.7	0.0076	0.0077	34
Phosphate	P_2O_5 Acid	47,256	49,000		26.7	NA	3.0	1.2617	1.3083	7
Potassium[16]		1,764	1,795					0.0144	0.0147	30
	KCl by Flotation	1,411	1,436	5.4		NA	2.2	0.0076	0.0078	
	KCl by Crystallization	353	359	19.2		NA	2.5	0.0068	0.0069	
Pumice	Crushed	4,109	4,156	6.1		NA	23.4	0.0251	0.0254	28
Rare Earth[17]	Rare Earth Oxides	18.5	20	62.6	103.0	NA	6.8	0.0031	0.0033	39
Sand & Gravel[18]		939,200	937,000					0.6692	0.6749	8
	Sand	392,100	395,400	0.2		NA	4.7	0.0784	0.0791	
	Gravel	537,100	541,600	1.1		NA	18.2	0.5908	0.5958	
Salt	Rock	43,412	43,680	1.5		NA	9.4	0.0651	0.0655	18
Silicon	Si in Silicon & metal alloys	538	530	1.3		NA	0.02	0.0007	0.0008	43
Silver[8]	*	1.3	1.3					0.0460	0.0460	22
	Ag in Silver Ore	0.4	0.4	115,089		NA	46.9	0.0460	0.0460	
	Ag in Base Metal Ores	0.9	0.9							
Stone[20]	Crushed	954,000	997,000	3.7		NA	25	3.5298	3.6889	3
Talc	Processed	1,204	1,270	24.6		NA	56.9	0.0296	0.0312	24
Titanium[21]						NA		0.3209	0.3217	10
	Ti Sponge	13.8	17.2	232.7		NA	0.7	0.0032	0.0040	
	TiO_2 by Chloride Process	NA	NA		418	NA	5.0	0.1517	0.1517	
	TiO_2 by Sulphate Process	NA	NA		406	NA	9.5	0.1660	0.1660	

*Silver production: 38,200,000 troy ounces, 1977; 38,000,000 troy ounces, 1978

TABLE 2-2 (Continued)

Commodity	Primary Product	Production 000 Short Tons 1977	Production 000 Short Tons 1978	Specific Energy kWh/ton Crushing	Specific Energy kWh/ton Grinding	Steel consumed lbs/ton	% of Total for Processing	Total Energy for Comminution 10^9 kWh 1977	Total Energy for Comminution 10^9 kWh 1978	Rank
Tungsten	W in W Concentrate	6.0	7.2	1724		613	5.2	0.0103	0.0124	32
Uranium[22]		14.0	NA					0.0876	0.0876	16
	U_3O_8 by Acid Process	4.7		1900	2560	260	6.0	0.0192	0.0192	
	U_3O_8 by Alkaline Process	4.7		1805	9500	1050	10.0	0.0486	0.0486	
	U_3O_8 by Resin Process	4.7		2100	2500	2100	6.0	0.0198	0.0198	
Vermiculite	Processed	359.0	337.0		152.4	NA	20.0	0.0547	0.0514	21
Zinc[23]	Slab Zinc	450.0	340.0					0.1076	0.0811	17
	Zn as Product or Co-product	320.0	241.0			30.2		0.1076	0.0811	
	Zn by ET Process	112.0	84.0	19.7	299	28.7	5.0	0.0357	0.0268	
	Zn by EL Process	144.0	109.0	21.6	329	31.5	6.6	0.0505	0.0382	
	Zn by RE process	64.0	48.0	20.7	314.3	30.1	6.2	0.0214	0.0161	
	Zn as By-product	130.0	99.0							

FOOTNOTES FOR TABLE 2-2

1. Includes 5.0 kWh/net ton of aluminum for the crushing and grinding of coke to make carbon anodes and 0.2 kWh/net ton of aluminum for the crushing and grinding of anthracite to make carbon cathodes.

2. Total energy required for comminution based on the Battelle Laboratories 1973 production estimate of 13,000 tons arsenic oxide.

3. Figures are based on the weighted averages of specific energies for "Nevada-type" ground barite (73% of production) and "Missouri-type" ground barite (27% of production). The "Nevada-type" ground barite requires 1.0 kWh/net ton for primary crushing and 20.65 kWh/net ton for grinding. The "Missouri-type" ground barite requires 16.52 kWh/ton for grinding. The percentages of production were based on production figures for 1976, "Minerals Yearbook," Bureau of Mines.

4. The equivalent weight of B_2O_3 for the years 1977 and 1978 was estimated using a ration of 0.51 ton B_2O_3/ton boron minerals and compounds as determined from 1976 production figures, "Minerals Yearbook," Bureau of Mines.

5. The primary product, lime, includes quicklime, hydrated lime, and calcined dolomite.

6. Estimated from 1973 production figures.

TABLE 2-2 (Continued)

7. Estimated from a specific energy of 5.0 kWh/net ton of ferrochrome; 1976 production figures for ferrochrome and chromium content of ferrochrome and chromite ore were used to estimate the grinding energy per tone chromium, "Minerals Yearbook," Bureau of Mines.

8. Does not include coking coals, "Energy Data Reports," January 1, 1979 (Department of Energy, 1979).

9. The precious metals ores from which gold is recovered include gold and gold-silver ores. The base metal ores from which gold is recovered include copper, lead, zinc, copper-lead, lead-zinc, copper-zinc, and copper-lead-zinc ores. The 1976 distribution of gold recovered from gold and gold-silver ores (61%) was used to estimate the 1977 and 1978 distributions, "Minerals Yearbook," Bureau of Mines.

10. The primary product of the commodity grain includes (SIC Code and Title):

    2041 - Flour and Other Grain Mill Products
    2044 - Rice Milling
    2046 - Wet Corn Milling

    The electricity purchased by the above groups was obtained from "Energy Accounting in the Food Processing Industry" by G. A. Barton and T. J. Lutton, Department of Agriculture (1979) for the year 1975. This figure was multiplied by a factor of 0.7 (supplied by a grain mill) to obtain an approximate estimate of the electricity used in the size reduction of grains. The 1975 figure is used for the years 1977 and 1978 in order to obtain an approximation of the energy requirements for those years.

11. Lead is recovered from lead ores and as co- or by-product from zinc, lead-zinc, copper-lead, copper-zinc, and copper-lead-zinc ores. An 85% recovery of lead from lead ores was based on 1976 production figures, "Minerals Yearbook," Bureau of Mines.

12. U.S. production was withheld by the Bureau of Mines so apparent consumption is used. Lithium products include lithium carbonate, lithium hydroxide, and lithium chloride, "Minerals Yearbook," Bureau of Mines.

13. U.S. production was withheld by the Bureau of Mines so reported consumption is used. Most magnesium is recovered by processing seawater. The crushing energy reported is that used to crush calcined dolomite. The calcined dolomite is mixed with seawater to form magnesium hydrate.

14. The production of ferromanganese in 1977 and 1978 is estimated from the reported consumption of manganese ore (in 1977 and 1978) using data from 1976, "Minerals Yearbook," Bureau of Mines. The distribution of ferromanganese between that produced by the electric furnace process and the blast furnace process is the 1973 figure given by Battelle Laboratories.

15. Molybdenum is recovered from molybdenum ores and as a by-product from, primarily, copper ores. The distribution between the two (66% recovered from molybdenum ores) is taken from 1976 data, "Minerals Yearbook," Bureau of Mines.

16. The distribution between KCl produced by the flotation process and KCl produced by the crystallization process from Battelle Laboratories (1973).

TABLE 2-2 (Continued)

17. Apparent consumption figures are used since production figures are withheld.

18. Distribution between sand and gravel production from 1976 production figures, "Minerals Yearbook," Bureau of Mines.

19. Silver is recovered from silver ores and from base metal ores. The base metal ores include copper, lead, zinc. copper-lead, lead-zinc, copper-zinc, and copper-lead-zinc ores. Silver which is recovered from gold and gold-silver ores, accounting for 3% of silver produced, is lumped with the silver produced from base metal ores. Distribution of silver from 1976 production figures, "Minerals Yearbook," Bureau of Mines.

20. Stone includes granite, limestone, marble, marl, sandstone, shell, slate, traprock, and miscellaneous, "Minerals Yearbook," Bureau of Mines.

21. The production figures for TiO_2 and the distribution of TiO_2 recovered by the chloride and sulfate processes are not given for 1977 and 1978. 1973 data from the Battelle report is used.

22. 1978 production figures for U_3O_8 are not available so the 1977 figure for the total energy required for comminution is used for 1978.

23. Zinc is recovered from zinc ores and as a by- or co-product from lead, zinc-lead, copper-zinc, copper-lead, and copper-zinc-lead ores. The distribution between the two sources is from 1976 production data, "Minerals Yearbook," Bureau of Mines.

24. Calculated from data in "Steel Grinding Media Used in the United States and Canada" (Nass, 1974) and "Status of the Iron Ore Industry - 1978" (Klinger, 1978).

The commodities gold, lead, molybdenum, and silver have each been identified in Table 2-2 as coming from an ore or as co- and by-products. Only the primary product from an ore being treated principally for that commodity has an energy requirement associated with it.

Steel consumed in crushing and grinding in the form of liners and grinding media (rods and balls) is given for 13 commodities in Table 2-2. The values are in pounds of steel per net short ton of product (lbs/ton). For copper ore, iron ore, and cement clinker, the values are for the grinding media only.

The energy required to produce the steel consumed in comminution by the 13 commodities noted above is given in Table 2-3. The values are in kilowatt-hours per net short ton of primary product (kWh/ton) and are referred to as specific steel consumption. The data has been converted from fossil fuel equivalent to electrical power equivalent to permit comparison with the direct electrical power information given for comminution energy. Again, consumption for copper ore, iron ore, and cement clinker includes grinding media only. "Total contained energy" of the steel used in comminution is calculated as explained in Appendix A. The result, given in billions of kilowatt-hours (10^9 kWh), shows the amount of energy that goes into the steel consumed in comminution for the particular commodity.

2.2.2 DISCUSSION

The total energy used for comminution in the United States and its distribution by commodity group are shown in the following tabular summary and Figure 2-1a.

The amount of electric power generated in this country is 2.2×10^{12} kWh per annum (DoE, 1978). On that basis comminution consumes approximately 1.3% of the electric power produced, equivalent to about 47.6×10^6 barrels of oil per annum.*

The 10 commodities with the largest size reduction energy requirements for comminution in 1978 are tabulated below.

Of the total energy used for comminution in 1978, 26.77×10^9 kWh or 93.9% was consumed by the 10 leading commodities, and the leading five accounted for 77.7%.

The 13 commodities in Table 2-3 require equivalent electric power of 3.48×10^9 kWh in the production of steel media consumed in grinding. Of these 13 commodities, cement, copper, and iron ore account for 3.39×10^9 kWh and lie in the top five consumers of energy for comminution. Allowing for appropriate media contributions for the other commodities, an estimate of 3.8×10^9 kWh for the total energy equivalent for steel consumption is not unreasonable. This media contribution represents an additional 12% energy requirement on top of the indicated total of 28.5×10^9 kWh consumed in comminution.

*The energy content of one barrel is 6,287,000 Btu and a Battelle (1975) estimate to produce one kWh is 10,500 Btu based on a 32% efficiency.

TABLE 2-3: Energy Requirements for the Production of Steel Consumed as Grinding Media and Liners

Commodity	Primary Product	Production 000 Short Tons 1978	Specific Steel Consumption kWh/T *	Contained Energy in Steel 10^9 kWh
Barite	Ground Barite	2,800	0.3	0.0008
Cement	Portland, Masonry	83,200	3.4	0.2829
Copper	Cu in refined Cu	1,500	464.1	0.6962
Feldspar	Ground Feldspar	780	2.9	0.0023
Fluorspar	Finished	130	4.2	0.0005
Gold	Au from Precious Metal Ores	0.0203	51.2	0.0000
Iron Ore	Pellets (10^3 lg tons)	70,800.0	34.0	2.4072
Lead	Pb from Lead Ores	451.0	39.8	0.0179
Molybdenum	Mo from molybdenum Ores	43.6	642.6	0.0202
Nickel	Contained Ni	10.6	464.1	0.0049
Tungsten	W in W Concentrate	7.2	1,042.1	0.0075
Uranium	U_3O_8 by Acid Process	4.7	442.0	0.0021
	U_3O_8 by Alkaline Process	4.7	1,785.0	0.0084
	U_3O_8 by Besin Process	4.7	3,570.0	0.0168
Zinc	Zn by ET Process	84.0	48.8	0.0041
	Zn by EL Process	109.0	53.6	0.0058
	Zn by RE Process	48.0	51.2	0.0025

*1 kWh is equivalent to 3.60×10^{-3} GJ

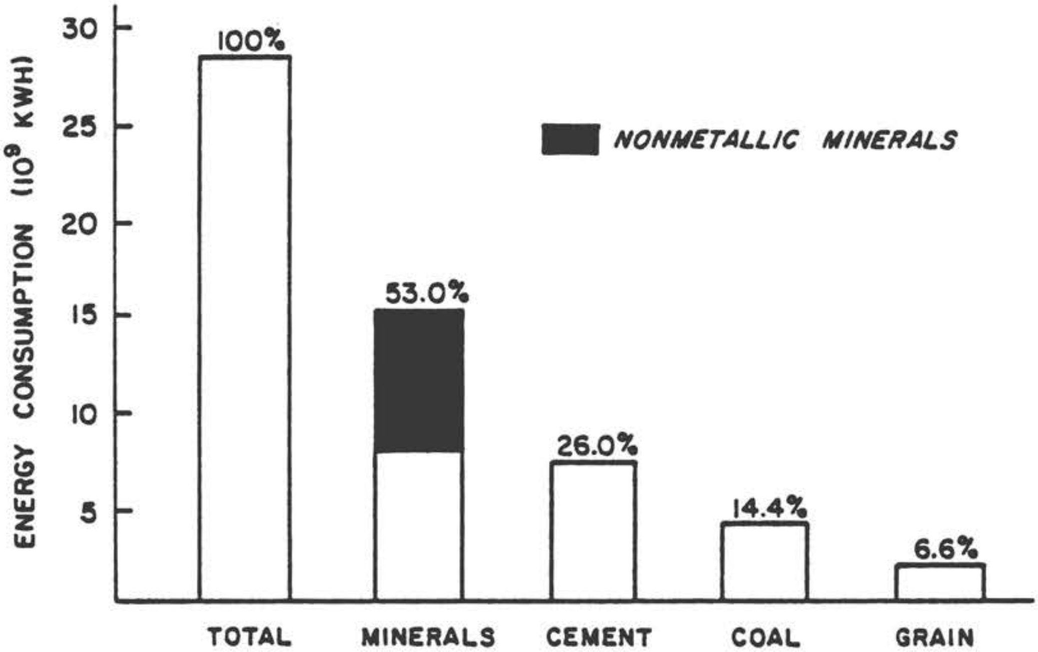

FIGURE 2-1(a) The energy for comminution in the mineral, cement, coal and grain commodity groups.

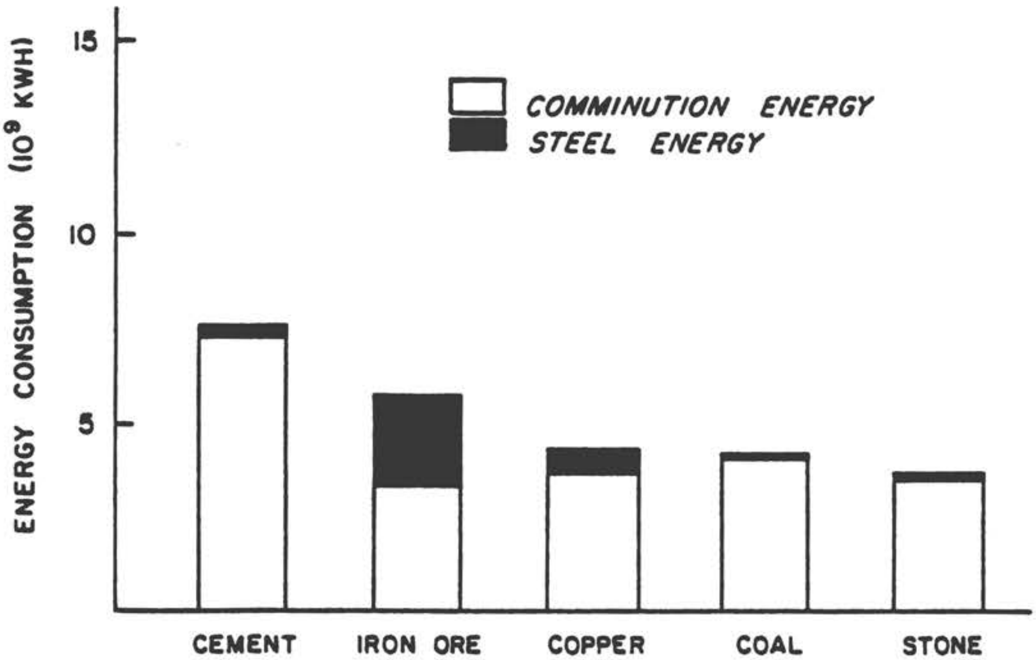

FIGURE 2-1(b) The energy for comminution and steel wear in the 5 leading commodities.

SUMMARY OF ENERGY USED FOR COMMINUTION

Commodity Group	Production(10^3 sTPY)	Energy(10^9 kWh)	Distribution(%)
MINERALS	-	15.1	53.0
Metallic	-	(7.9)	(27.7)
Nonmetallic	-	(7.2)	(25.3)
CEMENT	83,200	7.4	26.0
COAL	535,419	4.1	14.4
GRAIN	264,000+	1.9	6.6
	+10^3mTPY	28.5	100.0

TEN LEADING ENERGY CONSUMERS

Rank	Commodity	Primary Product	Size Reduction Energy (10^9 kWh)
1	Cement	Portland, Masonry	7.40
2	Coal	Bituminous and Lignite	4.10
3	Stone	Crushed	3.69
4	Copper	Cu in Refined Cu	3.62
5	Iron Ore	Pellets	3.33
6	Grain	Grain Mill Products	1.88
7	Phosphate	P_2O_5 Acid	1.31
8	Sand and Gravel	Washed	0.67
9	Clay	Bentonite, Kaoline, Fire Fullers, Common	0.45
10	Titanium	Sponge Ti, TiO_2	0.32

If the media energy requirements and comminution energy consumption are added, the top five commodities account for 80% of the total energy input. The distribution is shown below and in Figure 2-1b.

The two largest individual energy items, which are of approximately equal magnitude, are the energy for cement comminution and the energy for steel media in iron ore comminution. The latter is about equal to 25% of the comminution energy consumed by all commodities and therefore is an area that merits research. The amount of steel consumed by comminution for all commodities exceeds the 3.8×10^9 kWh figure derived above. The steel consumption estimate is 10^6 tons or about 1% of U.S. production.

LEADING FIVE COMMODITIES
COMMINUTION PLUS EQUIVALENT STEEL ENERGY

	Energy (10^9 kWh per annum)		
Commodity	Comminution	Steel	Total
Cement	7.40	0.28	7.68
Iron Ore	3.33	2.41	5.74
Copper	3.62	0.70	4.32
Coal	4.10	0.16*	4.26
Stone	3.69	0.14*	3.83

*estimated in proportion to cement

2.3 Product Size and Energy Input

The processing of a mineral or other commodity may involve several stages of size reduction, each stage being termed a task. Associated with a task is the specific energy required to bring about the size reduction. In this section, specific energies for various stages of reduction are tabulated. Given are specific energies for tasks that have been reported in the literature and specific energies for the same tasks as calculated from Bond's empirical energy-size reduction relationship (Section 3.3.1). This commonly used equation gives the specific energy required to reduce a particular material from a given feed size to a given product size. A qualitative discussion of the general relationships between energy input and product particle size in terms of the physics of breakage is also given here.

2.3.1 PRESENTATION OF FINDINGS

The energy requirements for comminution are tabulated by tasks in Table 2-4. The tasks follow roughly the same stages of size reduction as material in a mineral processing plant, except that ultrafine grinding is an extreme case. Specialized products such as pigments, pharmaceuticals, and chemicals are examples of materials reduced to ultrafine sizes. Typical feed and product sizes are given for each task in Table 2-4. The calculated specific energy from Bond's equation is that required for material having a Work Index of 13.8 kWh/ton as determined from a ball mill grindability test. This value represents an average of many materials (Bond, 1961). Also given in Table 2-4 are experimental specific energies reported in the literature for given tasks.

The observed specific energy of 0.5 kWh/ton for explosive shattering in Table 2-4 is calculated from the energy used to produce explosives, 15,000 Btu/lb (Battelle, 1975), an energy conversion of 10,500 Btu/kWh, and an average powder factor of 0.35 lb/ton for copper and iron ores.

2.3.2 DISCUSSION

Table 2-4 shows that the energy needed to perform each task increases as the product size decreases. When the observed specific energies are plotted (Figure 2-2), the increases are seen to be exponential. Note, for example, that only 1 kWh/ton is required to produce 1-cm particles, while more than 200 kWh/ton are required to produce 0.0001-cm particles. This increase is due to many factors and results not only from the type of device and the macroscopic conditions, but also from the mechanisms that operate at the individual particle level.

Any comminution device grinds an assemblage of particles having a distribution of sizes, and under these conditions the larger particles have a greater chance of being captured and broken than the smaller particles. Thus smaller particles require more events in which breakage can occur. Also, larger particles have many flaws which are depleted in the succeeding breakage events that occur in the daughter fragments. The reduced number of flaws means that there is less chance of breakage of smaller particles. Increased energy inputs are necessary, themselves, to raise the number of events by which smaller particles may be broken. These and many other conditions contribute to the higher specific energies required for small product sizes.

The range in reported specific energies for a particular task undoubtedly is due to differences in material hardness, the type of comminution device used, or the ratio of feed size to product size (reduction ratio).

TABLE 2-4: Comminution Energy Requirements Classified According to Task

Size-Reduction Task	Feed to Product Size Range (cm)	Calculated+ Specific Energy (kWh/ton)	Observed Specific Energy (kWh/ton)
Explosive shattering	∞ - 100	0.4	0.5
Coarse	100 - 2	1.0	0.15 - 4 [2]
Intermediate	2 - 0.02	8.9	0.3 - 10.25 [1, 2]
Fine	0.02 - 0.001	33.9	4.25 - 28.3 [2]
Ultrafine	0.001 - 0.0001	94.4	35.0 - 800.0 [3]

+Calculated according to Bond's equation using an average work index, W_I, of 13.81

(1) Bond & Wang, 1950
(2) Lowrison, 1974
(3) Herbst, 1978

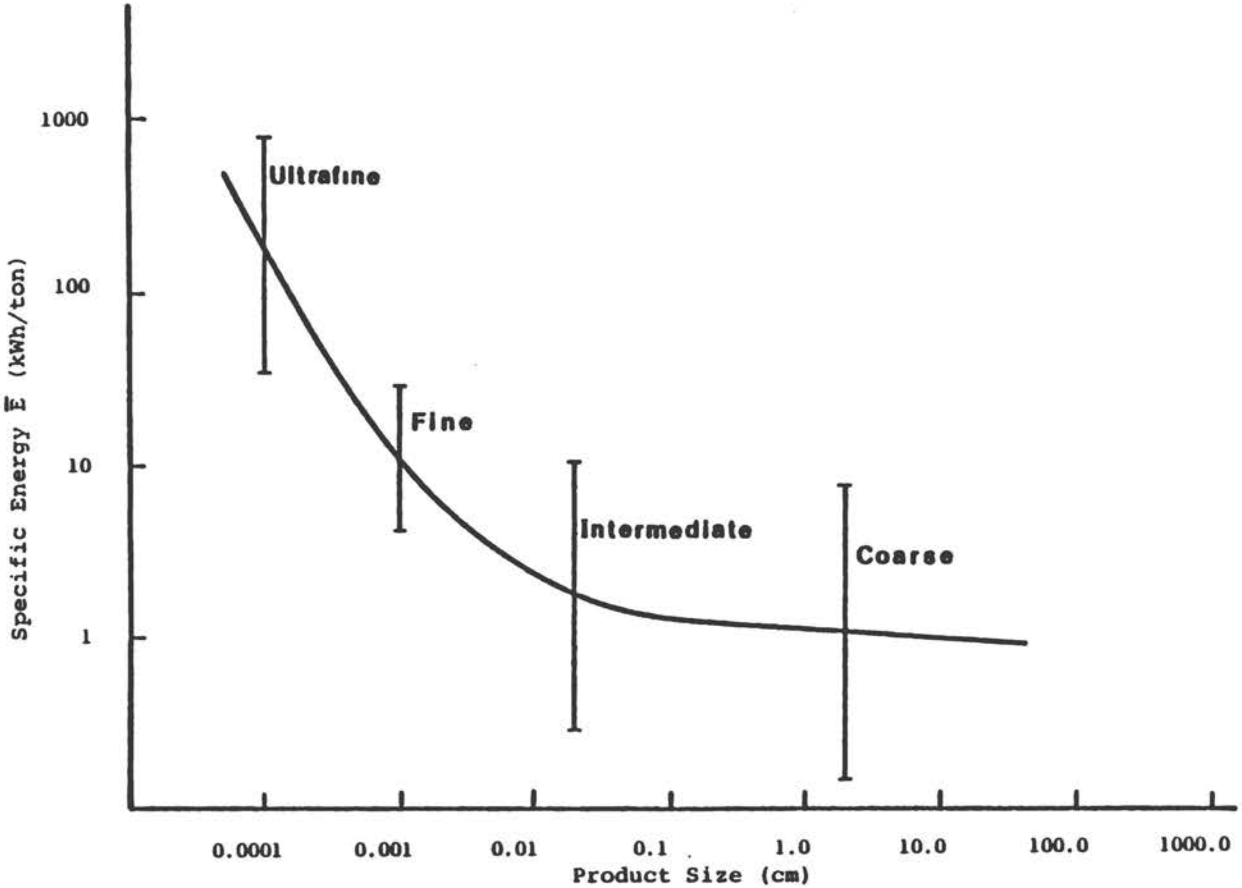

FIGURE 2-2 Reported energy requirements for size reduction tasks.

2.4 Equipment Type and Energy Consumption

Comminution devices usually perform just one task of size reduction, and their energy consumption is intimately tied to the task performed. For example, a crusher is assigned the task of coarse size reduction, and its energy consumption is correspondingly low. Energy consumption and task are closely related largely because all devices used in a specific size range exhibit similarities in modes of breakage. Devices grouped here according to the task they perform--coarse, intermediate, fine, and ultrafine--show common attributes with respect to grinding, feed and product sizes, specific energy consumption, and efficiency. Ranking of devices by energy requirements shows which types merit further attention with regard to improving energy utilization.

Comminution, as noted earlier, consumes steel as liners and media because of the abrasive action of the grinding process. The equivalent electric energy required to produce the steel consumed in comminution is presented here by type of device.

2.4.1 PRESENTATION OF FINDINGS

The energy requirements for comminution according to type of device are given in Table 2-5. The devices are grouped by task: coarse, intermediate, fine, and very fine grinding.

Examples of commodities ground in the device are shown in each case. The commodities are grouped into: minerals; cement; coal; pigments, pharmaceuticals, and chemicals; and agricultural products.

The feed and product sizes given for a device are those reported in the literature as typical. However, the reduction ratio--the ratio of feed size to product size--obtained from the sizes in the table may not be that actually encountered for a specific case.

The range of specific energy requirements for each device covers those reported in the literature for various materials, feed and product sizes, circuit configurations, and capacities. The arithmetic mean of the low and high specific energies is plotted against the feed and product sizes in Figure 2-3. Where possible, the efficiency of a device is given (Table 2-5, Figure 2-4). The definition of efficiency is discussed in Section 3.6.

The wear of steel liners and media for crushers, autogenous grinding, and rod and ball mills is given in Table 2-6 and was determined by the method given in Appendix B. The consumption of steel in terms of weight and in terms of energy used to process iron ore to make the steel is given per ton of material ground. These values are also plotted in Figure 2-5 against typical product sizes for crushers, rod, and ball mills.

TABLE 2-5: Comminution Energy Requirements Classified According to Device Type

Device	Material	Feed to Product[2] Size Range (cm)		Typical Energy Requirements (kWh/ton)	Estimated Energy Efficiency (%)
I. COARSE					
Jaw crusher	a.	100.	2.	0.15-3.[1]	33.[6]
Gyratory crusher	a.	70.	2.	0.3-0.5[1, 2]	
Rotary impactors	a., c	30.	1.	0.2-4[2]	
Autogenous mills (dry)	a.	15.	0.01	16[1]	
Roll crushers	a.	6.	0.4	0.8-4[2]	78.7[4]
II. INTERMEDIATE					
Rod tumbling mill	a.	5.	0.08	1.5-3.5[2]	
Hammer mill	a.	4.	0.09	0.3-1.5[1]	
Pan mill		2.	0.004	2.0[2]	
Ring ball mill	a., c.	0.2	0.005	7.5[2]	
Ring roll mill	a., c	0.2	0.005	10.10, 10.25[2]	30., 60[5]
Ball race	c.	1.0	0.020	9.0	11.8

TABLE 2-5 (Continued)

Device	Material	Feed to Product[2] Size Range (cm)		Typical Energy Requirements (kWh/ton)	Estimated Energy Efficiency (%)
III. FINE					
Ball tumbling mill	a., b., c.	0.1	0.0006	4.25-21.8[2]	5.6[4]
Vibration mill (dry)	a.	0.05	0.0006	13.3-28.3[2]	
Pin mill	e.	0.03	0.0001	14-37[2]	
Roller mill	b., e	7.0	0.01	9.5	
IV. VERY FINE					
Attritor	a., d	0.005	$<$ 0.0001	35.-800[3]	2.0[4]
Fluid energy	a., d	0.003	0.0001	248-600[2, 4]	0.7[4]

(a) Minerals, clay, stone
(b) Cement
(c) Coal
(d) Pigment, pharmaceuticals, chemicals
(e) Agricultural products

[1] Bond & Wang, 1950
[2] Lowrison, 1974
[3] Herbst, 1978
[4] Stairmand, 1963
[5] Heywood, 1947
[6] Bennet & Brown, 1941

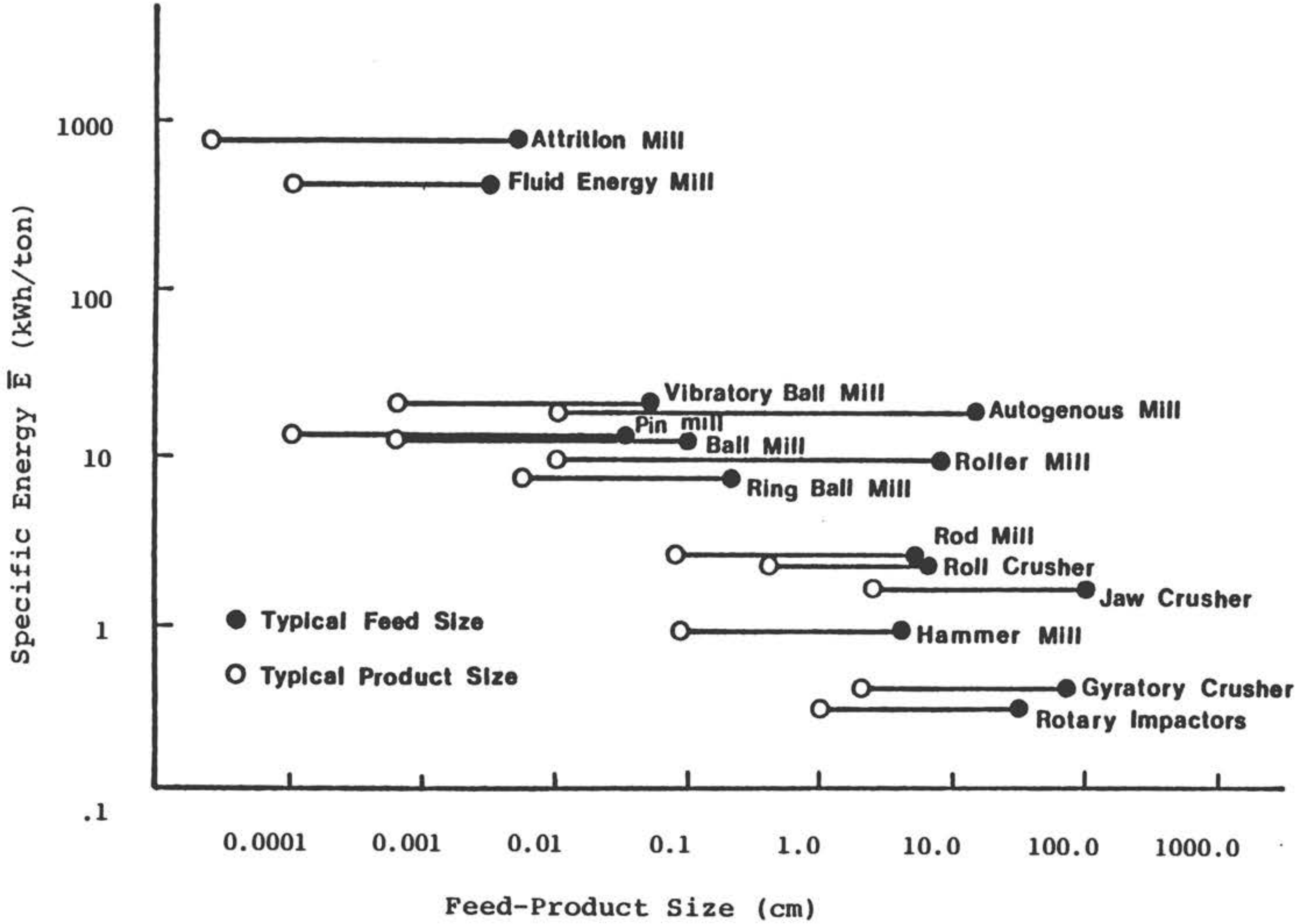

FIGURE 2-3 The reported average energy requirements for several devices.

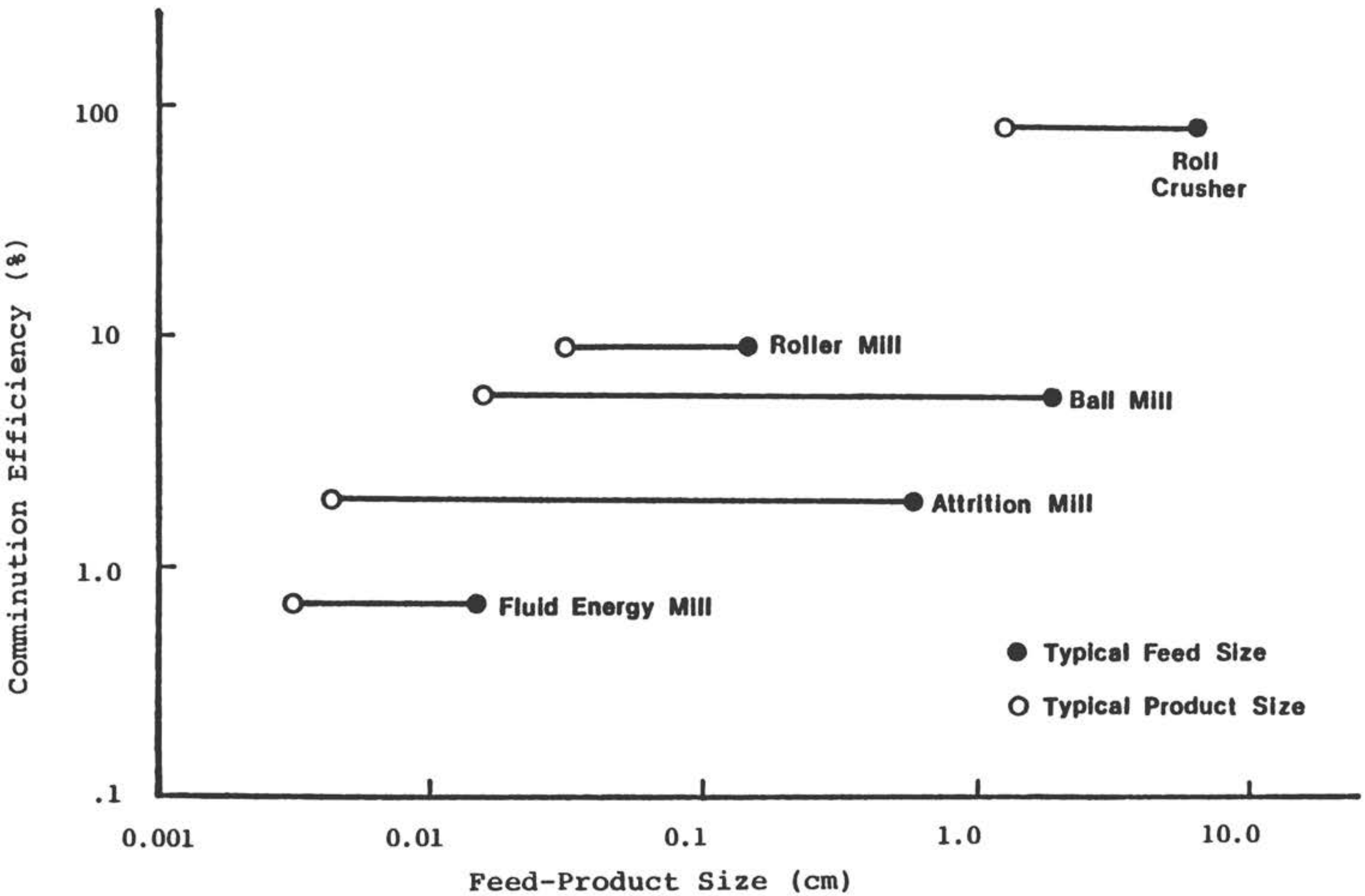

FIGURE 2-4 Efficiency of Comminution Devices. (After Stairmand, 1963)

TABLE 2-6: Equivalent Energy in the Steel

Device Type	Feed to Product Size Range (cm)	Size Reduction Energy (kWh/ton)	Liner Consumption (lb/ton)	Media Consumption (lb/ton)	Energy Content of Steel (kWh/ton)
Crushers	100-2	0.8	0.03	---	0.05
Autogenous Grinding	15.0-0.01	20.2	0.39	---	0.67
Rod Mill	5-0.5	1.3	0.03	0.34	0.63
Ball Mill	1.0-0.01	12.4	0.20	2.61	4.78

Crusher
  liners, lb/kWh = $0.002(40\ Ai + 9)$
Autogenous Mill
  liners, lb/kWh = $0.031(Ai - 0.015)^{0.30}$
Rod Mill (wet)
  liners, lb/kWh = $0.035(Ai - 0.015)^{0.30}$
  rods, lb/kWh = $0.35(Ai - 0.020)^{0.20}$
Ball Mill (wet)
  liners, lb/kWh = $0.026(Ai - 0.015)^{0.30}$
  balls, lb/kWh = $0.35(Ai - 0.015)^{0.33}$

Source: Bond, F. C., Eng. Min. J., p. 169, 1964

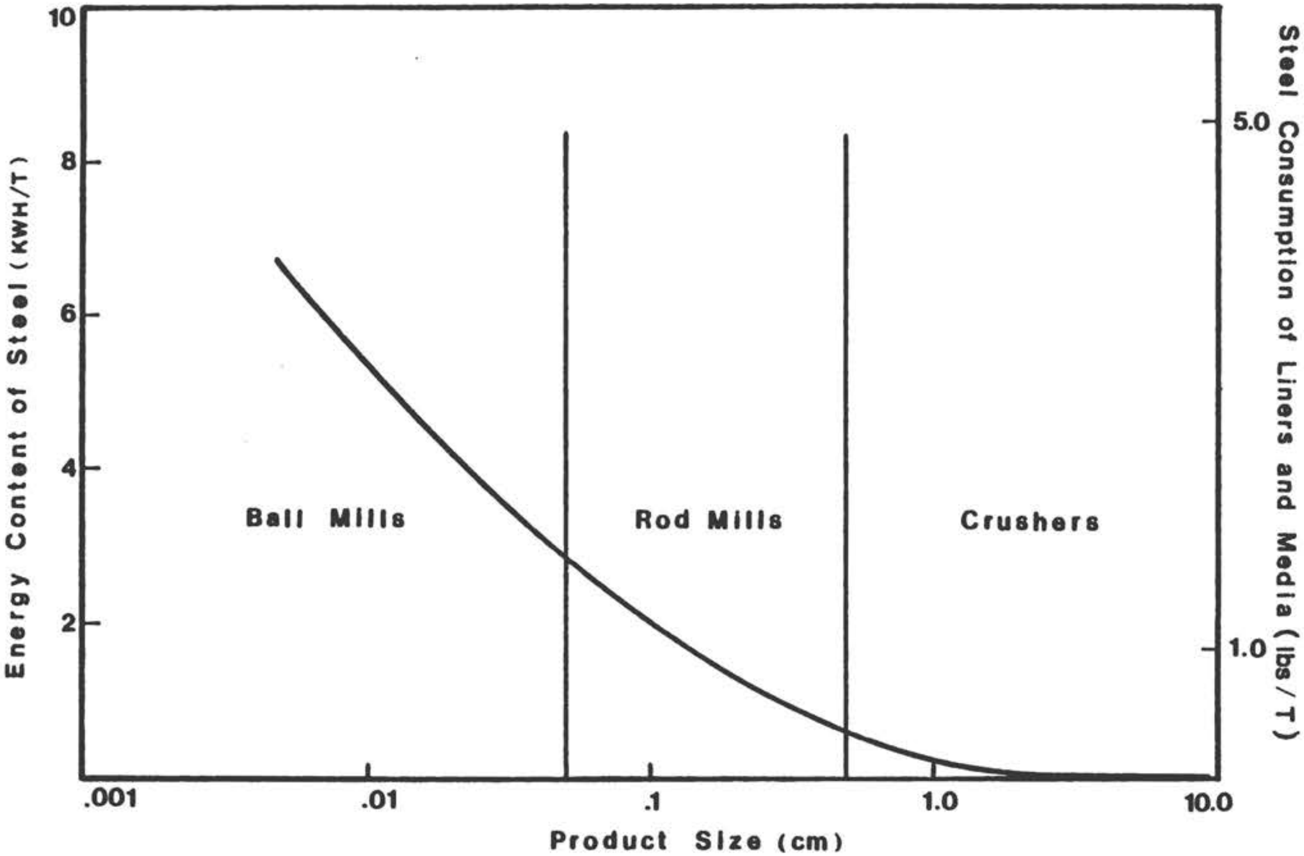

FIGURE 2-5 The consumption of steel by wear of liners and media and the energy content of that steel for 3 devices.

2.4.2 DISCUSSION

The energy requirements in Table 2-5 show that devices producing very fine product particle sizes--i.e., 0.0001 cm and below--require very high specific energy inputs, more than 200 kWh/T. Fine grinding devices producing a product 0.0001 cm and larger require specific energy inputs of 10 to 30 kWh/ton. Devices performing an intermediate grinding task require less than 10 kWh/T to produce product sizes greater than 0.01 cm. Product sizes above 0.1 cm are produced in coarse size reduction devices with specific energy inputs less than 1 kWh/ton.

The range in specific energy consumption in some devices may be as much as an order of magnitude. This range is caused by variations in material hardness, operating conditions, and circuit configuration, e.g., closed circuit or open circuit. Open circuit grinding devices require higher energy inputs to achieve the same throughput and particle size as closed circuit devices with a classifier.

An important factor to consider in evaluating a task is the reduction ratio achieved by the device. Coarse size reduction devices typically operate with reduction ratios on the order of 3 or 4 to 1. A reduction ratio of this magnitude can result in energy inputs of less than 1 kWh/ton. However, when the reduction ratio is increased to 10 to 1 or more, energy inputs of 5 kWh/ton or larger are encountered. Similarly, a ball mill operated at the usual reduction ratio of around 100 will consume about 14 kWh/ton for an average material--that is, a material that is not unusually hard or soft. A ball mill operated at reduction ratios greater than 100 requires high specific energy inputs. The range of specific energies for the attritor is 35-800 kWh/ton. The low figure of 35 kWh/ton would be that used to achieve a 0.001 cm size, while 800 kWh/ton is necessary to reduce the product to 0.0001 cm. the 20-fold increase in energy input to achieve a particle size 10 times smaller is indicative of the exponential nature of the energy-size relationship discussed in Section 2.3.

As shown in Table 2-5 the efficiency of comminution devices drops off dramatically as product size decreases. The efficiency of fine grinding devices such as ball mills has been considered to be 1% or less on the basis of new surface area produced. However, the efficiency of the ball mill is reported to be 5% (Stairmand, 1963) when efficiency is defined in the context of the free crushing concept. By contrast, the efficiency of coarse size reduction devices is high, as shown by the 79% efficiency reported for the roll crusher. Efficiencies of various comminution devices as determined by Stairmand are shown in Figure 2-4.

The environment in which grinding takes place determines to a large degree the efficiency of the process (Stairmand, 1963). In devices such as jaw, gyratory, and roll crushers there are few particle-to-particle interactions. The absence of these interactions, called free crushing, increases the compressive stresses that the device can be made to exert directly on the particles to be broken. Devices such as impact and tumbling types operate under packed and choked conditions where particle-to-particle interactions occur in large numbers. Also in such devices, the media, such as the balls and rods in tumbling mills, may contact each other several times before capturing a particle and breaking it. These media-to-media and media-to-liner collisions that occur without particle breakage decrease the efficiency of the device.

Figure 2-3 illustrates the increases in specific energy inputs as the product size decreases for the devices tabulated in Table 2-5.

In Table 2-6 it is shown that the energy content of the steel media and liners consumed in rod and ball mills is a sizable fraction of the energy consumed by comminution. The low rate of wear in crushers and autogenous mills is due to the absence of steel media. Hence the consumption of energy due to steel wear in these devices is a fraction of the energy consumed by comminution. The rate of steel consumption increases as product size produced by the device decreases (Figure 2-5).

The differences of wear rates in these devices result from the way material is broken. The liners in crushers do not contact each other so that wear results from particle-liner interaction. When media are employed, media-liner contact becomes important. Extensive wear of liners in ball mills--0.2 lb/ton of material ground--results from the numerous ball-liner collisions. However, the rate of wear in rod mills is the same as in crushers, 0.03 lb/ton. This is due to the relatively low specific comminution energy of rod mills when compared to ball mills. The importance of media in liner wear is further demonstrated by the large liner wear rate 0.4 lb/ton, in autogenous mills. The autogenous mill may employ no media or natural media such as pebbles. However, the many collisions between particles and liners cause a liner wear rate in autogenous mills that is twice that in ball mills.

The consumption of steel media in ball mills, 2.9 lb/ton, is an order of magnitude greater than in rod mills. As occurs in the wear of liners, the many media-media collisions in ball mills contributes heavily to media wear.

In is interesting to note that media-media and media-liner interactions are a factor in both the high rate of wear and low efficiency of the ball mill.

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CHAPTER III

THEORETICAL CONSIDERATIONS

3.1 Introduction

As is to be expected with a subject as important to the process industries as comminution, there has been a considerable effort over the years to quantify size reduction behavior, to elucidate the fundamental mechanisms involved in the fracture process, and to develop the pertinent theory. Over the years, several methods of quantifying the degree of size reduction experimentally have been developed and used in various industries. The methods of characterizing particles are summarized in Section 3.2.

To some extent, progress in the development of theories of comminution was hindered by early work which focused on the theoretical energy required to generate the new surface of the fragmented material. This work is reviewed in Section 3.3.

In view of the inability of this fundamental approach to offer industry the tools required for effective design and operating guidelines, there was a period during which empirical models were formulated and used extensively for design and scale-up. These energy-size reduction relationships are discussed in Section 3.3.1.

In the late 1950's and early 1960's a new approach to the fundamental of comminution was developed. It was clear that there was little future in the concept of integrating the fundamental behavior of individual particles into realistic representations of actual comminution devices. Moreover, the purely empirical approach offered no insight into the basic mechanisms involved. In seeking more realistic representations, a process analysis approach was taken and led to the establishment of the kinetic or population balance models. These are the basic forms of mathematical representation of comminution processes that are used and are still being developed today. This approach is reviewed in Section 3.2.3.

There are innumerable ways of moving particles into breakage zones and applying breakage forces, and each has particular results and applications. Section 3.4 deals with these concepts. Section 3.5. deals with the dynamics of comminution devices, and the final section of the chapter, Section 3.5, reviews the many different ways of defining efficiency in relation to fragmentation and comminution.

3.2 Particle Characterization

The breakage behavior of particles in a comminution device depends on their size, shape, and homogeneity as well as on other device-specific variables such as manner of loading, the intensity and velocity of loading, and the temperature and composition of the surrounding medium. Hence a fundamental understanding of the size, shape, and state of aggregation of the particles should aid in the selection of a device that maximizes energy utilization for size reduction.

In the mineral industry the size and shape of comminuted products greatly influence subsequent separation operations. In particular, the liberation of metal-bearing grains from the matrix of waste rock or gangue during size reduction is critical to achieving good separation in a flotation circuit. In current practice, regrinding is often employed after flotation to release grains from locked particles in flotation concentrates or tailings. By proper control of liberation during comminution, efficient separation of valuable metal can be achieved.

Similarly, particle properties play an important role in the production and utilization of cement, coal, and food products. In cement manufacturing, the specific surface area of ground clinker determines cement quality. The combustion characteristics of fine coal powder depend on the size of the coal particles. Furthermore, the degree of pyrite liberation from coal particles, and subsequent coal cleaning steps, are affected by particle size. In the food industry the particle size and texture of milled wheat flour is important in the production of cereals and bread.

This section briefly summarizes the current state of knowledge in particulate characterization and discusses details as applicable to each of the industries mentioned above.

3.2.1 PARTICLE SIZE

The term "size of a particle" (Allen, 1975; Irani and Callis, 1963; Beddow, 1980) is somewhat ambiguous since particles generally are irregularly shaped. Particle size is usually characterized by a linear dimension, such as particle "diameter," or other representative measure of spatial extent such as particle area or volume. Often conversions or shape factors are defined to allow transformation from one "size" to another. Because of the irregular shapes of particles a statistical average of a large number of measurements is assumed to represent their size. Sizes of comminuted products important in the process industries range from more than 100 cm to less than 10^{-5} cm. This wide range of sizes makes experimental characterization difficult.

Practical industrial systems usually involve an assembly of particles which is polydisperse rather than containing particles of a single size; hence the "size" of the assemblage is characterized by a size distribution, e.g., the percent by weight of particles retained in a size interval, with several intervals making up the entire size distribution (Herbst and Sepulveda, 1980). Figure 3-1 summarizes the techniques used to size comminution products today. Typical laboratory devices include microscopes, standard wire mesh sieves, microformed sieves, sedimentation devices, and automatic counting devices. Size analysis below 10^{-3} cm (10 μm) is difficult, and there is a definite need for simple, reliable, and inexpensive laboratory methods in the fine and ultrafine size ranges. The on-line devices used commercially are the Leeds and Northrup Microtrac analyzer, which uses a laser beam diffraction principle; the Autometrics unit, which uses attenuation of ultrasonic waves; and the HIAC unit, which uses a light obscuration principle. Each of these units is capable of measuring within a specified size range only. Other particle properties, such as surface area, are measured by nitrogen or helium adsorption, or permeability determination. No true on-line devices are available commercially for such measurements.

3.2.2. PARTICLE SHAPE

The use of a linear dimension to represent particle size may be deceptive in that two particles could have the same size but look completely different. Thus, in some instances particle shape must be used in conjunction with size (Heywood 1947). The simplest measure of shape is a quantity known as shape factor which relates particle volume or particle area to the linear dimension. The concept of a shape factor comes from the notion that for a regularly shaped particle the surface area is proportional to a linear dimension squared and the volume is proportional to a linear dimension cubed. On this basis, the surface area, a, and the volume, v, of a particle are related to a characteristic linear dimension, d, as follows:

$$a = C_2 d^2 \quad ; \quad C_2 = \text{surface area shape factor}$$

$$v = C_3 d^3 \quad ; \quad C_3 = \text{volume shape factor}$$

In general, shape for a given material depends on its size. Figure 3-2 shows how the ratio of the area to the volume shape factor, C_2/C_3, varies with particle diameter, d, for some commonly occurring minerals. Notice that for small sizes the "shape" of particles

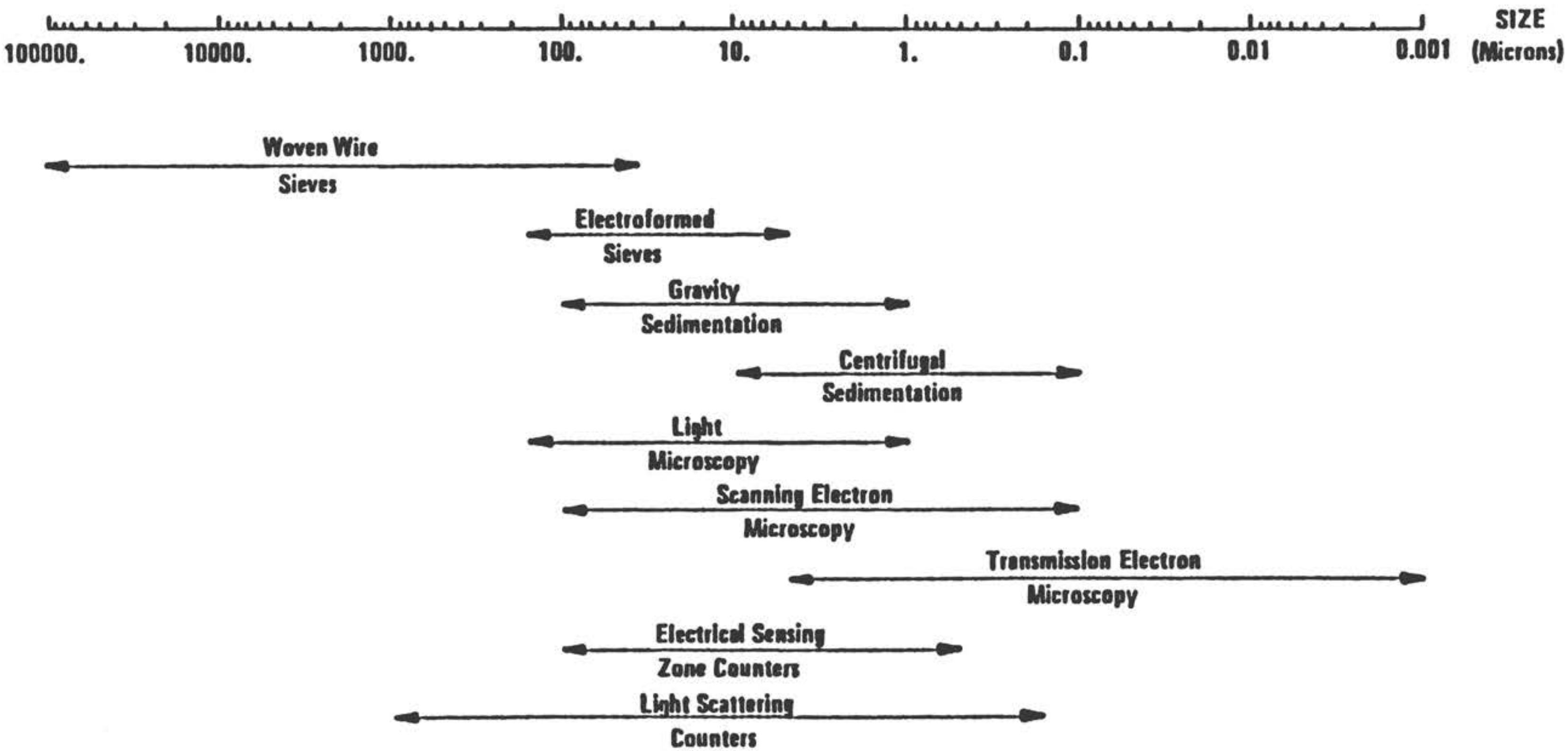


FIGURE 3-1 Normal range of application of various sizing techniques.

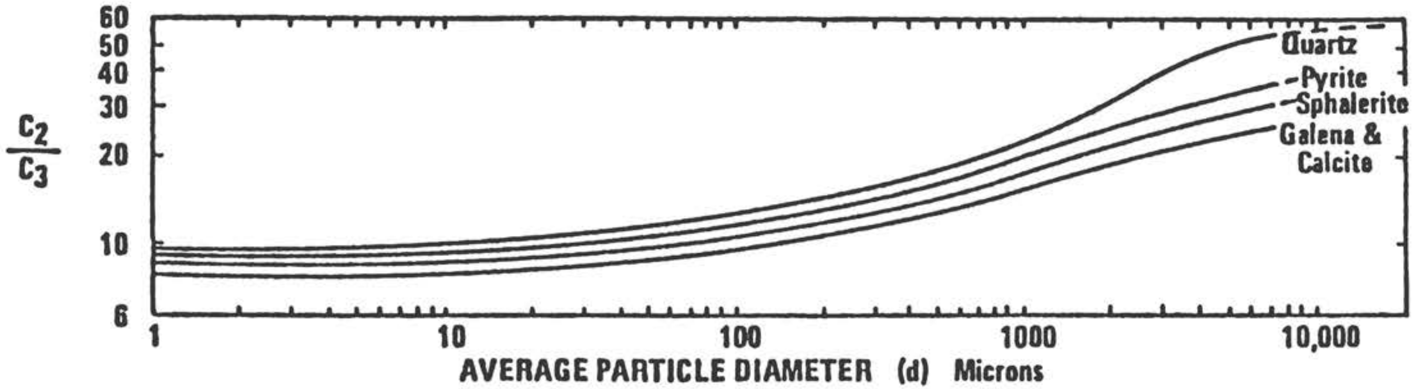


FIGURE 3-2 Ratio of area to volume shape factor, C_2/C_3, as a function of average particle diameter, d, for some commonly occurring minerals. (Herbst and Sepulveda, 1975)

becomes approximately constant, indicating the uniformity of grains in the fine size range.

In view of the importance of particle shape, more detailed analyses of the characterization of shape have been described recently by Meloy (1977), Davies (1978), and Beddow and Philip (1975). Briefly, these methods consist of mapping the particle outline, computing the center of gravity, CG, and setting up a polar coordinate system based on the radius vector from CG to the particle silhouette outline. Then the silhouette or shape outline is described by a Fourier type series, the coefficients of which are interpreted in terms of known geometric shapes.

By way of illustrating the usefulness of this type of information, Fong et al. (1979) have shown that the coefficients describing particle morphology change in a systematic way for a zinc particle undergoing reaction in an acid medium. This finding indicates that the acid first dissolves protrusions from the particle surface and then begins to attack weaker spots on the surface. These studies conceivably could lead to a better method of preparation of particles conducive to the reaction environment that they would be exposed to subsequently.

Rumpf (1965) studied single particle breakage from the point of view of size and shape of particles and the manner of loading, i.e., slow compression loading and impact loading. The particle resistance to compression and the distribution of fragments were measured. Rumpf contends that a more scientific basis for comminution technology could be developed from a consideration of these parameters. Unfortunately, inexpensive and easy-to-use shape analysis methods for particulate materials are not yet available.

3.2.3 CHARACTERIZATION METHODS USED IN SPECIFIC INDUSTRIES

As observed previously, the objectives of comminution for the various process industries vary considerably from industry to industry. Similarly, the methods of characterization vary considerably from industry to industry. The objectives and principal characterization methods for the major process industries are given herein.

3.2.3.1 Minerals Industry

Gaudin (1926) began the investigation of mineral liberation phenomena, in which a particular mineral grain of interest is released from a solid matrix of other minerals as would occur normally in metallic ore bodies. The primary purpose of comminution in a tumbling mill is to liberate valuable mineral grains from gangue. The size of the particles in the mill product determines the extent of liberation. Usually the coarse particles contain small inclusions of valuable mineral in a fairly homogeneous matrix of gangue, the fine particles approach a mixture of pure mineral grains and gangue, and the intermediate particles exhibit extreme variations in mineral-to-gangue ratio. Since liberation cannot easily be measured, the usual practice in the mineral industry involves controlling product size in crushing and grinding circuits. Figure 3-3 shows typical size ranges of products for unit operations in mineral processing. In some instances particles differing in size by four or five orders of magnitude are found in the same product. Analysis is difficult in these cases, particularly in the fine size range.

The behavior of multiphase particles in mineral separation processes such as flotation and leaching depends on the size and shape of valuable mineral grains in the larger composite grain matrix and of the completely liberated valuable mineral grains themselves. The commonly used size measurement devices are incapable of measuring the size of mineral grains imbedded in a solid rock matrix. As a result, several investigators (Jones and Barbery, 1975; Barbery and Huyet, 1977; King, 1979) have approached the problem by postulating the manner in which a grain occurs in the solid matrix and making suitable measurements. these measurements fall into two classes:

- o Area measurement;
- o Line measurement;

The area-measuring instruments use television scanning techniques to discriminate among the various minerals of interest by optical brightness. This value is either the optical reflectivity or the optical transmissivity of the mineral surface. An area-measurement instrument makes the necessary measurement very rapidly and can provide the first size moments of the volumetric particle size distribution.

The line-measuring devices operate more slowly than the area-measuring devices, but provide a greatly improved phase discriminating capability. Because of the loss of one "order" of information, it is only possible to justify the calculation of the first three moments of the particle size distribution from linear data.

A typical linear-measurement device is the Geoscan electron probe X-ray microanalyzer. In this device, the specimen is moved past a stationary electron beam by computer-controlled stepper motors, and the mineral phases in the specimen are distinguished by the X-ray signals that are produced. The lengths of the intercepts made across a selected phase are then measured and stored in computer memory from which the following quantities are calculated using stereological methods:

1) Volume fraction of the selected mineral

2a) Shape of that mineral

2b) Volumetric size distribution of the mineral

2) Specific surface area of the mineral

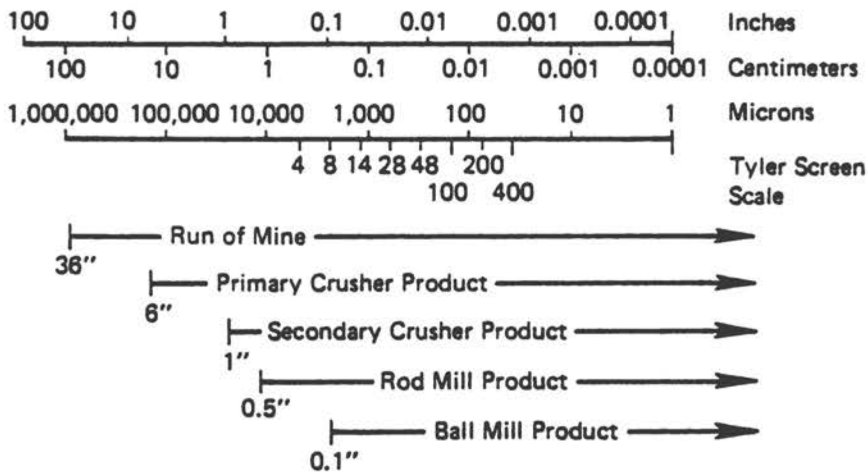


FIGURE 3-3 Typical size ranges for mineral processing products.

Practical use of this information would include determination of liberation characteristics of ores during flowsheet development and, possibly, on-line measurement for automatic control.

3.2.3.2 Cement Industry

In the cement industry, the surface area per unit weight has become a standard for characterizing cement powder. Acceptable fineness is around 3200-4200 cm^2/gm of cement. This measurement, known as a Blaine surface measurement, is made by measuring the pressure drop that results from the flow of air through a standard packed bed of cement. The quality of the cement for construction is characterized by standard tests for (a) bleeding, (b) workability, (c) degree of hydration, (d) drying shrinkage, and (e) setting strength, of which the strength is important. Strength is commonly measured in pounds per square inch of compressive load withstood by a 10-in. cube or a 1-in. diameter by 2-in. long cylinder of setting cement, after allowing a certain time for setting. The 28-day strength of cement is around 5000-7500 psi. Until now, the setting strength of cement has been attributed solely to the Blaine area of cement powder, with a Blaine area of 3600 cm^2/gm giving rise to strengths acceptable in the construction industry. However, a recent study (ERDA, 1979) has shown that the particle size of cement is important, based on the following findings:

1) By controlling cement particle size to below 20 microns, with a Blaine area of only 2600 cm^2/gm, strengths equalling that of normally ground cements of 3600 cm^2/gm Blaine area can be achieved.

2) The amount of ground clinker in the 2.5 micron particle size range has large effects on bleeding, water requirement for flow, and strength development.

3) Controlled product particle size in cement grinding results in cements of as high or higher strengths at ages from 1 to 60 days at Blaine areas of 450 to 800 cm^2/gm, substantially lower than the normal grinds of the same composition.

It is estimated that the adoption of particle size control in clinker grinding by the entire U.S. cement industry would result in a 27% saving in grinding energy and an 8.5% saving in kiln fuel. To achieve such control, however, reliable, on-line (real time) surface area measurement devices need to be developed.

3.2.3.3 Coal/Utilities Industry

Most utility coal is pulverized and then burned to generate steam in high capacity (more than 100,000 lbs of steam per hour). The size of boiler feed coal is 65 to 80% passing 74 microns. Coal for combustion is characterized by measurement of (1) ignition point, (2) flammable limits, (3) ignition lag, (4) flame stability, and (5) flame velocity. In addition, the composition of coal determines its use as boiler feed as shown below:

	Moisture	Fixed Carbon	Volatile Matter	Calorific Value Btu/lb
Anthracite	2.8-16.3	80.5-85.7	3.2-11.5	11,480-15,010
Bituminous	2.2-16.0	44.9-78.2	18.7-39.2	11,960-15,240
Subbituminous	17.5-27.0	40.5-46.4	32.6-36.1	9,180-11,050
Lignite	39.1	31.4	29.5	7,440

The moisture content of the feed coal before pulverization can be up to 25%, but drying with hot air during pulverization can decrease moisture to less than 1%. In general, for bulk handling and processing purposes coarse coal is characterized by (1) free swelling index, (2) size stability, (3) bulk density, and (4) grindability index.

To abate pollution problems originating from coal combustion, the federal government has established standards requiring that power plants larger than 25/MW emit no more than 1.2 lb of sulfur dioxide, 1 lb of particulate matter (ash), and 0.7 lb of nitrogen oxides per 10^6 Btu of heat generated. To satisfy this standard, the maximum allowable sulfur content would be about 0.8% for eastern coals and 0.6% for western coals. Only a small number of mines can meet this standard without coal cleaning. Physical removal of sulfur and ash before combustion is common practice today.

3.2.3.4 Food Industry

Comminution is employed to mill wheat, corn, rye, and durum in cereal production technology (Matz, 1970). Milling is done to release the endosperm from the grain and to reduce the endosperm size. Each class of cereal flours has its own peculiar specifications, but there are specifications common to all. These more general tests are moisture, protein, ash, color, fiber, particle, size, and fat. The specifications depend on the product, such as cookie, pastry, cake, cracker, etc.

Standard tests have been developed to determine the moisture, protein, ash, fiber, and fat content of flours. The property particle size, which is common to other industries, is measured by sieve analysis, microscopic methods, sedimentation techniques, and air permeation techniques. Particle size is specified in Fisher units; a 12-20 Fisher unit is satisfactory for most cereal processes.

3.2.4 RECOMMENDATIONS

The acceptance of uniform standards for particle characterization across industry boundaries would simplify comparisons greatly and may allow the development of characterization equipment that can serve several industries. Research needs exist for the following recommended hardware developments:

- Rapid and inexpensive size analysis for ultrafine particles (1 μm to 0.01 μm)

- Rapid shape analysis for samples containing large numbers of particles

- Rapid and inexpensive characterization of liberation in the laboratory

- On-line measurement of liberation

- On-line measurement of surface areas

3.3 Fragmentation Science

3.3.1 INTRODUCTION

The comminution of mineral ores (e.g., iron, copper), coals, cement raw mix blends, cement clinkers, etc., involves assemblies of millions of particles. Each of these millions of particles, when broken, generates multitudes more of finer particles, so that untold numbers of particles are involved in each step of the comminution process. However, in a succession of comminution events, many similarities can be expected, especially when considering the fracture of specimens of equal size, or when comparing measures of comminution events in terms of particle dimensions prior to fracture.

Comminution events can be categorized into two main modes. In the major mode, the particle is subjected to compressive stresses. If the compressive stress at some point within the particle exceeds the strength of the particle, it may break into several major portions plus associated fine particles. In the minor mode, the particle is rubbed or jostled so that stress concentrations exist at some surface sites on corners or protrusions; these will then be torn off as the particle is abraded into a smooth shape like a boulder or pebble.

Relatively little research has gone into the abrasive wearing of rock, probably because it appears to be a slower and less efficient means of grinding. The size distribution of an abraded material consists of worn-down particles from the original feed plus fine grains as large as the natural grain size of the material and finer, with only a few or no particles of intermediate size. One group of investigators (Crabtree et al., 1964; Kinasevich et al., 1964) used a chipping abrasion model with the impact-compression model for the more predominant mode of breakage in comminution simulation; in general, however, comminution by abrasion is lumped into a general breakage distribution function.

3.3.2 STRESS-STRAIN CURVES

When a specimen is squeezed in a press, applied force can be plotted against deformation. In most metallurgical work or work with soil or concrete, the specimen has a regular shape such as a cube, prism, sphere, or cylinder. In such cases the stress corresponding to the applied force and the strain corresponding to the deformation are easily calculated. These calculations are useful because the materials tested are usually homogenous, and test results can be transposed from one specimen to the next.

Stress-strain curves for rocks--i.e., curves of stress versus axial strain (Figure 3-4)--have been interpreted [Bieniawski (1967)] to show what happens to the specimen from the initiation of loading until the specimen ruptures. Moreover, similar plots of stress versus lateral strain and versus volumetric strain are often of interest. In particular, the stress versus volumetric strain curve may show an expansion during the later stages of loading.

The stress-strain curve is also related to measures of cracks within the specimen. During the loading process, cracks of the type defined by Griffith (1921) form and grow until final rupture occurs. There are two measures of importance: the size of the crack, more commonly referred to as crack length, and the crack velocity.

Bieniawski (1967) showed how the various portions of the crack velocity vs. crack length curve relate to the various regimes of the stress-strain curves (Figure 3-4). Toward the end of the loading process, the crack velocity approaches a terminal velocity which is related to the material's modulus of elasticity and specific gravity. Poncelet (1946, 1963) showed that the terminal velocity was $0.5\sqrt{\text{shear modulus/density}}$, whereas later investigators (Roberts and Wells, 1954) arrived at $0.38\sqrt{\text{modulus of elasticity/density}}$. The results are virtually the same. Computer simulations of crack growth are now being examined (Paskin et al., 1980) and indicate that the Griffith criterion is not necessarily valid at the atomic level.

3.3.3 IRREGULAR SPECIMENS

The specimens that are crushed and ground in industrial equipment are not spheres, cubes, prisms, or cylinders nor are they generally smooth pebbles or boulders. Rather they are irregular fragments of a variety of shapes. Tests on specimens of these irregular shapes have generally involved specimens selected to be within a narrow size range as determined by sieves of size ratio $\sqrt{2}$.

Tests on irregular specimens have been of two related varieties. In one approach, each particle is loaded in compression until it fails and the load at fracture is reported (Bergstrom et al., 1963; Yashima and Saito, (1972). Some experimenters have simply reported fracture load (Bergstrom et al., 1963), whereas others have attempted to calculate a stress from the fracture load and some other measures (Schonert, 1971). In one such calculation the fracture load is divided by the square of the initial (or final) distance between press platens

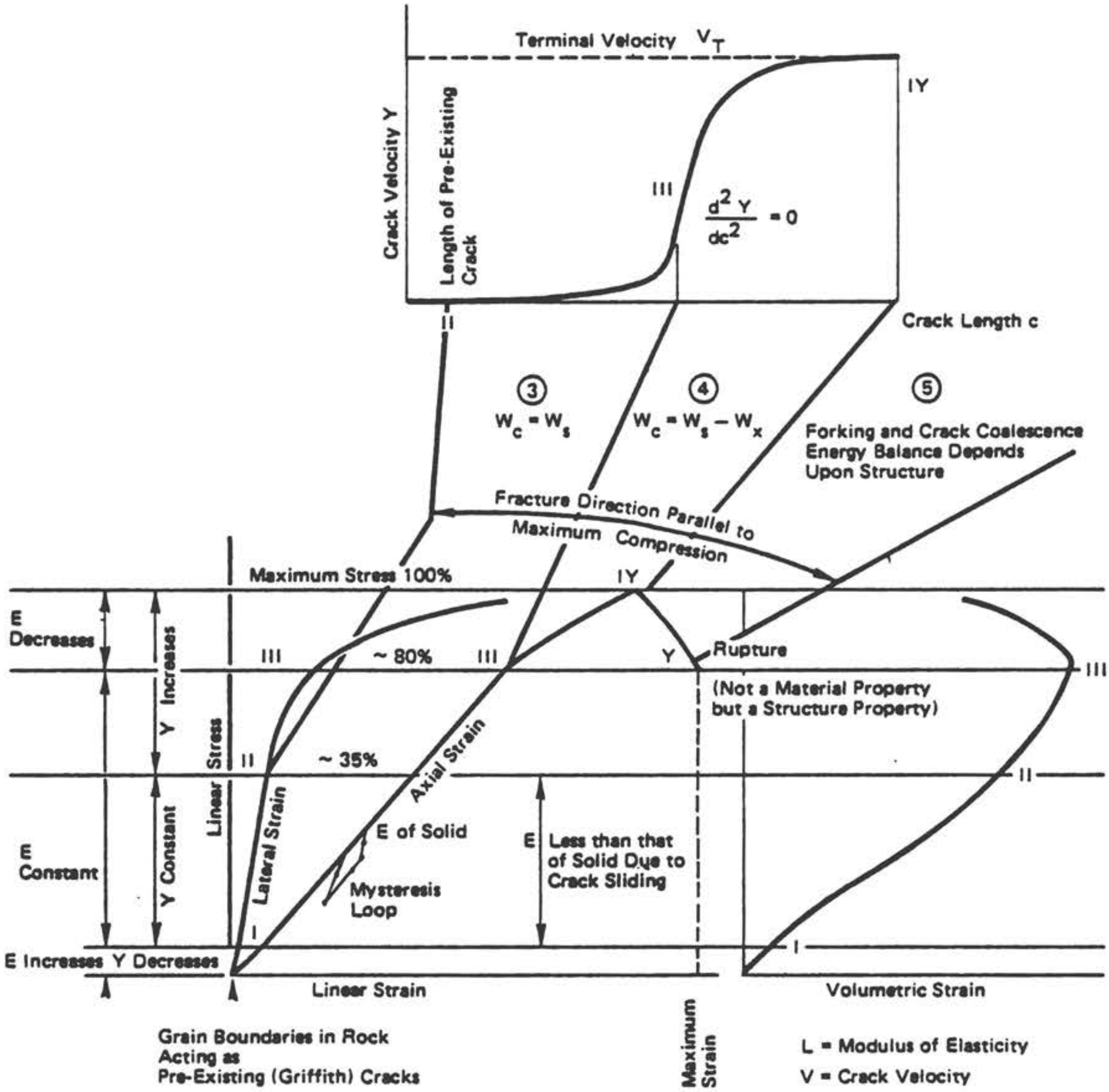


FIGURE 3-4 Stress-strain relationships and mechanism of brittle failure of rock. (After Bieniawski, 1967).

(Hiramatsu and Oka, 1966). Another method uses for this distance the calculated diameter of a sphere with equivalent mass (Bergstrom et al., 1963). Sometimes empirical coefficients are introduced to relate the calculated stress more closely to values obtained using standard shaped specimens (Hiramatsu and Oka, 1966).

Tests have also been made by subjecting the specimen to a known energy input and observing the proportion of specimens that fracture under this energy input. As the energy input increases, a higher proportion of particles break. Schonert (1971) and, more recently, Krogh (1978) have made this kind of observation (Figure 3-5).

The energy may be applied by different techniques. A weight dropped from a known height onto a specimen is one manner of obtaining a known energy input. A specimen of known mass and velocity can be made to strike a stationary target. The latter case has been called a shatter test and has been used for investigating the crushing characteristics of coal. A related test is the number of 450-mm (18-in.) drops a "green pellet" will sustain. This test, common in iron ore pelletizing technology, is a measure of the number of conveying transfers that the freshly formed pellet can be expected to undergo before fracturing.

3.3.4 SPECIMEN SIZE EFFECT

The force required to fracture specimens of different sizes can be related to specimen size. It has been found that the larger the specimen, the greater the required fracture force. If the specimen were homogenous in strength, thereby eliminating size effects, it would be expected that the required fracture load would vary as the square of the specimen size or as the 2/3 power of the specimen mass (Bergstrom, 1963). However, many studies show that fracture load varies to a power of specimen mass which is less than 2/3. Indeed, an exponent of 0.5 is often observed. This can be shown to be the case also in the optimum ball size equation of Bond (1958). Weibull (1939) introduced a modulus of uniformity which measures the departure of strength from that for a homogenous substance as a function of specimen volume.

There is some question as to whether the fracture load-specimen mass relationship should be described by a single function. Yashima (1972) describes the relationship by a set of connected straight-line segments and calculates the slope of each, and Krogh (1978) did this also for jasper.

Unless there is some apparent justification for a break in the curve, such as a natural grain size at the juncture, or a change in particle shape characteristics at that size, it would seem more appropriate to use only a single size effect function throughout.

The extent of the range of a single size effect is extremely wide. For example, Millard et al. (1955) showed that the same size effect relation applies on a sample of coal from a specimen weight of 10^{-9} g to 1.2 kg (30 microns to 3 in.).

It has been shown (Bergstrom, 1963) that data on coal cubes (Gaddy, 1956) from 50 mm (2 in.) up to 1.5 m (5 ft) in size can be assembled in

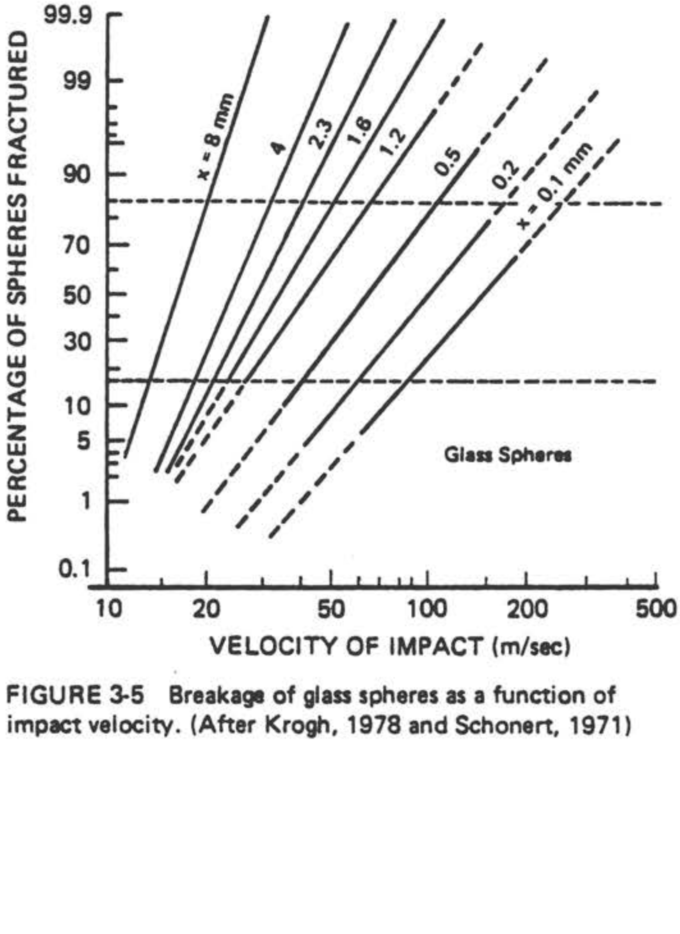


FIGURE 3-5 Breakage of glass spheres as a function of impact velocity. (After Krogh, 1978 and Schonert, 1971)

the same manner. Bergstrom (1963) has published the fracture load data shown in Figures 3-6, 3-7, and 3-8 on quartz, taconite, and limestone for which some specimens were as small as 325 mesh and others as large as 50 mm (2 in.).

In compression tests of specimens of a single size fraction, the varying shapes of successive specimens that are broken result in various contact geometries, which result in turn in various stress concentrations and hence a fairly wide range of fracture loads. The range can often be as great as an order of magnitude. Therefore, a fairly large number of specimens ought to be broken so that the calculated average load will be a valid average. Moreover, the average of specimen weights ought to be used rather than a calculated weight of a sphere of a diameter corresponding to some mid-size of the sieve fraction.

The distribution of fracture loads of each narrow sieve fraction can be described by several means, including simply a statement that it is nearly normal, but skewed slightly toward the smaller loads (Austin et al., 1973). It is convenient to plot fracture load on a log axis, where for n specimens the probability value is the rank divided by n + 1. The resulting curve may be approximated by a straight line, i.e., yielding a log-normal distribution, but the data may also describe a curve that can be fitted by a special case of the Weibull distribution (Stanley and Newton, 1977).

3.3.5 ENERGY BALANCE FOR SINGLE PARTICLES

Various investigators (Rumpf, 1965; Schonert, 1971) have attempted to account for the dissipation of all the energy input to a specimen as it is loaded to fracture. The resulting energy can take the form of kinetic energy, sound energy, potential strain energy, occasionally light energy, thermal energy, and surface energy of the new surfaces generated. Of these varieties of energy, only the increase in surface energy has a direct relation to the objective of the comminution event, namely, the reduction in particle size, which can be measured in terms of an increase in specific surface if the surface is not deformed by plastic flow. A number of investigators have concluded that the new surface energy accounts only for about 1% of the energy input required for industrial grinding. With such a small percentage, it is difficult to see how specific surface measurements can be effectively related to the specific energy measurements for comminution. Nevertheless, good correlations of specific surface and specific energy have been observed (Hahn, 1957; Zeleny, 1957; Kenney, 1957; Bergstrom, 1963).

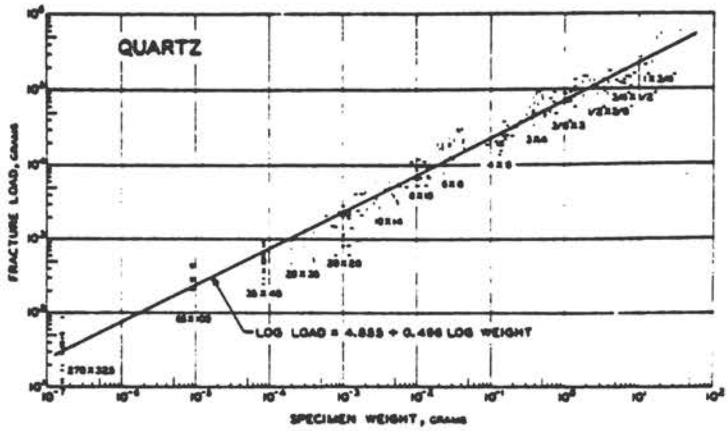
3.3.6 ENERGY CONSUMPTION INVESTIGATIONS

3.3.6.1 Historical Background

The first significant theoretical approach to energy consumed in comminution was that of von Rittinger in 1867. He proposed that the

FIGURE 3-6:

Fracture load vs. specimen weight for quartz. Up to twenty specimens were tested for each mesh fraction. All the specimens of any particular mesh fraction are identified by - or ⁞. Average weights are used for the smaller mesh fractions. (Bergstrom, et al., 1963)



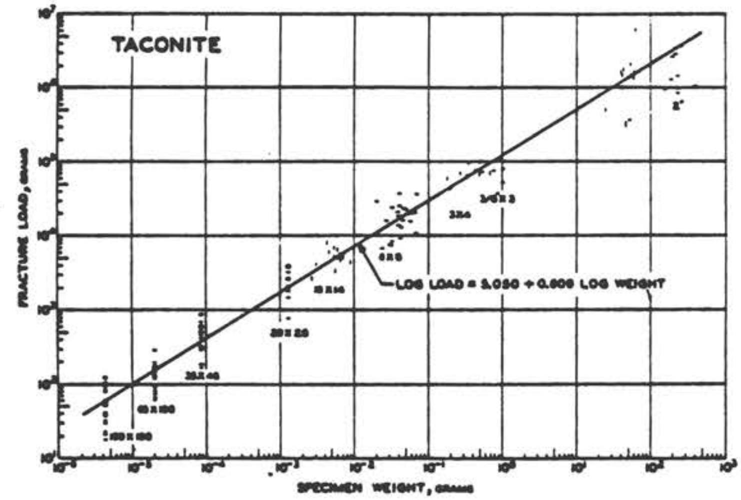
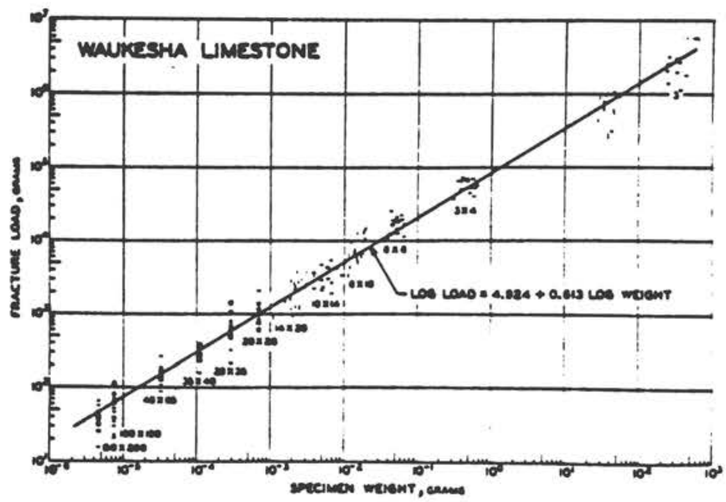


FIGURE 3-7:

Fracture load vs. specimen weight for taconite. (Bergstrom, et al., 1963)

FIGURE 3-8:

Fracture load vs. specimen weight for Waukesha limestone. (Bergstrom, et al., 1963)



energy consumed in size reduction was proportional to the area of new surface produced:

ΔA, or

Energy = $K\Delta A$

where K is a proportionality constant. This relationship can be formulated for experiments in the following manner. If D represents the average diameter of a particle, the average surface area is proportional to D^2 and the average volume is proportional to D^3. For a fixed mass of material the average number of particles is proportional to the mass divided by the average volume per particle. The above assumption for a given mass can be written as:

Energy = K (final surface area - initial surface area) 3.3-1

= K (final number of particles x average surface area - initial number of particles x average surface area)

$$= K\left(\frac{1}{D_f^3} \times D_f^2 - \frac{1}{D_i^3} \times D_i^2\right)$$

$$= K\left(\frac{1}{D_f} - \frac{1}{D_i}\right)$$

where D_i and D_f are the initial and final diameters. Experiments (Lowrison, 1974) have shown that this relationship is valid over a limited range of energy input both by ball milling and ball drop tests. The tests were on brittle materials such as sand, calcite, fluorite, etc.

It was realized long before the Griffith crack theory that fracture of a material was caused by internal microcracks which propagated when the strain reached a certain critical level. Thus, energy has to be supplied to a body to produce the required level of strain within the body to propagate the crack, and the strain energy of a body is proportional to its volume. On this basis, Kick (1885) proposed a different relationship of energy and size reduction. With a group of particles of similar size, a certain amount of energy, proportional to their volume (D^3), must be applied to distort them up to the point immediately before crack propagation. Therefore, most of the energy required for comminution would be to increase the strain energy and this would have to be done each time the particles were fractured. Thus, the energy required to reduce particles of average initial diameter, D, to an average final diameter would be proportional to the number of times, n, that the particles were deformed. With some assumptions, these ideas lead to a relationship quite different from that of von Rittinger. In this model:

total deformation energy 3.3-2

= K_1 x number of steps of size reduction

= $K_1 n$

where K_1 is a constant. It was further assumed by Kick that each step of volume reduction has the same ratio, r, of initial to final volume, e.g.,

$$\left(\frac{D_0}{D_1}\right)^3 = r, \left(\frac{D_1}{D_2}\right)^3 = r, \text{ etc.} \qquad 3.3\text{-}3$$

where D_0 is the initial diameter and D_1 and D_2 the resulting diameters of the first and second stages of reduction. Thus, the product of two steps yields an exponent of r equal to the number of steps, for example:

$$\left(\frac{D_0}{D_1}\right)^3 \left(\frac{D_1}{D_2}\right)^3 = r^2 \qquad 3.3\text{-}4$$

Cancellation results in the relation:

$$\left(\frac{D_0}{D_2}\right)^3 = r^2 \qquad 3.3\text{-}5$$

It is seen that for a number of steps, n, we may write:

$$\left(\frac{D_0}{D_n}\right)^3 = r^n \qquad 3.3\text{-}6$$

Solving for n yields:

$$n = \frac{3}{\log r} \log\left(\frac{D_0}{D_n}\right) = K_2 \log\left(\frac{D_0}{D_n}\right) \qquad 3.3\text{-}7$$

and substituting for n in the expression for energy yields:

$$\text{total deformation energy} = K_1 K_2 \log\left(\frac{D_0}{D_n}\right) \qquad 3.3\text{-}8$$

$$= K \log\left(\frac{D_0}{D_n}\right)$$

where the product of the constants K_1 and K_2 has been replaced by a single constant, K.

There are obvious difficulties with both the Kick and von Rittinger models in any real situation. Kick assumes identical deformation ratios, r, which might conceivably be satisfied in an idealized experiment in which all particles have the same size, but certainly not in practical comminution. Nevertheless, there must exist some function of the strain energy in comminution. There are two main difficulties with the von Rittinger model. A real crack propagates at high velocity and, except on the cleavage plane of a single crystal, forks and branches in its course so that many fragments of different sizes are usually created. Thus the final diameter, D_f, can be specified only as a size distribution function and this function varies with the material.

Modern experiments have shown a second difficulty with the von Rittinger model. Parts of the fracture surface may be distorted or even glassy. This means that energy has been put into plastic flow, so the surface free energy is different from that assumed by von Rittinger.

3.3.6.2 Elastic Energy Contribution

An important factor not appreciated by the early workers is that half of the work of crack propagation goes into elastic strain and the remaining half may or may not be new free surface energy. This is easily shown in the following way.

Three energies must be accounted for when a crack spreads in an elastic solid:

1. External work, which increases the crack size

2. Surface energy of the crack, which opposes its increase

3. Elastic energy of the body, which can assist or oppose according to the type of loading

The two extreme types of loading are:

(a) Constant force

(b) Constant displacement

Both of these lead to the same result.

The proof of this theorem will be shown for case (a), constant force.

Take the force direction to be that of Figure 3-9(a), where the ellipse represents the crack. Since the body is elastic, Hooke's law applies and the force F is proportional to the elongation X. Thus, for a force F and elongation X, the point B in Figure 3-9(b) represents the mechanical state of the body just before the crack enlarges, and the area of triangle ABQ is the elastic work that has been done on the body to bring it to State B. This area is seen to be 1/2 (FX). If the loading is that of case (a), constant force, the body elongates by ΔX as one of the stretched atomic bonds at the tip of the crack is broken. The work done is the area BCRQ, which is $F\Delta X$, and the new elastic energy is the area of the triangle ACR or $1/2[F(X+\Delta X)]$. The change in elastic energy is the difference in the areas of the two triangles, ACR - ABQ or $1/2[F(X+\Delta X)] - 1/2[FX] = 1/2[F\Delta X]$. This same argument can be made for case (b), constant displacement (Cottrell, 1964).

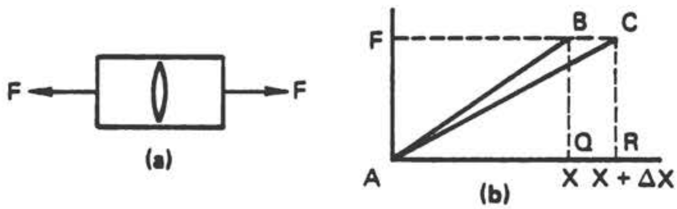


FIGURE 3-9 Energy requirements for crack propagation.

Thus, the elastic energy accounts for 50% of the work done, and the remaining 50% can become the energy of the new crack surface. If plastic flow has occurred or dislocations have been generated, the elastic energy still has accounted for 50% of the work done, but not all of the remaining work goes into the creation of new surface. Therefore, it must be recognized that if the process of fracture were 100% efficient, only 50% of the work would go into new surface area.

3.3.6.3 Energy Product Size Relationships

It is important to try to relate the amount of energy introduced into a particle to the size distribution (or surface area) of the resulting fragments. For single particles, it has been shown that as the energy input increases, the resulting product is finer. Indeed, there appears to be an excellent correlation of specific surface and specific energy input (Figure 3-10); since specific surface is inversely proportional to a measure of particle size, the particle size measure (for example, an 80% passing size or a Schuhmann size modulus) is observed to be inversely proportional to specific energy input for single particles.

When the specimen breaks and the resulting fragments possess sufficient kinetic energy, under certain conditions they will break again upon impact with some other solid. In this case there are two distinct energy-product size relationships, both of which show that product size is inversely proportional to energy input but with differing coefficients (Bergstrom and Sollenberger, 1961). Krogh (1978) has observed that the extent of crushing appears to be independent of how the energy is supplied. That is, if the energy is supplied via low impact speed but high mass or via high impact speed but low mass, the probability of breakage is the same, as shown in Figure 3-11. Similarly, both shatter tests and compression tests on coal (Evans and Pomeroy, 1966) can yield the same size distribution. It is likely, however, that more energy can be introduced into a particle before rupture occurs by using high impact speed rather than the slow, compressive techniques. The likelihood does not reflect different efficiencies, but simply operation at a different point on the curve.

An attempt has also been made by Snow and Paulding (1973) to predict fragment size distributions from the calculated stress distributions within spherical or disc-shaped specimens. Similarly, Meloy and Faust (1968) have attempted to consider energy density within a specimen as having an effect on fragment size distribution.

A recent variant of the compression test reported by Hess and Schonert (1976) uses a combination of compression and shear. The fracture energy is reportedly decreased by this means, but the extent of breakage is also reduced.

When individual specimens break, the size distribution of the resulting particles can be measured. When this size distribution is plotted as log particle size vs. log percent passing sieves of ratio $\sqrt{2}$, a steep slope over the several coarsest fractions is generally

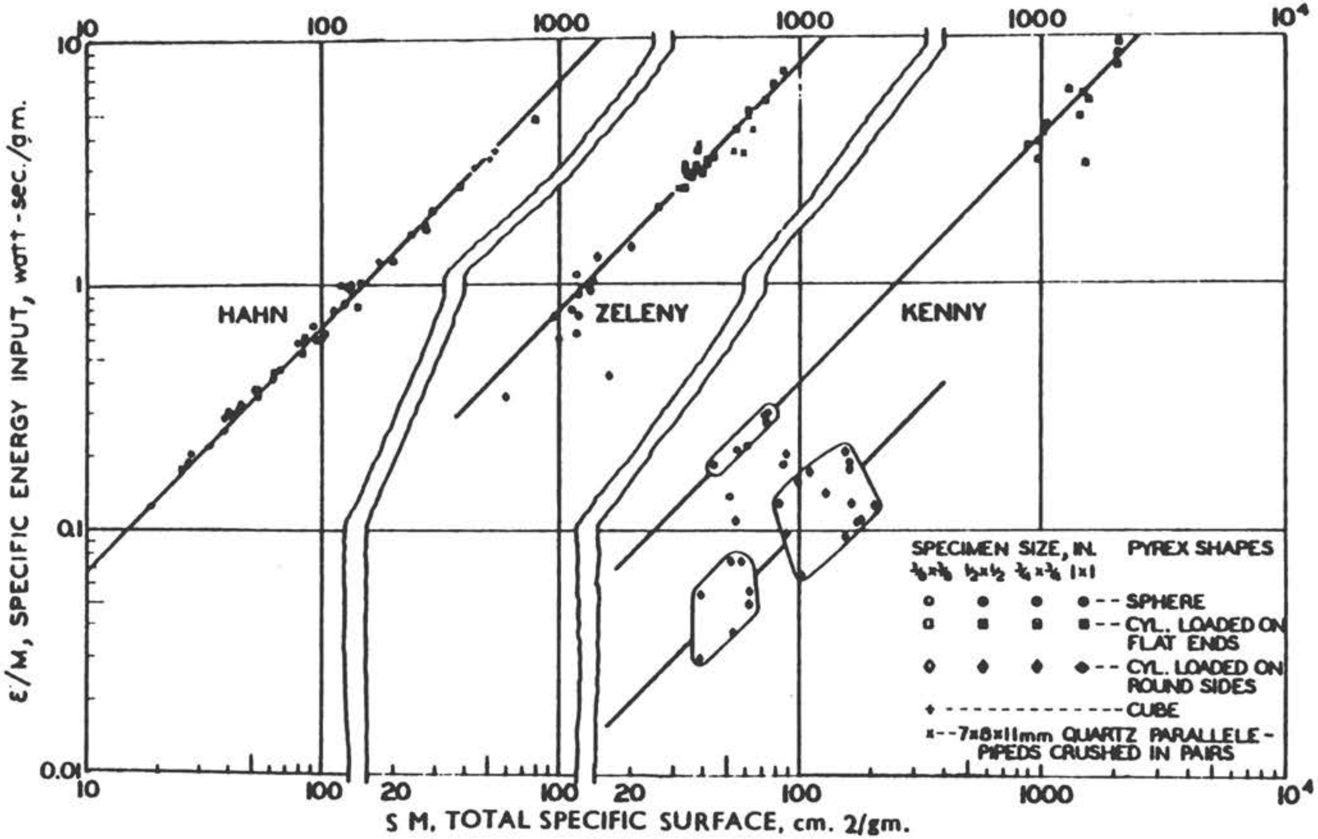


FIGURE 3-10: Relationship between specific energy and specific surface for impace and compression crushing. (Bergstrom, 1963).

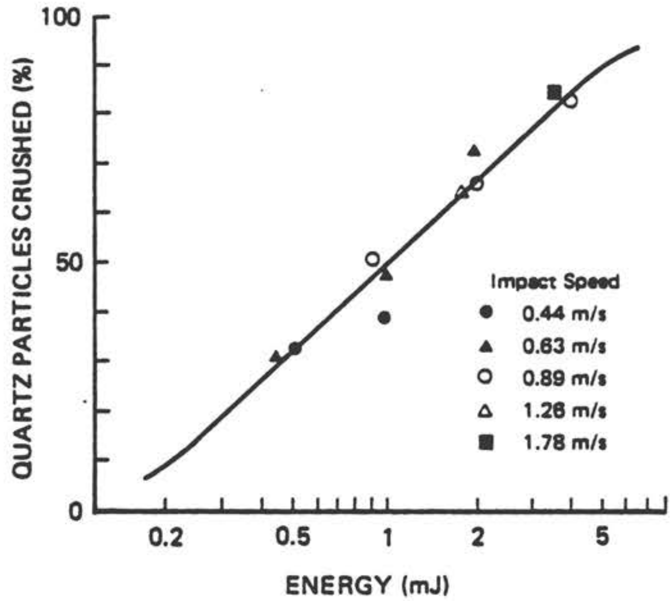


FIGURE 3-11 Breakage-energy relationship for different impact velocities. (After Krogh, 1978)

observed, with a sharp transition to a line with slope approaching 1.0 in the finer sizes. If, however, a number of specimens are similarly broken, the resulting composite sieve analysis usually yields a much smoother curve.

Such smooth curves have led some investigators to seek an equation that describes the distribution. The simplest of these equations is the Schuhmann equation, $y = ax^m$ (Schuhmann, 1940), which yields a straight line on plots of log percent passing vs. log size. Other well-known equations, which predict the size distribution of the product with varying degrees of accuracy, are those of Rosin and Rammler (1933), Broadbent and Calcott (1957), Gaudin and Meloy (1962), Gilvarry (1961), and Bergstrom (1966).

It has also been observed that the same general shape of size distribution curve results when specimens of different initial sizes are broken. Hence, it is found that, if the size is described relative to the starting size, the same equation can be used for the entire range of observations. For example, the Schuhmann equation can be written $y = a\left(\frac{x}{k}\right)^m$, where k is the initial size or the 100% passing size. Other forms of the equation sometimes used to describe fragment size distributions include the Rosin-Rammler (1933), Gaudin-Meloy (1963), and Gilvarry (1961) distributions.

Very often, however, these types of equations do not adequately describe size distributions. The distributions can be better described by vectors of amounts passing (or within) each of the size categories.

3.3.6.4 Multiple Particle Breakage

No one apparently has yet mathematically summed up the product size distribution arising out of all the distributions of fracture loads encountered in the breaking of a single mesh fraction and then for all the amounts of material in each mesh fraction. It would seem likely that this could be done mathematically, but it is much easier to do experimentally. In some such experiments, particles are spread out on a platen and the top platen is pressed against the assembly of particles, squeezing the tallest specimen, then both it and the next tallest, and continuing until all the specimens are broken or the compressive force is removed for some other reason. These experiments yield smoother stress-strain and product-size distribution curves than do single-particle experiments. Similarly, an assembly of particles may be squeezed in a press, simulating the action in the crushing chamber of a jaw or gyratory crusher. The Protodyakanov (1963) test is similar to this, but does not appear to have been used domestically.

Results of recent experiments by Hoffman et al. (1974) on crushing beds containing a binary mixture of glass spheres are shown in Figure 3-12. They indicate that the probability of the larger spheres' being broken is reduced as the proportion of large spheres is reduced. The specific surface of ground material, as measured by the Blaine technique, is inversely proportional to the square root of

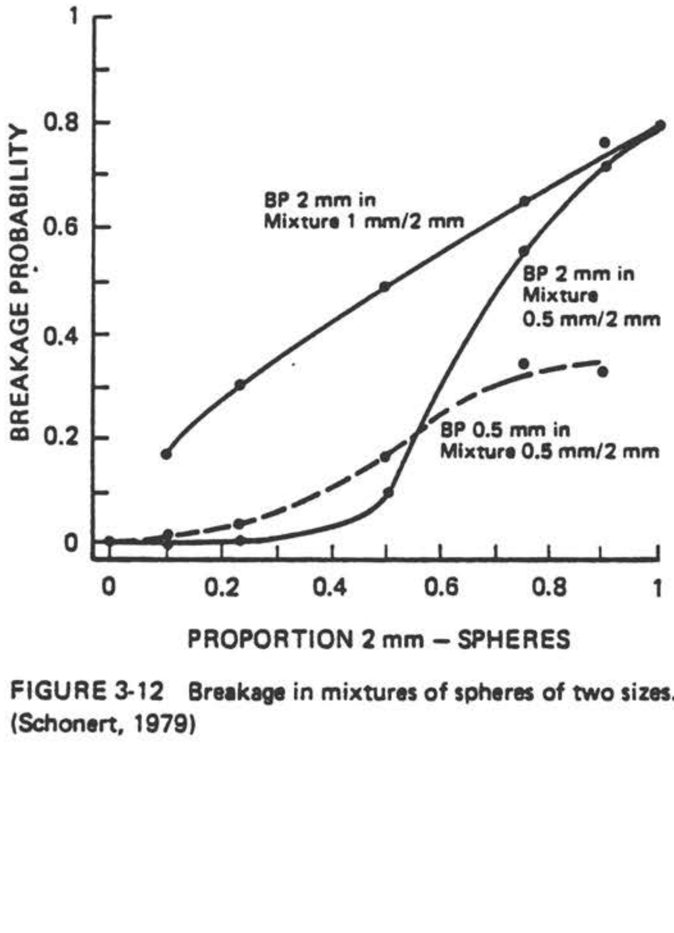


FIGURE 3-12 Breakage in mixtures of spheres of two sizes. (Schonert, 1979)

product size, (Bond, 1961 and Bergstrom, 1980), rather than being simply inversely proportional to size. This shows that the crushing of packed beds must take into account other factors than those required for the free crushing of single particles.

3.3.7 SUMMARY AND RECOMMENDATIONS

3.3.7.1 Summary

It appears that the fracture physics of single particles can be extended to a collection of particles. The point is important because commercial comminution operations are investigated in similar terms--e.g., energy and probability of breakage, size distributions, etc.--and although assemblies of tons of particles are involved and the number of events per unit time is exceedingly great, the capacities can be given as tons per hour and the energy requirements as kWh/ton. In comparing the specific energy requirement from commercial equipment to that from single particle data, however, it must be remembered that commercial equipment also conveys the particles and often exerts forces that do not result in fracture. Therefore, commercial test data do not show as high efficiencies as laboratory test data.

Fundamental research as to the means by which energy may be more effectively directed at the particles to be broken, into transport through mills, and into classifiers, and mill circuit controls may yield more energy savings.

In this it should be noted that the characteristic breakage pattern of single particles is closely related to the breakage function calculated by modern methods of mill analysis. Coupled with the breakage function is the selection function, which indicates the rate at which particles in each size category undergo fracture. Proper manipulation of the breakage and selection functions permits the calculation of the mill product size distribution from the feed distribution. Everell (1972) has tied fragmentation physics to the breakage in a ball mill, and Schonert (1979) has done the same with respect to a roll crusher.

3.3.7.2 Research Needs and Recommendations

Continued research into single particle fragmentation may yield a type of test more suitable than present tests for estimating specific energy requirements for crushing and grinding equipment.

Although such continued research is unlikely to show a dramatic decrease in fragmentation energy requirements for single particles, unless outside the usual ambient conditions of temperature and pressure, it may uncover a means of directing the applied forces more effectively at each particle so that less energy is wasted.

Research into the breakage of beds of material may similarly disclose means by which the efficiency of single particle fragmentation can be approached for beds.

Research into the breakage of assemblies of particles of different characteristics, i.e., hard vs. soft, may yield methods by which the fracture of either type can be accentuated over the other. Similarly, research may uncover means by which the valuable mineral may be more readily liberated from its composite host.

3.4 Process Models

3.4.1 INTRODUCTION

An understanding of the fundamentals of comminution processes is desirable both for the development and design of new or improved comminution devices and for the optimization and control of existing equipment. As in any branch of technology, there are numerous ways in which comminution processes can be analyzed, each approach having its particular advantages and disadvantages. Thus, in the development of new equipment, we should be concerned with the basic mechanics of the process whereas, for optimization and control, it may be sufficient to understand the response of the device without necessarily understanding the underlying reasons.

It is important to recognize that in addition to applying stress to particles, a comminution machine must also provide the means of setting the particles up so that stress can be applied. Thus, materials handling (particle transport) is an integral part of the process. In coarse crushing systems, it is often feasible to handle the large feed particles individually so that the materials handling aspect can be quite carefully controlled. Such individual handling is rarely possible in fine grinding because of the very large numbers of very small particles involved. In these systems it is often necessary to rely on the random motion of particles in the device to provide adequate opportunity for stress application. There can be little doubt that the performance of fine grinding equipment frequently will be limited by the materials handling aspects rather than the actual particle breakage aspects of the process.

The study of comminution fundamentals falls under three broad headings:

- Comminution Models - These are generally macroscopic models in which the machine is treated simply as a means of performing certain actions on the feed particles. The performance of the device is then described in terms of the parameters of the model.

- Machine Dynamics - Analysis of the actual transport and breakage processes which are responsible for the performance of a given device.

- Mill and Circuit Design - Application of fundamentals to the prediction and simulation of grinding system behavior.

In this section, which deals with comminution models only, the current state of the art is described and specific areas are indicated where fundamental knowledge is lacking and where more research is needed.

3.4.2 ENERGY-SIZE REDUCTION RELATIONSHIPS

It has long been recognized that comminution, in effect, is a process of creating new surface area. This has been interpreted as meaning that the energy required to produce unit area of new surface, by grinding, should be proportional to the specific surface energy of the material being ground. Unfortunately, however, this concept totally ignores the fact that, even under ideal conditions, the energy going into new surface represents only a part of the energy needed for fracture. Much of the actual energy input goes into stored strain energy which is not recovered and eventually reappears as heat. In any practical comminution device, of course, this is compounded by energy wasted--in fruitless ball-ball contacts, for example.

If we consider, as an example, the grinding of cement clinker to a (Blaine) surface area of about 3000 cm^2/g and assume a reasonable value of about 1000 $ergs/cm^2$ for the surface energy, the energy required for production of new surface would be about 0.075 kWh/ton. Typical plant practice, on the other hand, involves the expenditure of more than 30 kWh/ton, indicating that less than 0.25% of the energy goes into the creation of new surface. Since there is no reason to expect the remaining 99+% to be in any way related to surface energy, it seems very unlikely that there should be any direct correlation between energy consumption and the production of new surface.

The concept of an energy/size correlation nevertheless has led to a number of relationships that can be used in mill design. The success of these relationships, however, can be attributed much more to the direct proportionality of energy and time than to the surface energy concept. In most grinding machines, the power drawn is more or less constant, so that energy consumption is a direct measure of time. Furthermore, it is clear that any reasonable change in mill design that leads to increased energy consumption will also lead either to increased capacity of the device or to more effective breakage of the particles. Any of these factors, therefore, should give rise to a rough correlation between energy consumption and the production of fine particles (i.e., a new surface).

Probably the best known of the so-called energy/size relationships is that developed by Bond (1952), which can be expressed as:

$$E = 10\, W_i \left(\frac{1}{x_p^{1/2}} - \frac{1}{x_f^{1/2}} \right) \qquad 3.4\text{-}1$$

where x_p and x_f are the 80% passing sizes (microns) in the product and feed respectively and W_i is an empirical factor called the Work Index. Standard test procedures have been developed for determining Work Indices for different materials.

Equation 3.4-1 is used as the basis for the selection and sizing of mills in conjunction with empirical relations for power draw as a function of mill size, conditions, etc. The principal applications are to tumbling mills (ball and rod mills). The relationship has been used in crusher design, but these machines are more commonly selected from standard tables for typical materials: coal, quartz, limestone, etc. Again, it should be emphasized that Equation 3.4-1 is not a fundamental relationship between grinding energy and product size. Rather, it is an empirical relationship which, with the proper application of numerous empirical correction factors, is found to correlate mill performance reasonably accurately.

Another well known energy/size relationship (Charles, 1957), can be expressed by:

$$E = C\left(\frac{1}{k_p^{\alpha}} - \frac{1}{k_f^{\alpha}}\right) \qquad 3.4\text{-}2$$

where k_p and k_f are, respectively, the (extrapolated) 100% passing sizes for the product and feed. The exponent α normally lies between 0.5 and 1.5 and often appears to correlate with the slope of a log-log plot of the actual product size distribution. C is a constant, similar to, but not equal to, the Bond Work Index. Through the introduction of a second adjustable parameter, α, the Charles relation generally provides a better description of a given grinding system than does the Bond equation. However, since much more information has been collected on the appropriate scaling factors, etc., for Bond's law than for the Charles equation, the former is more widely used in mill design. Other energy/size relationships, such as the well known laws of Kick and Rittinger, are similar in form to Equations 3.4-1 and 3.4-2 but are seldom used in mill design.

It is clear that the energy/size relationships have practical value in mill design, but it is doubtful that they have any real connection with the energy actually needed for grinding as opposed to driving a mill. They are in no sense thermodynamic models of the grinding process. They are, in fact, nothing more (nor less) than purely empirical correlations of operating data.

There can be no doubt that the empirical approach has served well in the past--most if not all of the grinding mills in operation today have been designed by these methods. Obviously there are inadequacies, however. As with any kind of empirical correlation, interpolation is relatively straightforward and safe; extrapolation beyond the normal range of operating conditions is risky. In the absence of a true predictive capability, the empirical models cannot be used for evaluating possible grinding circuits on paper--actual, full-scale tests must be carried out--nor can they be readily applied to automatic control systems.

3.4.3 KINETICS

A more realistic analysis of grinding systems can be obtained by considering the process to be analogous to a chemical reaction in which large particles react in the mill to produce a set of smaller, product particles. An early practical application of this approach was described by Broadbent and Calcott (1956) who analyzed crusher performance in terms of the probability of breakage and the size distribution of the product fragments at any stage in the crushing process. Their formulation, however, is of limited value in the analysis of continuous, fine-grinding processes--in tumbling mills, for example--where there are no readily identifiable stages of crushing. For such systems, it is generally more convenient to describe the process in terms of the kinetics models of grinding.

If we arbitrarily divide the particle size spectrum into a number of finite, narrow, size intervals, the grinding reaction can be treated as a process of transforming mass in one size class into mass distributed through a set of smaller size classes. We can then define two fundamental parameters of the breakage process:

S_j = specific rate at which particles in size class j are broken

b_{ij} = mass fraction of the fragments produced by breaking size j which fall into size class i.

It is often convenient to express the primary breakage distribution in cumulative form such that

B_{ij} = mass fraction of the fragments produced by breaking size j which are smaller than size i.

Thus the set of S values describes the kinetics of the process and takes into account the rate at which particles are subjected to stress as well as the likelihood of the stress's leading to actual fracture. The B (or b) values characterize the products of each individual breakage event.

An important feature of most of the population balance models is the so-called First-Order Hypothesis, which states that the rate of breakage of any size is proportional to the amount of that size in the mill. It should be emphasized that there is no a priori reason why grinding should, if fact, be a first-order process. However, this assumption does appear to be valid for a large number of experimental systems.

If we express the size distribution of the mill contents by

W_i = mass fraction of all particles in the mill which fall into size class i

the first-order hypothesis can be written, for the largest particles present (class 1):

$$\frac{dW_1}{dt} = -S_1W_1^1 \qquad 3.4\text{-}3$$

In fact, Equation 3.4-3 can be applied within the first-order hypothesis if S is permitted to vary functionally with size distribution or time. If S_1 is independent of time, Equation 3.4-3 can be integrated to give

$$W_1(t) = W_1(o)e^{-S_1t} \qquad 3.4\text{-}4$$

where $W_1(o)$ expresses the relative amount of size i material in the mill at time zero (i.e., in the feed to the mill).

To carry out similar mass balances on the other, smaller sizes, it is necessary also to consider material broken into the class from the breakage of larger sizes. This leads to a general form of the Batch Grinding Equation given by:

$$\frac{dW_i}{dt} = \sum_{j=1}^{i-1} S_jb_{ij}W_j - S_iW_i \qquad 3.4\text{-}5$$

The analytical solution to Equation 3.4-5 was given by Reid (1965).

3.4.4 BREAKAGE PARAMETERS

Obviously, determination of the breakage parameters S and b is a necessary prerequisite to the application of the population balance approach to real grinding systems. The specific rate of breakage of the largest size, S_1, can readily be evaluated by considering Equation 3.4-4, which states that a semilog plot of the relative amount remaining in that size versus time should yield a straight line whose slope is proportional to S_1. If we carry out a series of experiments in which the largest size in the feed is class 2, 3, 4... etc., we can obtain values for S_2, S_3, S_4... etc., provided these values can be shown to be independent of the presence of other (especially coarser) sizes in the mill.

The validity of this concept was tested by Gardner and Austin (1962), who used radio-tracing techniques to determine rates of breakage of small particles in the presence of a range of sizes, both coarser and finer. The concept has generally been accepted in subsequent work and appears to be justified in terms of the consistency of the results obtained. However, it remains an assumption in that its general validity has not been demonstrated and probably should be investigated further.

The breakage distribution parameters are more difficult to obtain experimentally since they require determination of the complete size distribution resulting from a single (average) breakage event. Several methods have been developed for evaluating these parameters. These include: direct measurement, based on the grinding of single size fractions for short times (Herbst and Fuerstenau, 1968; Luckie and Austin, 1972) and the use of correction procedures to account for secondary breakage; radio-tracer techniques (Gardner and Austin, 1962; Snow and Meloy, 1971); and back-calculation procedures by nonlinear regression of the product size distributions (Herbst et al, 1971; Klimpel and Austin, 1977).

Based on a large number of experimental grinding studies, primarily in ball mills, there appear to be distinct patterns in the behavior of the breakage parameters, S and B. For any given mill, the specific rates of breakage, S, generally increase with increasing particles size, but eventually pass through a maximum when the particles become too large to be broken effectively in the device. For the smaller sizes, the specific rate of breakage generally follows a simple power law:

$$S_i = a\,x_i^{\alpha} \qquad 3.4\text{-}6$$

where a is a constant and the exponent α typically has values fairly close to 1.

The primary breakage distribution parameters, B_{ij}, are normally found to be relatively independent of the size being broken. In fact for many (but by no means all) systems, normalization of the distributions with respect to the feed size yields a set of values that falls on a single curve. Thus, for geometric size intervals, $B_{ij} = B_{i-j}$ or, in other words, B_{ij} is a function of x_i/x_j only and is independent of the absolute value of the feed size, x_j. Again it is found that, for the fine sizes, the cumulative breakage distribution parameter, B_{ij}, typically follows a simple power law:

$$B_{ij} = b_o \left(\frac{x_i}{x_j}\right)^{\beta} \qquad 3.4\text{-}7$$

where b_o and β are constants. For those cases where B_{ij} is not normalized, the coefficient b_o becomes a function of the size, x_j, of the particles being broken.

The existence and specific form of these consistent patterns in the breakage behavior account for the success of the earlier, empirical, grinding laws. It can be shown, for example, that if Equations 3.4-6 and 3.4-7 are valid and if $\alpha \approx \beta$, which often is approximately true in practice (Herbst and Fuerstenau, 1968), simple relationships such as Charles' Law can be obtained directly from the batch grinding equation (Kapur, 1970).

3.4.4.1 Nonlinear Systems

Some grinding systems, rod mills for example, and many cases of fine, wet grinding, are found not to behave in a linear fashion. Specific rates of breakage are found to be affected by the accumulation of fine particles in the mill and by protection of small particles by the larger particles in the feed. Various linearization techniques have been described (Grandy and Fuerstenau, 1970; Shoji and Austin, 1974; Herbst and Mika, 1973) which can be used in the analysis of systems that exhibit this kind of behavior.

Departures from ideal behavior can also be expected in the grinding of heterogeneous materials. If particles of a given size possess a distribution of breakage characteristics, the weaker particles presumably will break preferentially with the result that the average specific rate of breakage decreases with time as the weaker components are broken out of the class, leaving only the stronger, less easily broken particles. This kind of behavior has been proposed to account for the so-called abnormal breakage of very large particles in ball mills (Austin et al., 1973).

3.4.5 TRANSPORT PHENOMENA

3.4.5.1 Continuous Grinding Systems

An attractive feature of the kinetic model or population balance approach is that the extension to continuous grinding is relatively straightforward. If the breakage parameters are independent of position in the mill, the size distribution of the product leaving a continuous mill operating at steady state can be obtained from:

$$P_i = \int_o^\infty W_i(t) \frac{d\phi}{dt} dt \qquad 3.4\text{-}8$$

where $\phi(t)$ is the residence-time distribution function, defined as the fraction of material entering the mill at time zero which has left at time t (Reid, 1965). The $W_i(t)$ are defined as in Equation 3.4-5.

Equation 3.4-8 can be simplified considerably in special cases such as plug flow or for a fully mixed mill when $\phi(t)$ takes on simple analytical forms. The general case can also be treated as n fully-mixed in series, which leads to considerable computational simplification. In cases where there are significant variations in filling level, etc., along the length of a mill, variations in the breakage parameters might be expected and more complex equivalents to Equation 3.4-8 should be used. This situation might be expected to arise in the case of continuous dry grinding in ball mills. Further complications will occur if the transport of particles through the mill is size-dependent.

For the straightforward cases described by Equation 3.4-8, the only additional information required is the residence-time distribution, which can be obtained from tracer studies. For the more complex cases, a thorough understanding of the mass transport laws is also required.

3.4.5.2 Application to Specific Grinding Systems

The general, population balance approach, with appropriate modification, can be applied, in principle, to any kind of grinding device. It is useful to classify grinding systems into two broad categories:

i) Retention devices, in which material is held for a finite time within the device and subjected to repeated breakage action.

ii) Once-through devices, in which particles are subjected, usually one at a time, to a single breakage action, and the products pass immediately out of the machine.

Typical examples of retention devices are the tumbling mills (ball, rod, pebble, etc.), roller mills, choke-fed crushers, and even the hand mortar. The roll crusher is an example of a once-through device.

Retention devices normally can be analyzed in terms of the general mathematical formulation outlined above. The majority of studies to date have been concentrated on tumbling mills, especially conventional ball and rod mills, although there have also been a number of studies of roller-type mills, particularly the Hardgrove machine. Obviously, since there are significant differences in the mechanical action of these devices, there must be corresponding differences in the specific methods needed for analysis. Some systems, crushers for example, cannot be operated in the batch mode, and must be analyzed with the machine operating continuously under steady-state conditions. Several of these devices have some kind of internal classification system that preferentially removes fine (product) particles. Since this system is usually an integral part of the device, the device must be analyzed as a grinding circuit and the classifier selectivity function must be incorporated into the analysis.

Internal classification is also a necessary feature of autogenous and semiautogenous mills in which large lumps of rock are used as the grinding media in a more or less conventional tumbling mill. Analysis of these mills is further complicated by the fact that, in addition to the normal mode of breakage from the action of the grinding media, self-breakage of the large lumps (the media) is also important (Austin et al., 1977).

The once-through grinding devices can also be modeled using the population balance approach. The rate-of-breakage concept has no meaning in such systems since oversize particles are broken immediately upon entering the crushing zone. Mills of this kind, in principle, can be modeled simply by specifying the complete breakage distribution matrix (B_{ij}). Alternatively, these systems can be treated in terms of hypothetical grinding circuits in which the products of each breakage event can either pass out of the machine or, if they are still larger than the gap setting, be returned to the crushing zone for rebreakage (Austin and Van Orden, 1979).

3.4.5.3 Application to Industrial System

Obviously, the principal value of comminution models lies in their application to the design and operation of industrial grinding systems. To date, such applications of the population balance models have been somewhat limited, largely because of the absence of the necessary data base, scale-up laws, etc. Nevertheless, significant progress has been made, especially in the past decade, and increasing applications can be expected in the near future.

To apply these models to industrial mills, it is clearly necessary to determine the appropriate breakage parameters, etc. For many systems, ball mills for example, the breakage parameters are most conveniently determined from batch tests on small, laboratory-scale mills. If the appropriate scale-up laws are known, the parameters determined can then be applied to the corresponding, full-scale mills.

In general, it appears that the breakage distribution functions, B_{ij}, are relatively insensitive to the size of the mill. It seems that, providing a particle breaks, the manner in which it is broken has little effect on the size distribution of the products.

Specific rates of breakage, on the other hand, depend quite strongly on the size of the mill. For ball mills operating under conditions of dynamic similarity--i.e., with the same fractional ball loading, material hold-up, and relative rotational speed--it is found that the specific rate of breakage increases roughly with the square root of mill diameter.

Herbst and Fuerstenau (1973) have shown that specific rates of breakage can be correlated directly with the specific energy consumption of the mill. An important feature of this result is that it implies that much of the data collected, on Work Indices for different materials, for example, potentially can be applied to the estimation of specific rates of breakage. Again, the correlation between energy input and rate of breakage should not be interpreted to mean that the two are directly related, but rather that they are both manifestations of the complex mechanical action of the tumbling-ball charge which is responsible both for the power needed to drive the mill and for the grinding that occurs.

It is interesting to note that it is the rate of breakage that appears to correlate with energy rather than the size distribution of the primary breakage products. Correlation with the latter would, of course, be consistent with the surface energy concept.

Some information is also available on the effects on breakage rates of such process variables as ball loading, ball size distribution, material hold-up, and pulp density (in wet grinding). The effects of chemical grinding aids have also been investigated.

It should be emphasized that this brief survey of the current state of the art of modeling comminution systems is intended as background material only and in no way represents an exhaustive review of the extensive literature on the subject. For more detail, the reader is referred to the review articles (Austin, 1972; Herbst et al., 1973; Snow and Luckie 1973a, 1973b, 1976).

3.4.6. RESEARCH NEEDS

The research needs discussed here are also relevant to Section 3.5, Device Dynamics.

It is clear that while significant progress has been made in the development and application of comminution models, considerably more research is needed to provide a sound technological basis for the design and operation of efficient comminution systems. Research needs in comminution fundamentals can be classified into three broad areas: model development, data acquisition, and application to design and control.

3.4.6.1 Model Development and Evaluation

- o Further development of basic population-balance models is needed, especially with respect to those systems that exhibit nonlinear behavior.

- o The direct incorporation of process objectives--liberation of ores, performance of cement powders, etc.--into the comminution models should be investigated.

- The development of specifically modified models for different kinds of mills, such as rod mills, crushers, autogenous and semiautogenous mills, etc., should be continued.

- Further investigations of parameter estimation procedure are needed, including the development of standardized laboratory tests and simplified procedures for large-scale mills.

3.4.6.2 Data Acquisition

There is considerable need for the development of a broad data base, including both grinding and transport parameters, for different materials being ground in different types and sizes of mills. The possibility of correlating existing data with the modern grinding models should be explored.

3.4.6.3 Application to Grinding System Design and Control

The ultimate goals of research into comminution fundamentals are:

- The development of methods of mill design based on the optimization of grinding kinetics with respect to machine dynamics, material transport, and classifier performance.

- Application of the fundamental principles to circuit simulation and automatic control.

3.5 Device Dynamics

3.5.1 INTRODUCTION

It cannot be overemphasized that practical comminution machines are required to perform a materials handling function in addition to applying stress for particle breakage. Furthermore, there may be different aspects to the materials handling function itself. For example, there are particle transport effects in batch ball milling which contribute to the specific rate of breakage--particles will only be broken if they are appropriately located in the mill. In continuous mills, additional transport effects are involved in moving the material through the mill. In general, it is convenient to lump the first kind of transport effects into breakage kinetics, discussed in Section 3.4.3, and to consider the second kind in terms of a mass transport law.

3.5.2 INTERNAL MECHANICS (BREAKAGE)

Various kinds of crushing action are employed in different comminution devices. Lowrison (1974) has classified the different kinds of equipment according to the principal comminution actions:

- Compression or nipping
- Blow or impact
- Tumbling or projection
- Cutting or shredding
- Attrition

3.5.2.1 Compression Machines

The compression, or nipping, devices include jaw, gyratory, and roll crushers and disc, pan, and roller mills. In each of these, particles are broken by slow compression between two surfaces. Details of the design and mode of operation of these devices are covered in the literature (Lowrison, 1974; Taggart, 1945) and need not be repeated here. It is perhaps useful to subdivide these devices futher into those in which there is a fixed gap between the crushing surfaces (e.g., jaw, gyratory, and roll crushers) and those in which a fixed force is applied and the crushing surfaces can come almost into contact (e.g., roller mills).

In the fixed gap devices, particles will be broken if they are larger than the gap size but smaller than some limiting size determined by the so-called angle of nip. The latter is illustrated, for the case of crushing between rolls, in Figure 3-13. It can easily be shown that the limiting angle of nip is defined by:

$$\tan \theta/2 = \mu$$

where θ is the angle of nips and μ is the coefficient of friction between the crushing surface and the particle being broken. Large particles, for which $\tan \theta/2 > \mu$, will slip at the surface rather than be drawn into the gap and broken. Another common feature of these devices is that, because of the fixed gap, essentially any particle in the appropriate size range must be broken.

For the other kinds of nipping devices, the upper size limit will again be determined, in general, by the angle of nip. For these systems, however, a fixed force is generally applied and there is no fixed gap size. The effective gap size may depend to some extent on the amount of material in the mill and perhaps on its size distribution. In general, there is no minimum size which can be broken, although it is to be expected that there will be some degree of protection of fine particles by coarse.

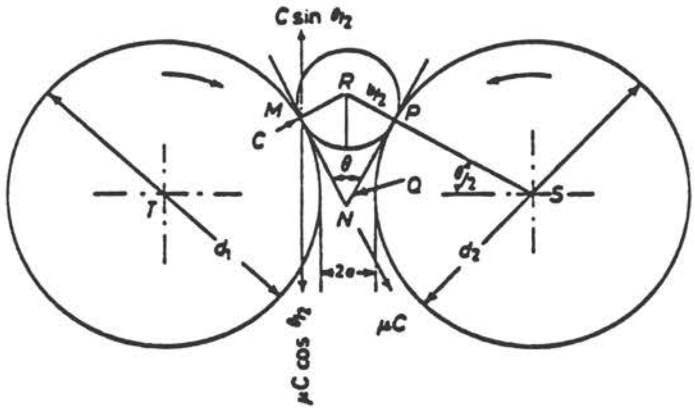


FIGURE 3-13 Angle of nip, θ, of a particle of diameter b between circular surfaces of diameter d_1 and d_2.

3.5.2.2 Impact Mills

The impact mills, which include hammer mills and vibratory mills, operate by subjecting particles to sudden high stress through impact. Details of the design and construction of these devices are given in Lowrison (1974) and Taggart (1945). In hammer mills, free particles are stressed by impact with swinging hammers and again by collision with fixed surfaces in the mill. Breakage may be due to either of these impacts. In vibratory mills and the older stamp mill, particles are broken by high speed compression between two surfaces.

3.5.2.3 Tumbling Mills

Probably the most important comminution devices, at least in terms of installed capacity and overall energy usage, are the tumbling mills. These mills, in effect, are a special class of impact machines in which particles are broken by impact with a tumbling mass of loose grinding media. The latter may take the form of steel (or other) balls (ball mills), steel rods (rod mills), or large pieces of the material being ground (pebble mills and autogenous mills).

In general, tumbling mills consist of a horizontal rotating cylinder filled to somewhat less than half of its volume with the grinding media. Normally the material being ground more or less fills the interstices in the charge of grinding media. Rod mills generally are used for relatively coarse feeds and products and ball mills for finer grinding. Autogenous mills can be used to combine coarse crushing and grinding in a single operation. Control of these devices often dictates the addition of some steel grinding media, in which case the term semiautogenous grinding is used. Tumbling mills normally are rotated at speeds corresponding to about 60-70% of the critical speed, defined as the speed of rotation at which centrifuging of the outermost layer of grinding media begins.

At any instant, the charge in a tumbling mill can be considered to consist of two regions (Hogg and Fuerstenau, 1972): a fixed zone, in the bulk of the charge, where there is little or no motion relative to the mill shell; and a shear zone, in the vicinity of the free surface of the charge, where the material undergoes a violent, tumbling motion before being reentrained in the fixed part of the bed. It is presumably in the shear zone that most of the grinding takes place.

Energy input to the mill goes primarily into raising the charge through the fixed zone (Hogg and Fuerstenau, 1972), i.e., as potential energy. This is converted into kinetic energy in the shear zone, where it becomes available for grinding or is dissipated as heat. Since energy consumption is determined by motion in the fixed zone, while grinding occurs in the shear zone, it is clear that there is very little direct relation between energy and grinding. At the same time, there is no doubt that the amount of energy available for grinding and the capacity of the mill are both determined by the same factors, which can lead to an apparent relation between energy and grinding. It should be noted, however, that the power required to drive a tumbling mill is roughly constant, regardless of whether or not grinding is taking place.

The effectiveness of grinding in tumbling device is probably determined by the nature of the motion in the shear zone. At relatively low speeds of rotation, the material undergoes a kind of rolling, tumbling motion known as cascading, which involves frequent collisions between the individual particles, balls, etc. At higher speeds, particles, etc., are projected out of the main body of the charge and are in free flight for part of their path through the shear zone. Motion of this type is known as cataracting (Figure 3-14) and involves fewer, though presumably more energetic, collisions than does cascading. It seems likely that these collisions can lead to direct transfer of kinetic energy back to the mill shell with a corresponding reduction in the specific power requirements (Hogg and Fuerstenau, 1972). Under normal operating conditions, the motion in the shear zone is a (presumably optimum) combination of cataracting and cascading. The former gives high-energy impacts to break tough particles, while the latter gives high impact frequencies and the opportunity for higher rates of breakage.

3.5.2.4 Cutting and Shredding Machines

Cutting and shredding devices operate through the cutting action of some kind of knife blade which is either applied to stationary particles in contact with a fixed surface or makes use of the inertia of the particles in a high-speed impact. Machines of this type are used principally for nonbrittle materials such as metals and organics.

3.5.2.5 Attrition Mills

A number of fine-grinding devices, including stirred ball mills, colloid mills, and, to some extent, fluid-energy mills, produce fine particles largely by abrasion or attrition of the feed material. Generally these are low-capacity devices for specialized applications and currently do not contribute significantly to the nationwide consumption of energy in grinding.

3.5.3 MATERIAL TRANSPORT

The capacity of grinding machine may be determined by the actual breakage process, as in typical crushing devices, or it may be controlled to some extent by the transport of material through the device. Various modes of particle transport are used in different kinds of comminution machinery. The simpler forms include:

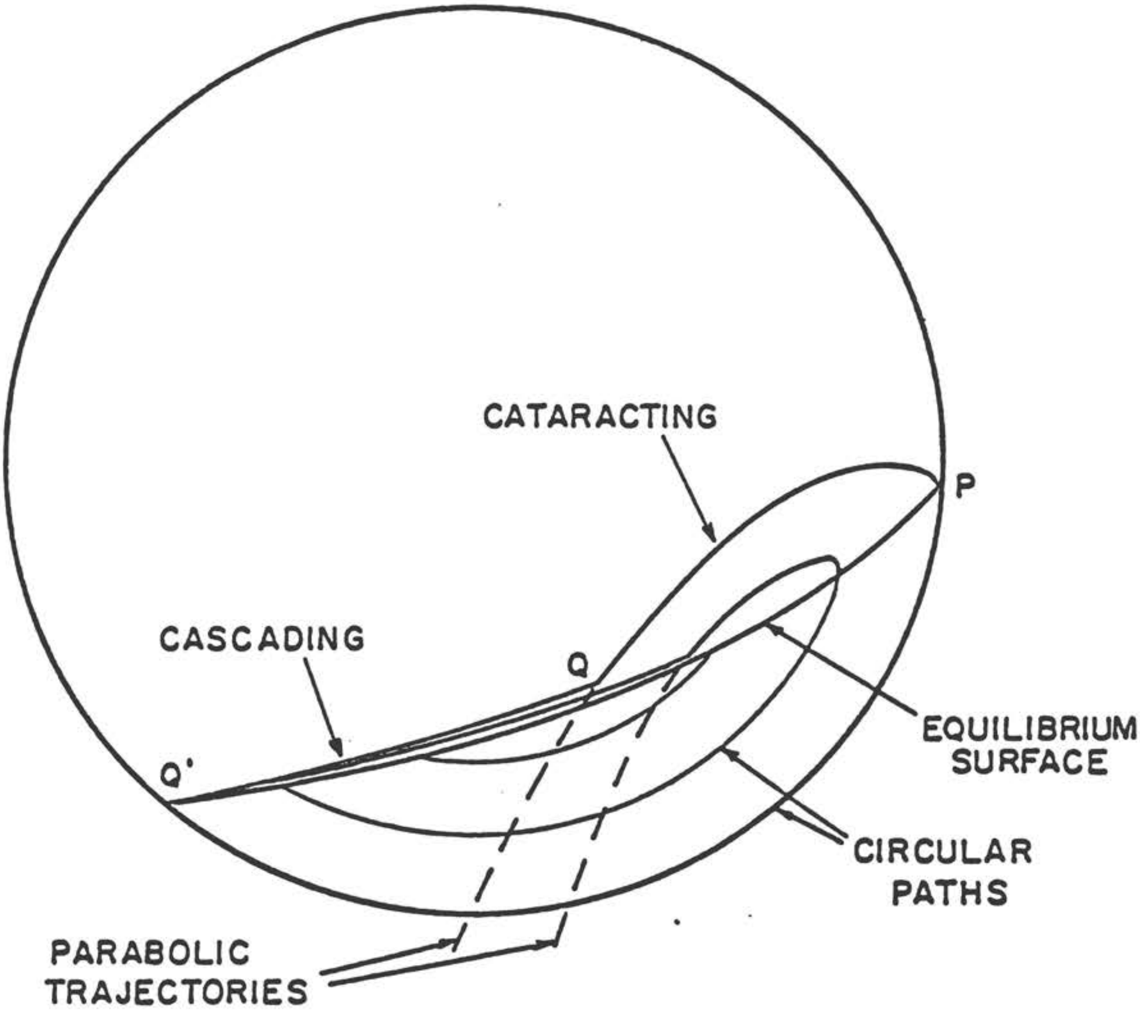


FIGURE 3-14 Schematic representation of combined cataracting and cascading motion in a rotating drum.

- o Simple gravity flow (jaw, gyratory, and roll crushers)
- o Mechanical action of ploughs and scrapers (pan mills)
- o Fluid flow (air-swept hammer mills, fluid-energy mills)

Other systems involving combinations of these mechanisms include:

- o Gravity plus mechanical (dry tumbling mills)
- o Mechanical plus fluid flow (air-swept roller mills)
- o Gravity plus mechanical plus fluid flow (wet tumbling mills, air-swept tumbling mills).

Since tumbling mills generally involve the most complex transport mechanisms and are probably the most important nationally in terms of energy consumption, the following discussion is limited to material transport in tumbling mills.

3.5.3.1 Transport in Rotating Cylinders

A common feature of tumbling mills is that the basic machine generally consists of a horizontal rotating cylinder that contains the grinding media. Material is fed into the cylinder through an axial circular opening and normally discharges at the other end, either through a similar opening (wet ball and rod mills) or through a grate (dry ball mills and some wet ball mills). Rod mills often discharge through the periphery of the mill, either at the end or in the center (in the latter case, the mill is fed from both ends simultaneously).

The flow of dry powders through simple rotating cylinders (i.e., in the absence of grinding media) has been analyzed in considerable detail (Saeman, 1951; Vahl and Kingma, 1952; Hogg et al., 1974) and the basic transport laws are quite well understood. Briefly, axial transport through such a cylinder results from the existence of a slight inclination of the charge with respect to the horizontal axis of the cylinder. As material passes through the shear zone at the free surface (see Figure 3-14), its actual motion is along the line of steepest descent, which gives it a small component of velocity parallel to the cylinder's axis. Motion in the fixed part of the charge is exactly perpendicular to the axis. Individual particles, therefore, follow a kind of spiral path along the mill as they alternately pass through the fixed and shear zones in response to rotation of the cylinder. By analyzing the geometry of such motion, it is possible to predict the distribution of material along the length of the cylinder as a function of feed rate, cylinder dimensions, and speed of rotation, etc. Different kinds of discharge mechanisms lead to different boundary conditions at the discharge point. Predictions based on these simple models have been shown to be in excellent agreement with experimental data (Hogg et al., 1974; Karra and Fuerstenau, 1978).

3.5.3.2 Dry Ball Mills

Unfortunately, the addition of grinding media to a simple cylinder leads to a very considerable increase in the complexity of the transport mechanisms. In addition to modifying the relationship between material filling and the geometry of the charge in the mill (Hogg et al., 1975), the grinding media also can have a number of specific effects on the actual transport mechanisms. Since the basic driving force for transport through the mill--gravity--is not affected by the grinding media, it is reasonable to expect that the gravity flow mechanism responsible for transport in a simple cylinder will also contribute to flow through a ball mill. The presence of grinding media will lead to interference in powder flow, in both the axial direction and the transverse plane, and may also provide an additional transport mechanism through the random, backward-and-forward displacement of particles. A further complication is the fact that the material entering the mill necessarily is fed onto the surface of the charge of grinding media. Material must, therefore, percolate radially into the interstices of the ball charge at the same time that it is transported along the mill. In other words, transport in a ball mill, unlike that in a simple cylinder, should be treated as a two-dimensional problem.

The problem is complicated even further by the partial mobility of the grinding media themselves. At high particle fillings, the charge becomes expanded and the media, free to act much like any other particles in the mill, tend to migrate toward the discharge end where they accumulate and may lead to increased resistance to particle transport, and ultimately to choking of the mill.

There is evidence that transport through the mill may also be influenced by the grinding process, particularly by the production of excessive amounts of very fine particles. Recent experiments (Pour-Madani, 1979) seem to indicate anamolous transport behavior at very low flow rates, possibly because of the detrimental effect of fines on the flow of the dry powder.

Experimental studies (Hogg et al., 1975; Pour-Madani, 1979; Wood, 1979) of particle transport in dry, grate-discharge ball mills, ranging in diameter from 4 in. to 3 ft, indicate that, in general, the hold-up in the mill varies in a roughly linear fashion with feed rate. This represents a significantly stronger dependence than that found in the absence of grinding media, where the hold-up increases with about the 0.6 power of the feed rate.

The same experimental studies have indicated that the typical distribution of particles along the mill's axis is as shown in Figure 3-15. A rather surprising feature of these results is the low filling level close to the feed end of the mill. This is believed to be caused by enhanced flow of particles over the surface of the ball charge. It is interesting to note that quite similar distributions have been observed in batch ball mills.

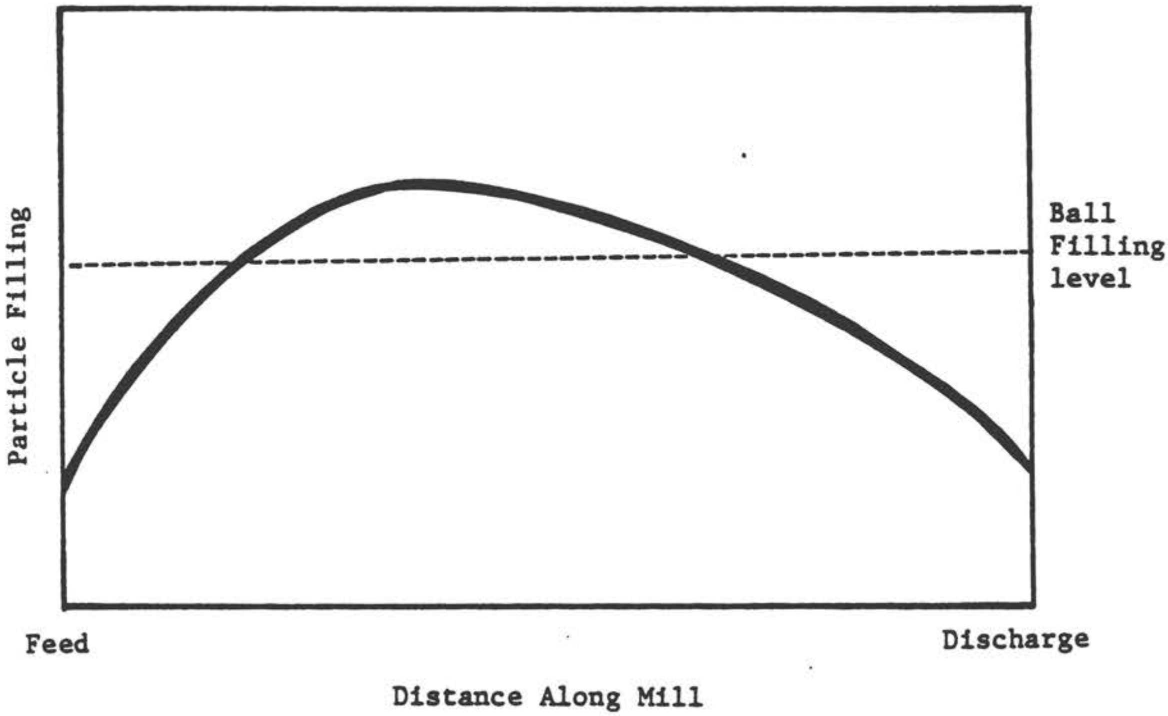


FIGURE 3-15 Typical distribution of particles in a dry, grate-discharge ball mill.

3.5.3.3 Dry Rod Mills

The material transport mechanisms in dry rod mills are generally similar to those in ball mills. Some differences can be expected, however, as a result of the continuity of the media (rods) along the length of the mill, which leads to reduced resistance to axial flow of powder. Variations in the porosity of the media charge along the mill are completely determined by the values at the ends of the mill. Migration of the grinding media is clearly not possible in these mills.

3.5.3.4 Air-Swept Ball Mills

Particle transport in air-swept ball mills is probably controlled largely by entrainment in the airstream, which also provides for internal classification through preferential pickup of the finer particles. Since the latter will generally be found in the greatest quantity at the discharge end of the mill, the air-sweeping action will lead to the development of a filling gradient along the mill. Consequently, it is to be expected that the normal, gravity-flow mechanisms will also play a role in these mills.

Entrainment in the airstream will generally depend on particle size and also on presentation of the particles to the stream--i.e., only those particles in the free surface of the rotating charge can be selected for transport out of the mill.

3.5.3.5 Wet Ball and Rod Mills

It is often considered that particles move through wet mills predominantly by a slurry transport mechanism. However, in view of the high solids concentrations involved, it is likely that some of the transport is provided by mechanisms similar to those found in dry mills. It is also likely that particle transport is impeded by the grinding media through a kind of filtration mechanism. This postulate is supported by experimental evidence that the mean residence time in the mill is generally longer for particles than for the water.

While some data exist (Imaizumi et al, 1968) which indicate that the hold-up in a grate-discharge wet ball mill varies with the square root of the feed rate, there have been very few reports of systematic studies of transport in these systems.

3.5.3.6 Autogenous Mills

Transport in dry autogenous mills such as the Aerofall type, in which breakage is largely through impact of the particles with the mill shell, is probably similar to that in simple rotating cylinders, but has not been studied in detail. In wet autogenous and semiautogenous mills, the transport mechanisms are probably similar to those in wet ball mills, with the additional complication that the large lumps, while serving as grinding media, are themselves broken and can eventually be discharged from the mill. Thus, transport in these mills is partially controlled by the breakage process.

3.5.4 RESIDENCE TIME DISTRIBUTIONS

The residence time distribution of particles in any grinding machine is clearly important in determining the performance of the mill. Residence time distributions have been studied extensively using radioactive (Gardner et al., 1980) and other tracer techniques (Kelsall and Reid, 1970), and the results can generally be described in terms of simple convective diffusion models. However, predictive models showning the effects of system variables such as feed rate, etc., are not generally available. The influence of particle size is also poorly understood.

It should be recognized that the use of simple models for residence time distributions represents an approximation at best. It is known, for example, that the axial velocity of the particles varies continuously along the mill, at least in dry mills. The simple models, on the other hand, almost invariably are based on the assumption of constant velocity.

It is clear that the mean residence time is determined directly by the mass transport laws. The spread of the distribution is obviously determined by the internal mechanics of the mill, but the specific relationships are not generally known.

3.5.5 RESEARCH NEEDS

Research is needed in the following areas:

- The materials handling/mass transport aspects of comminution should be considered in more detail. The fundamental mass transport laws and the factors affecting residence time distributions need to be evaluated.

- Fundamental studies of the mechanics of grinding systems are needed in order to correlate such parameters as rates of breakage, with feed properties, mill geometry, and mill hold-up, among others.

3.6 Definitions of Efficiency

3.6.1 INTRODUCTION

The measurement of the efficiency of comminution has been debated for more than a century. New information obtained by careful investigators in the past 10 or 20 years has indicated that this controversy arose largely from imprecisely posed questions. In this section, efficiency and its measurement will be clearly defined in its several meanings. Thus, although the numbers and the methods used in obtaining them may be debated, the terms used in calculations will have a common meaning.

Comminution in general is the crushing or grinding of material to make smaller particles, but in specific instances its goal is clearly defined. The goal may be simply to make rock smaller to use as a foundation for a road. It may be to grind coal to a proper size for a coal-oil slurry; if the coal dust is too coarse it will sink and if too fine it will agglomerate. The goal may be to liberate inclusions; these may be desirable metals or minerals in ore which will be subsequently refined, or inclusions of sulfur in coal which are to be eliminated to reduce pollution.

A conceptual definition of efficiency would be the ratio of the desired goal to the cost of achieving it, the cost being in money or energy (work). Such a generalized formulation is essentially the cost-benefit ratio used in economic analysis and as such is a very useful quantity to a mining company. A scientific definition of efficiency is much more restricted. It is the minimum (or theoretical) energy required to achieve a certain goal, divided by the actual energy required. Thus effiency is dimensionless, since both numerator and denominator have dimensions of energy, and it can be neither greater than unity nor negative.

3.6.2 CRUSHING

We will first consider the case of simply crushing rock. A natural measure of the degree of crushing is the increase of surface area. It was shown in the section on Fragmentation Science (Section 3.3) that the two classical treatments of the theoretical energy required to increase the surface area lead to different laws:

$$\text{Von Rittinger:} \quad \text{Energy} = K\left(\frac{1}{D_f} - \frac{1}{D_i}\right)$$

where D_f and D_i are the final and initial particle diameters, respectively; and

$$\text{Kick:} \quad \text{Energy} = K \log \left(\frac{D_i}{D_f} \right)$$

where the K's of the two equations are different. Difficulties with both of the models have been pointed out. They include the use of single, ideal values for the initial and final sizes of the particles, instead of the distributions in sizes that occur, and glassy surfaces on finely ground material, which indicate plastic flow instead of clean fracture.

Bond and Wang (1950) developed a semiempirical theory of comminution which was developed further by Bond (1952, 1961). This theory led to a simple relationship for the energy of breakage which involved the difference between the reciprocals of the square root of the screen mesh size through which 80% of the particles would pass. The theory contains elements of both the von Rittinger and Kick theories in that it assumes that the energy acutally used in crushing and grinding is proportional to the length of the extension of the crack tips. Thus, it includes the energy initially stored as strain energy until a crack tip propagates and the particle fractures. At this point the strain energy is released as heat, although some of the strain energy may remain in the fractured parts of the particle. Any kinetic energy in the particles, resulting from the sudden fracture, would be converted to heat by collision with other particles or with the wall of the crushing device.

Others have modified Bond's formula and have shown that the square root of the mesh size is not the only suitable function. Implicit in this modified model is the theorem, demonstrated in the section on Fragementation Science, that in crack propagation half of the energy goes into elastic strain. In energy conservation this elastic strain energy is recoverably as work when released. In a practical process, however, this elastic energy is dissipated as heat, either by collisions between particles or by internal friction. This heat is low grade and is not useful for further comminution. However, it must be charged to the energy consumed in achieving the desired goal. The remaining half of the energy, i.e., that not lost to strain, is available for fracture; some of it can be lost to the device, to friction between particles, or to plastic flow. In sum, since only half of the crushing energy can go into fracture, the maximum efficiency in achieving new surface area can be only 50%.

The foregoing approach is misleading. It would be better to consider only the half of the energy that is available for fracture as the theoretical value, so that maximum efficiency can be 100%. It should also be recognized that in any practical compression of particles of different sizes, some will have sufficient stress to propagate cracks while others will have too little and will recover elastically. As mentioned, this recovered elastic energy is dissipated as heat and accounts for part of the input energy. Thus, the efficiency is not constant but varies with particle size distribution. What must be considered is a mill efficiency that is based on some suitable form of averaging and that recognizes that less than ideal conditions prevail. Only with this practical viewpoint can areas for improvement of efficiency in a total mill operation be delineated.

3.6.3 ASSOCIATED ENERGY

Carey and associates (1933, 1953) have developed a practical method for measuring crushing energy. A single layer of particles of common size was crushed between two plates in a press to a smaller size. The energy can be determined from the measured force and the known displacement of the crushing device. By doing this successively for smaller and smaller particles, an associated energy curve can be constructed for a given material; the energy to produce any given size from another, larger size can be obtained from the curve. Some examples of associated energy curves are shown in Figure 3-16. By this means, the minimum energy required to reduce a given mill feed size to the produce size can be determined. This permits the calculation of two types of efficiency: (1) the mill efficiency and (2) the grinding efficiency.

$$\text{mill efficiency} = \frac{\text{increase in associated energy from feed to product}}{\text{total energy of mill}}$$

$$\text{grinding efficiency} = \frac{\text{increase in associated energy from feed to product}}{\text{energy supplied for grinding}}$$

Note that associated energy is used in the numerator of these expressions. The theoretical energy of the new surface area could be used, but, as pointed out above, this is unachievable and therefore is an unrealistic standard of measure.

The two efficiencies above have very important distinctions, for there are a variety of sources of energy loss. For example:

i. The energy entering a mill does not all go to the grinding process.

ii. Even the energy delivered to the grinding device is not all involved in the grinding or crushing process. There are large losses from friction, both in the particular device and from interparticle friction.

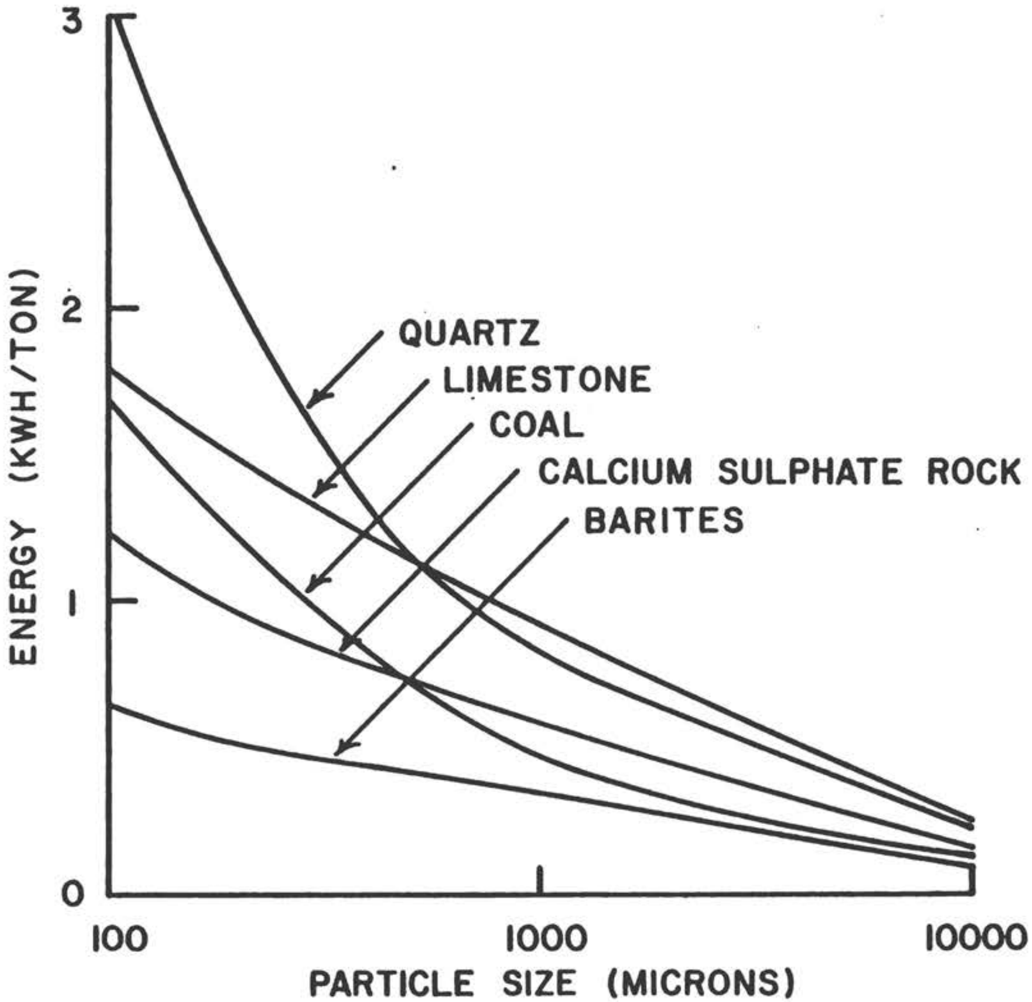


FIGURE 3-16 Associated energy curves for various minerals.
(After Carey and Stairmand)

iii. The packing of fine particles around a larger particle will reduce the effectiveness of the crushing process through interparticle friction. The fines may be removed by a classifier, but this requires energy. Thus, energy is lost in attempting to increase the efficiency by removing the fines.

Energy losses in mill operation (point i above), and in device friction (point ii) are readily determinable. Energy losses from fine particle friction must be determined by experiment. The Carey-Stairmand curve for associated work of particle reduction was performed for a single layer of noninteracting particles and therefore represents the ideal case. What one would like to have is a series of associated energy curves for various size mixtures of interacting particles. One could then arrive at an optimum energy for mill operation which reflects the trade-off between the classifer energy required to separate the particles and the energy lost by limiting classifier operation. Note that such an operating optimum will be less efficient than that indicated by the associated energy curves.

3.6.4 CRUSHING EFFICIENCY LOSSES

Toward the foregoing goal, Carey and Stairmand (1953) investigated the energy of crushing for less than ideal cases. In the first experiment they studied the effect of a deep bed of particles, a situation usually present in a ball mill. The same type of slow crushing experiment described above was performed, but with a bed five or six particles deep instead of a single layer. Single layer particle crushing was taken to be 100% efficient, and measurements were made of the relative efficiencies, i.e., the ratio of the amounts of energy required to achieve the same degree of plate separation in crushing the multilayer particles to reduction ratios of 1.16, 1.28, and 1.40 (the reduction ratio is defined as the separation of the plates before crushing divided by the separation after crushing).

The results are shown in Figure 3-17. It is seen that for a reduction ratio of 1.32, the relative efficiency of crushing a multilayer of particles is 72%. Note that crushing ratios in a ball mill are not well defined, but are believed to range up to 1.5 or even 2.0.

In a second series of measurements, the particle size was not uniform--a distribution of particle sizes corresponding to a natural grading was used. Figure 3-18 shows the results compared with 100% efficiency for single-layer crushing. It is seen that for the same reduction ratio as above, i.e., 1.32, the efficiency relative to the single layer was 57%.

A third experiment was conducted similar to the one that yielded the results in Figure 3-18. In this case, however, the resulting particles were measured to find the percentage smaller than 200 mesh (76 μm), and these were remixed with the bed. Figure 3-19 shows the reduction of efficiency with the increase of fine particles in the

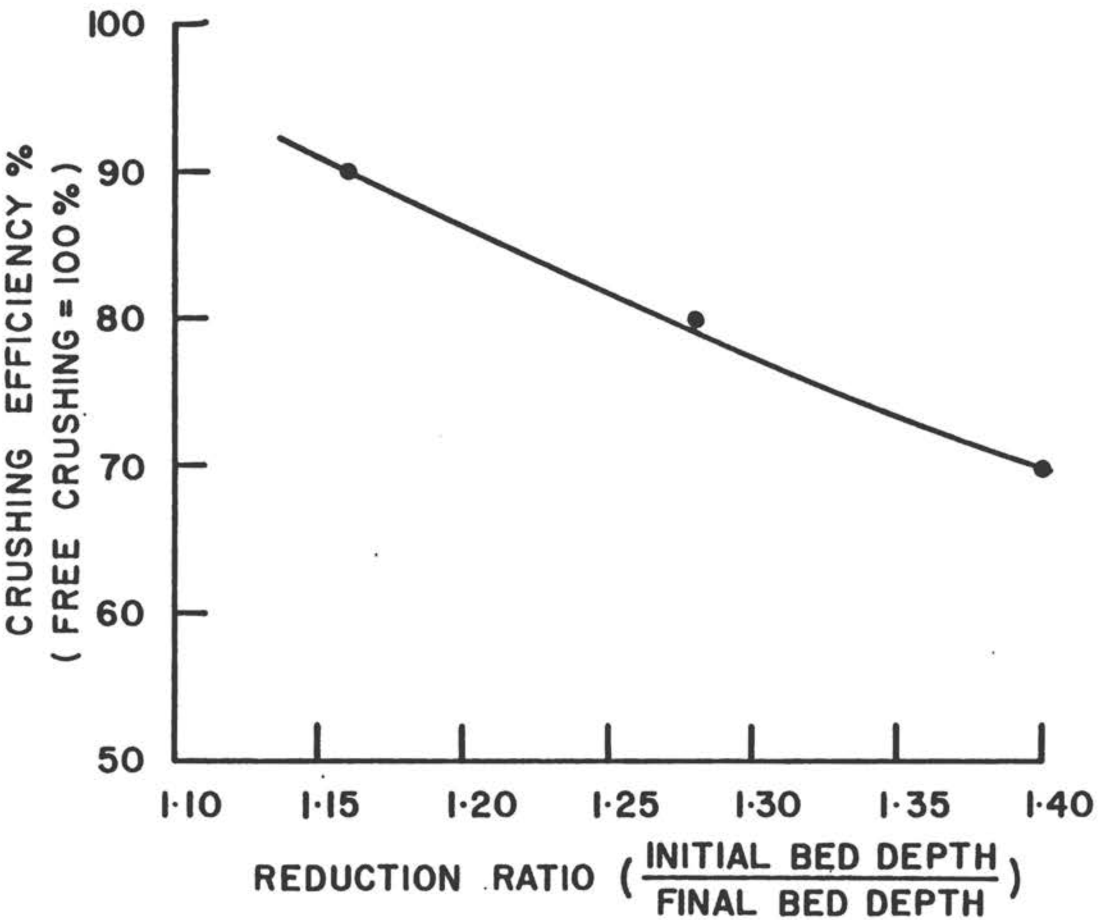


FIGURE 3-17 Efficiency of crushing in deep beds of uniform particles. (After Carey and Stairmand)

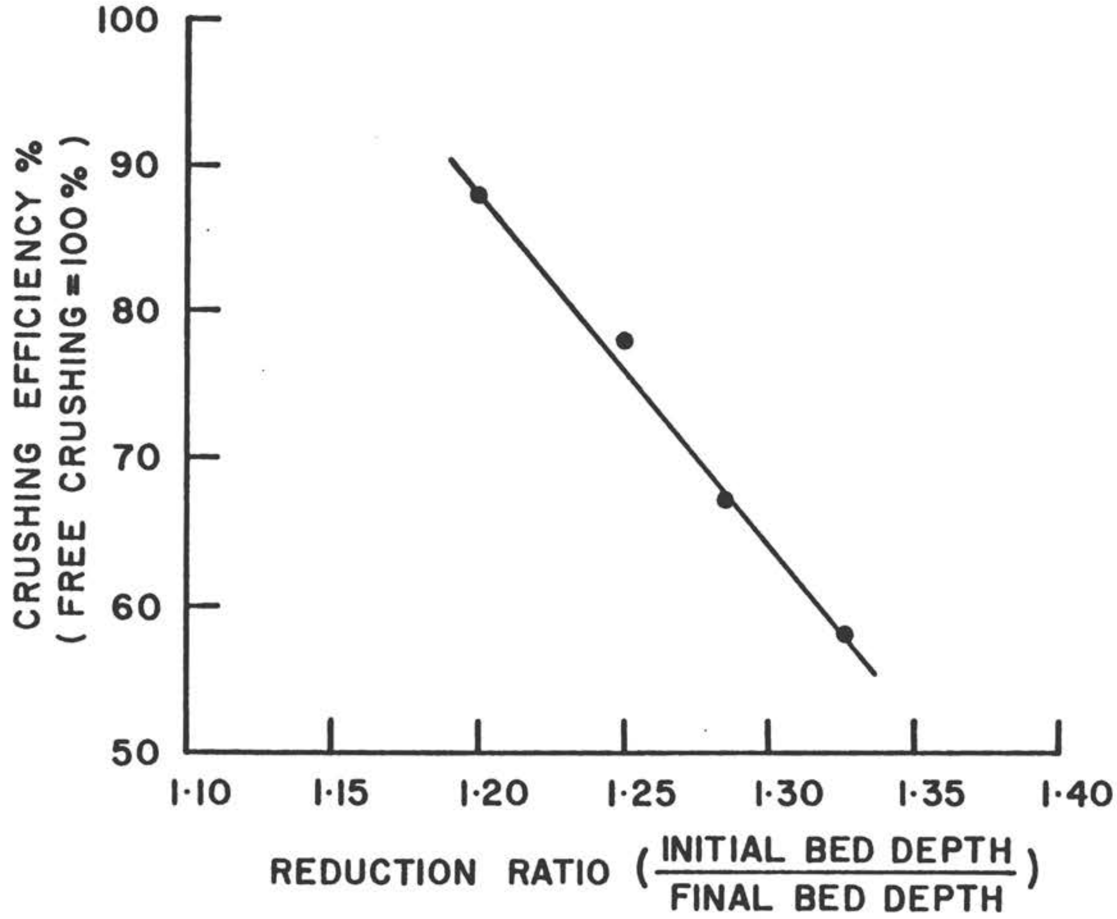


FIGURE 3-18 Efficiency of crushing in deep beds of particles of size distribution associated with natural grading. (After Carey and Stairmand)

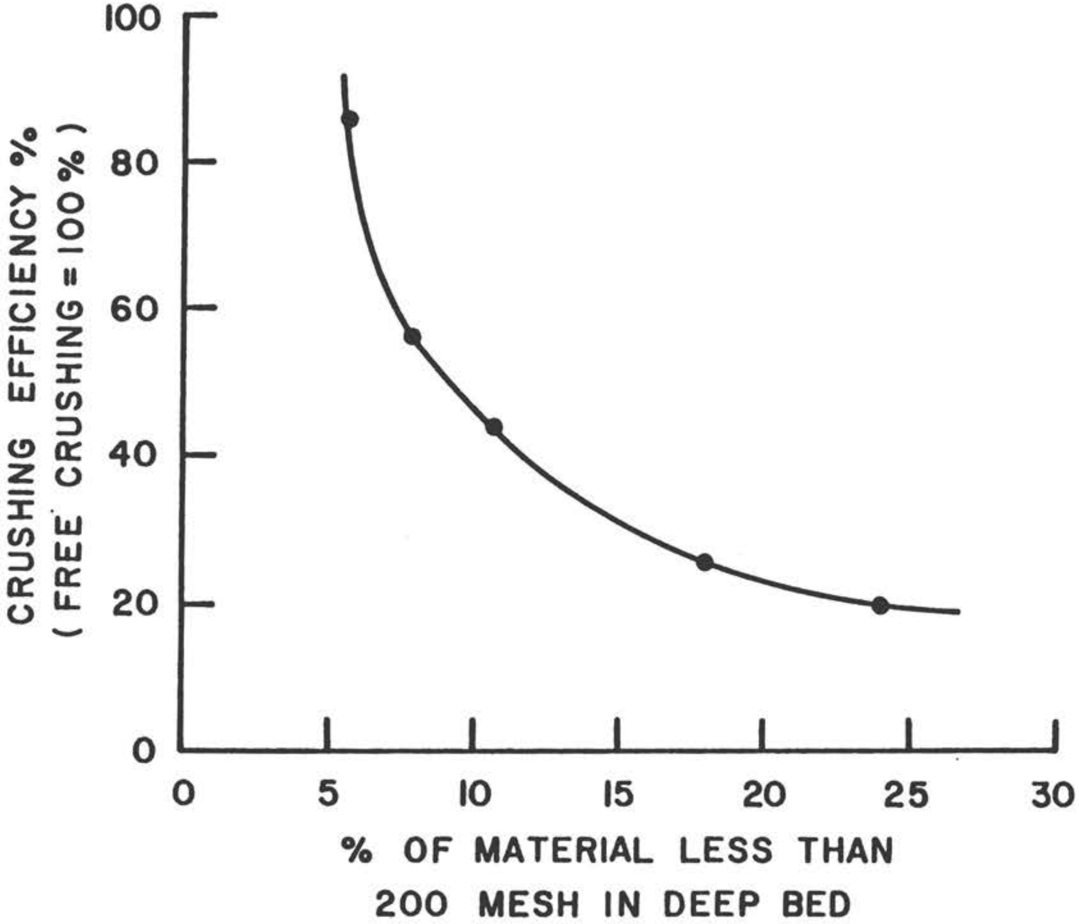


FIGURE 3-19 The effect of fine particles on crushing efficiency. (After Carey and Stairmand)

bed. It is seen that the efficiency is rapidly reduced to 20% at a percentage of fine particles believed present even in closed circuit operations that employ classifiers.

The generation of fine particles in quantities of 15% or more takes place in only a few minutes in a ball mill. According to Figure 3-19, therefore, the efficiency of a ball mill cannot be intrinsically very high even under optimum design with classifiers. However, it is obvious that a crushing device that approaches the free particles condition will have the highest efficiency.

Recently it has been shown, at least for small glass spheres, that crushing energy can be lowered if the sphere is compressed and sheared simultaneously (Schonert, 1979). The sum of the work for compression and shearing is the energy input. At the present state of the art the efficiency can only be compared to that of the associated energy of free crushing. Thus, if a lesser energy is required, a higher efficiency can be calculated over that for simple crushing. As devices are developed for performing this mode of comminution, a minimum associated energy may be developed in the laboratory so that the efficiency of a device can be ascertained.

The same reasoning applies to the other comminution devices, whether they be ultrasonic, pulse heating, or a novel invention. The difficulty is in establishing a comparable model, such as free crushing, when the particle size distribution differs considerably from that obtained by free crushing. Statistical analysis of the energy for each size group can be used, however, and it is expected that reasonable results can be obtained.

3.6.5 MILL EFFICIENCY

There is still another way to consider efficiency, and that is the energy losses in the mill itself. An example (Lowrison, 1974) of percentages of energy consumption for a dry ball mill is given below, although it should be noted that there may be differences resulting from maintenance by the user or even initial difference between makes.

Ball Mill Energy Consumption

Bolt Friction	4.3%
Gear Losses	8.0
Heat Losses from Drum	6.4
Heat Absorbed by Air Circulation	31.0
Heat Absorbed by Product	47.6
Energy Unaccounted (2.1%) & product reduction 0.6	2.7
TOTAL	100%

The associated energy for product reduction during this operation was 0.6%. Although 2.1% of the energy is not accounted for, which is a reasonable experimental error, it is seen that 12.3% (8 + 4.3) of the energy of operation is lost by the device itself. The device's efficiency for delivering energy to the material, therefore, is 100 - 12.3 = 87.7%. This measure of efficiency is useful for comparing similar devices made by different manufacturers or the effects of age, maintenance, or wear. Again, however, from the viewpoint of comminution the device has an efficiency of only 0.6%.

3.6.6 COMPARISONS OF SIZE REDUCTION EFFICIENCY

We may conclude from the dicussion thus far that the most realistic comparisons of efficiency of size reduction are the following:

1) In a calculation of grinding efficiency, the numerator of the fraction should be the increase in associated energy of free particle size reduction from feed to product.

2) The denominator should be either

    a) The energy actually delivered to the grinding device, when the net grinding efficiency of a device is to be considered; or

    b) the total energy applied to the system when a classifer is used. This will give the net grinding efficiency of the system and can be compared to (a).

3) When comparing two equivalent devices, an independent measure of frictional losses within the devices can be made, and thereby the relative efficiency of energy delivered for size reduction can be obtained.

4) Efficiencies of devices operating on different materials can be compared only if the materials have the same brittleness.

We may also conclude from the discussion that the closer a free particle situation can be approached, the higher will be the efficiency of size reduction. Other recent considerations (Schonert, 1979) suggest that the free crushing energy provides the most realistic basis of efficiency calculation and point to the need for research into devices that approach the free crushing condition and yet maintain high capacity.

3.6.7 EFFICIENCY IN LIBERATION

Up to this point we have considered only the efficiency of size reduction, not the efficiency of attaining a modified goal. One of these goals might be the attainment of a narrow size distribution of the product, as might be required for coal. The fines that are discarded take away considerable surface energy. If free crushing can simulate the size distribution of the product, one might consider efficiency to be energy to achieve this distribution by free crushing divided by the actual energy, multiplied by the fraction of the total free crushing energy of the particles of the desired size that are retained.

$$\text{efficiency of achieving a desired product size} = \frac{\text{associated energy of distribution}}{\text{energy to achieve the distribution}} \times \frac{\text{associated energy of desired fraction}}{\text{associated energy of distribution}} = \frac{\text{associated energy of desired fraction}}{\text{energy to achieve the distribution}}$$

It is seen that the efficiency is proportional to the fraction of the desired size but, when terms are cancelled, efficiency is equal to the associated energy of the desired fraction divided by the total energy required to achieve it. If it is the fines that are to be discarded, the efficiency of the process can be very low, since the fines have the most energy (Figure 3-16).

The estimation of efficiency of liberation has not been clearly defined in the general case. In the liberation process, inclusions of the desirable material are to be separated from an ore. For simplicity of discussion we will refer to these inclusions as the metal to be separated, although it may be a mineral. An assay of the ore indicates the percentage of the metal it contains. The metal inclusions have some size distribution, $N(X)$, where N is the fraction of particles of size X. This distribution can be determined by measuring the sizes and numbers in polished faces of the ore.

The procedure is to crush the ore so that the metal particles are freed for further processing. Suppose, for example, that the particle size distribution of most of the metal in an ore ranges from 10^{-4} to 1 cm. A hypothetical distribution, $N(X)$, is shown by the curve in Figure 3-20. To liberate most of the metal, the ore must be ground to a size distribution in which very few, say 5%, of the particles are larger than 10^{-4} cm. Such a distribution is indicated by curve 1 of Figure 3-20. It is seen that the mean size of the ground material is considerably smaller than required to liberate metal particles 10^{-4} cm in size. Energy of grinding is therefore wasted.

In order not to waste energy, some of the metal to be liberated must be discarded. This is shown in curve 2 of Figure 3-20, which is the size distribution produced by another grinding time or even by a different type of grinding. Curve 2 is drawn so that point X_2 on the

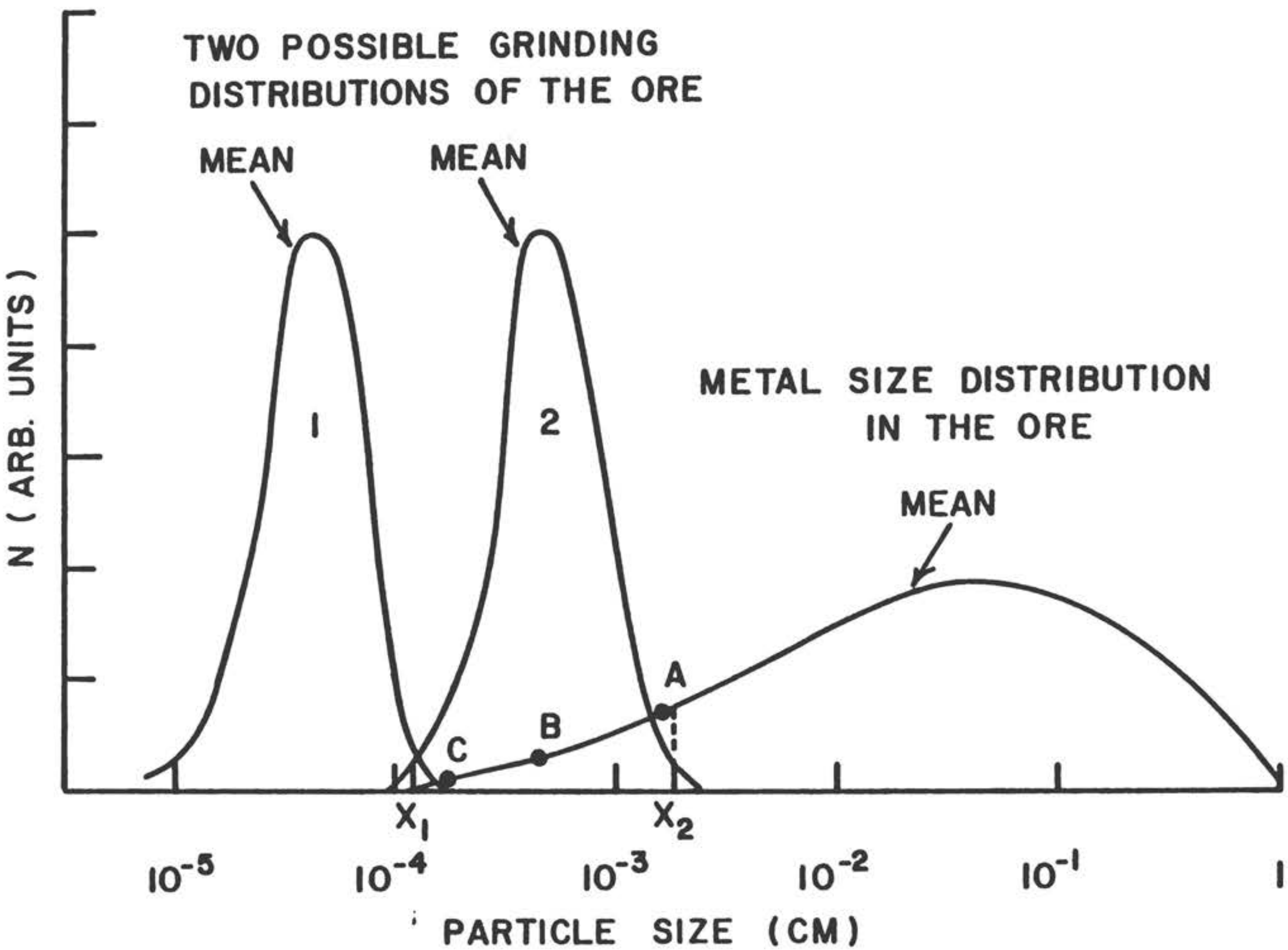


FIGURE 3-20 Broad curve is a hypothetical size distribution of metal in an ore. Curves 1 and 2 are hypothetical size distributions for two different grinding devices or times in the same device. X_1 and X_2 are the upper limit sizes at 95% of distributions 1 and 2, respectively.

abscissa indicates that 95% of the ground material is of size X_2 or smaller. Therefore, all of the metal of sizes larger than this, i.e., to the right of the dashed line, will in principle have been liberated. A metal inclusion of size A has a high probability of having been liberated, but may be enclosed by a piece of ore of size X_2 and therefore not liberated. A metal inclusion of size B has had about a 50% chance of being liberated, while one of size C is effectively discarded with the tailings.

Decisions on recovery fraction vs. energy can be made by elementary statistical analysis when the size distribution functions are known. It is therefore extremely important to characterize the size distributions of both the ground material and the metal in the ore. Without such information, cost-benefit decisions cannot be made. An additional complication is that the size distribution of the metal in the ore changes, sometimes rapidly. A method of continuously monitoring this distribution and adjusting the grinding size distribution accordingly can lead to considerable cost savings.

Efficiency in its usual sense cannot be established at this time because of inconsistencies in the dimensionality of the terms. In simple crushing, either the theoretical or the associated energy of creating new surface area can be divided by the actual energy to obtain efficiency as a dimensionless quantity not exceeding unity. In the case of liberation, one has dissimilar measurements: the fraction liberated and the energy required by a given device to do so. If one of these terms is divided by the other, the result is not dimensionless; although the result is extremely useful in a cost-benefit analysis, the amount liberated per unit expenditure of energy is not the efficiency in its restricted sense of being a dimensionless quantity between zero and unity.

What can be examined, however, is the efficiency of a given grinding as a function of the amount liberated. A hypothetical plot of efficiency vs. liberation is shown in Figure 3-21. The size distribution of the ground ore is determined, and the efficiency of a given method of grinding or time of grinding is obtained by the free crushing model discussed earlier. If the size distribution of the metal in the ore is known, through a graph such as that of Figure 3-20, the efficiency of grinding for a given percentage of liberation can be plotted as in Figure 3-21.

As indicated in Figure 3-21, the crushing efficiency would be expected to be high initially because mechanically weak particles are broken first. The efficiency decreases but slowly after that because there is still very little accumulation of fines that can pack around the larger particles and absorb the crushing energy. As these fines accumulate, the efficiency drops rapidly, as shown in Figure 3-19.

A plot like Figure 3-21 will assist a mill operator in deciding what percent liberation to achieve. If, in operation of a mine, the size distribution of the metal inclusion changes, the fact will be reflected in the curves of Figure 3-20. For example, if the mean size of the metal inclusions becomes smaller, a finer grind will be required to liberate the same percentage and the curve of Figure 3-21 will move to the left and change shape to some extent. The sacrifice in

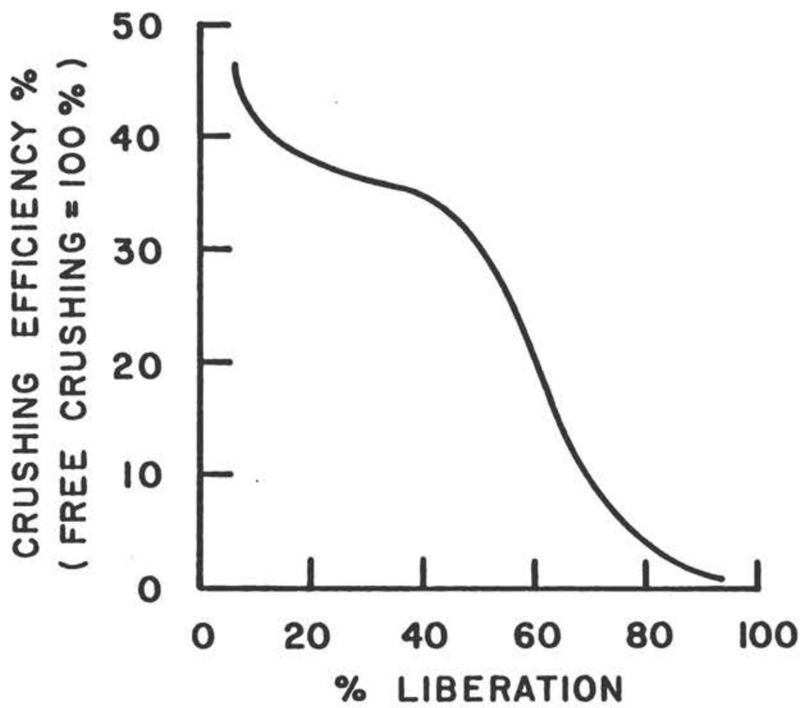


FIGURE 3-21 Hypothetical efficiency vs. percentage liberation from an ore.

efficiency to maintain the same percentage liberation will be immediately observable. A cautionary note is in order, however. The size distribution functions of grindings 1 and 2 in Figure 3-20 may be quite different, and each would be expected to differ from that obtained in the Carey-Stairmand free crushing model. Therefore, statistical summations must be made for each.

It may be concluded that, once having obtained a reasonable approximation of the efficiency of reducing a given ore to different sizes by the associated energy method, it is important to monitor frequently the size distribution of the metal inclusions. Such monitoring is needed so that changes in grinding efficiency for fractional liberation can be estimated continuously.

3.6.8. RECOMMENDATIONS

From the foregoing consideration of the definition of efficiency, two factors emerge:

1. Energy utilization in comminution is roughly distributed as follows:

	%	Relative Value
Machine losses	13.0	20
Heat losses	85.0	130
Size reduction	0.6	1

Both the heat losses and machine losses are related to machine design. It is clear that the potential for energy saving is far greater from research into mechanical design and machine dynamics than it is from research into the physics of fracture itself.

2. There is a complex relationship between overall energy-effectiveness and particle liberation as a function of size. Research is needed to develop practical definitions and measurement techniques for liberation and to establish methods for using this information for minimizing overall grinding and separation energy (cost) requirements.

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CHAPTER IV

PRACTICAL CONSIDERATIONS

4.1 Introduction

Although the theoretical considerations discussed in Chapter 3 are necessary to clarify the fundamentals of comminution and to give meaningful direction to applied research, it is important that the ultimate practical objectives of comminution as a unit operation be kept clearly in mind. Several research areas relate more directly to practical objectives than to theoretical understanding. These areas are the subject of this chapter.

Section 4.5 deals with the large cost and energy investment associated with the metal losses caused by corrosion and wear of the liners and tumbling media in comminution systems. Because of the commonly acidic nature of ores--particularly sulfides--it has been common practice in the industry to add any lime required for the separation process to the grinding mills, since this reduces media consumption.

Considerable work has also been done to clarify the effects of other chemical additives or grinding aids, particularly those that may influence any of the basic mechanisms involved in comminution devices. Previous research and research needs in grinding aids are discussed in Section 4.6.

A very important part of closed circuit grinding systems is the classifier, which separates finished product from the coarse material that is returned to the system for further grinding. Section 4.3 covers classification devices for both dry and wet grinding and identifies the areas requiring further research.

It is impossible to operate any process efficiently without effective monitoring devices; given the ability to measure the key process variables, knowledge of the dynamic characteristics of the process is necessary to implement appropriate automatic control and process optimization procedures. There has been considerable work in the mineral industry in the past 20 years in instrumentation development, control applications, and optimization, and this work is summarized in Section 4.4. Although the practical need for design and scale-up procedures has been resolved in the past by empirical methods, large safety factors are required. The potential and research needed for improved design and scale-up procedures, based on our improved fundamental knowledge and more precise mathematical representation of comminution, are discussed in Section 4.2.

Autogenous grinding is a form of tumbling mill system that has attracted a great deal of attention over the years, and studies in recent years have led to more precise understanding of the perfomance characteristics. Research of autogenous grinding and areas for further research are covered in Section 4.2.

The final two sections of this chapter review work directed at new methods of comminution (Section 4.7) and at reducing the overall energy input into comminution by exploring the optimization of existing methods and application of new techniques for size reduction at the mine site (Section 4.8).

4.2 State of the Art

4.2.1 INTRODUCTION

The trend in mineral processing plants in the past two decades has been to install increasingly large individual grinding units to minimize capital costs. Fewer units are required to process a given tonnage and smaller buildings are required. However, the larger units in typical large-tonnage operations do not decrease specific grinding energy consumption. In following the trend to larger units, it became apparent that the operability and energy-efficiency of rod mills and ball mills are size-limited. A most pronounced trend in the past two decades has been the installation of large numbers of autogenous and semiautogenous grinding mills, among them the largest single grinding units of any type. Gearing problems in very large mills appear to limit further pronounced increases in size.

With alternative grinding systems available, the designer of mineral processing plants is faced with selection of a system, scale-up from laboratory or pilot plant tests, and design of the plant, recognizing the constraints of minimizing capital costs and of attaining lowest operating cost. The text that follows briefly describes the grinding units employed in processing plants for bulk mineral operations. Included are sections on selection of grinding system, scale-up, and research opportunities.

4.2.2 COMMINUTION DEVICES

4.2.2.1 Rod Mills

The largest rod mills now operating are about 15 ft in diameter by 20 ft long with 2300 hp connected. Kjos (1979) identifies two factors that appear to limit further size increase. The first limitation is an inability to manufacture grinding rods longer than 20 ft which will provide as good service as rods used now. Rods must resist not only abrasive wear, but also bending and premature breakage, and must wear uniformly longitudinally as well. The second limitation is believed to result from inadequate slurry transport characteristics within the tumbling charge, perhaps because of the large reduction ratio and the smaller void space in the rod charge as compared to a ball charge. These factors could limit the rate at which power can be transmitted to the ore.

4.2.2.2 Ball Mills

In conventional grinding circuits, a ball mill grinds the rod mill product to a finer size. The rod mill is usually open circuit, whereas the ball mill is close-circuited with a size classifier. Kjos (1979) also points out that mill manufacturers recognize that as the diameter increases to about 16 ft, grinding efficiency decreases. Larger mills could be manufactured but those factors that influence energy transfer from the tumbling charge to useful work in size reduction apparently are not yet understood.

The largest ball mills in mineral processing plants are the 17 ft x 41.5 ft units at Eveleth Taconite with 6900 connected hp. At Bougainville, single stage ball mills 18 ft x 21 ft were installed with 4200 connected hp. Steane and Hinckfuss (1978) indicated that these mills were about 20% inefficient as measured by the work index. The work indices observed from pilot plant tests with a 5.85 ft x 4.87 ft ball mill correlated well with standard grindability tests. The authors concluded that the inefficient grinding perhaps resulted from inadequate mixing of the ore pulp, particularly the coarse material, in the tumbling ball charge. Subsequently, a ball mill of the same diameter but 3 ft longer with 5360 connected hp was installed; throughput increased proportionately at no further decrease in efficiency. The operators at Bougainville have increased crude throughputs via improved classification and mill operating time, making up for the decreased grinding efficiency.

The grinding inefficiency described above is apparently not universal. Pinto Valley's 18 ft x 21 ft ball mills are reported to have slightly exceeded design throughput. If the present understanding of factors influencing grinding efficiency is not sufficient to predict responses in 18 ft versus 16 ft diameter mills, an empirical resolution may be necessary with a side-by-side installation of two such mills in one plant.

Single stage ball mill grinding after tertiary crushing as at Bougainville has recently been applied at a number of other nonferrous operations as listed by Kjos (1979). Shoemaker and Gould (1978) have reported that the limitation of rod mill size and hence throughput has resulted in the selection of single stage ball milling to minimize the number of grinding units needed to process a given tonnage.

For dry ball milling, Rowland and Kjos (1978) propose an inefficiency factor of 1.3 in scale-up based on the Bond Work Index. Dry ball milling may have specific advantages in certain cases, such as regrinding iron ore concentrate at Wabush Mines with subtantially less ball consumption than experienced in wet grinding. Another specific situation that calls for dry grinding is grinding an earthy type iron ore to the proper size consistency for pelletizing. In such cases, no concentration steps follow size reduction, and dry grinding is preferred to avoid having to dewater a slurry with very poor filterability. Another application is the grinding of coal for use in firing steam boilers in electrical generating plants.

4.2.2.3 Autogenous Grinding

A number of installations of autogenous grinding have been made with varying degrees of success in meeting operational design criteria. The largest units currently operating are the 36 ft x 15 ft mills at Hibbing Taconite, each driven by two 6000 hp motors. At the Empire Mine, the first autogenous grinding mills were 24 ft x 8 ft long, the second group 24 ft x 12 ft long, and the very recently activated third group 32 ft x 15 ft long. The connected horsepower increased progressively from 2200 to 3450 to 8750 hp. It is too early to compare the specific energy consumptions of the three sizes of mills.

The use of autogenous grinding offers the possiblity of lower capital costs because there is no need for a fine crushing plant with crushers, screens, and conveyors. Currently the selection of autogenous grinding is probably being scrutinized very carefully because the energy consumption may be from 10% to as much as 50% more than for conventional grinding. However, in comparing autogenous to conventional grinding, media consumption should be considered as well as consumption of lifters and liners. If one assumes that 1.7 kWh/lb (electrical equivalent) are used in the manufacture of grinding media, the energy equivalent of 1.5 to 2.0 lbs of grinding media per ton of crude ore is equivalent to 2.5 to 3.4 kWh per ton of crude.

Two operations with up-to-date conventional grinding-concentration plants followed by parallel circuits with autogenous grinding and concentration provide realistic comparisons of capital and operating costs. Fahlstrom and coworkers (1975a, 1975b) reported that the capital cost of the Aitik autogenous plant was about 20% less than for the crushing, rod-mill, pebble-mill plant. Operating costs through concentration in the autogenous plant are 25% lower, in spite of a 25% higher energy consumption in grinding.

At Palabora, the new section of the plant equipped with the 32 ft x 15.5 ft autogenous mills, driven by two 6000 hp motors each, was stated by Betts (1979) to have an operating cost about 71% of that for the conventional plant. The cost comparisons included grinding energy and operating maintenance labor and supplies. The following summary illustrates the differences in grinding energy consumptions, liner wear, and media consumptions for the two types of grinding, including the auxiliaries in each case and the fine crusher plant.

	Autogenous	Conventional
Grinding energy--kWh/ton	11.5	9.5
Liners--gm/ton	52.4	30.9
Media--gm/ton	-	353.0

The energy consumption for the autogenous system is 2 kWh/ton higher, but net metal consumption for the conventional system is higher by about 331 gm/ton, which is equivalent to about 1.2 kWh/ton. Operating time in autogenous plants is usually less than for conventional plants but Betts (1979) reported an operating time of 91% in 1978, with expectations of getting to 96% once the problems with the mill shells are corrected.

Autogenous milling is suited to specific types of ores; the decision to employ this method of grinding should be based on a realistic evaluation of test data. As reported by Villar and Dawe (1975), the Tilden autogenous grinding circuits employ intermediate cone crushers to crush the excess pebbles not needed for media in the pebble mills. These pebbles are crushed to about -3/4 in before being returned to the primary mill. When the excess pebbles are not removed from the circulating load, feed rates drop significantly and grinding energy consumption increases. The unsuitability of many ores for autogenous grinding has led to many installations of semiautogenous primary mills, which also eliminates the need for a fine crushing plant.

4.2.2.4 Semiautogenous Grinding

Semiautogenous grinding appears to be more universally suited to a variety of ore types than does autogenous grinding. In some cases, plants were designed for fully autogenous grinding, and because they did not meet design operating needs or in order to surpass design, have been converted to semiautogenous grinding mills. However, such a midstream change is not ideal, because adding a 5 to 8% ball charge to the mill reduces the ore charge. With such a reduced ore charge, the 4 or 5 in. balls cascading against liners and lifters could cause excessive breakage, resulting in increased downtime for maintenance. A semiautogenous mill should be designed as such, so that adequate volumes of crude ore can be maintained in the mill at all times. The Palabora mills previously described were designed to accommodate a ball charge if required.

Some unique situations exist for which semiautogenous grinding is a good selection. For example, Kjos (1979) indicates that at Pima, the clayey ore would be difficult to process through a fine-crushing plant. Another case might be to avoid operating a fine-crushing plant in a situation that requires extreme measures of dust control.

In semiautogenous mills, the cost and energy associated with the consumption of grinding media, which can be of the order of 0.5 to 1.0 lb/ton, and increased wear of liner-lifter materials must be considered. Operating time in semiautogenous milling is considered usually to be substantially less than for conventional grinding circuits (85% vs. 95%), but McManus (1979) reported that the 32 ft x 15 ft mills at Lornex averaged 94% operating time in 1978. The use of a mechanically powered liner-lifter handler with articulated arms extending into the mill through the feed-end opening has undoubtedly reduced mill maintenance time and improved safety in the handling of heavy metal pieces. It is usually assumed that wear of media plus liners is higher for semiautogenous grinding. However, Bassarear (1980) pointed out that, for 1974 through 1977, liner wear and also total steel wear were less in the semiautogenous section of the plant than in the conventional section, including fine crushing.

4.2.2.5 Crushing

There does not seem to have been much new in the past 20 years with regard to crushing units. Gyratory crushers of the 60 in. x 110 in. size have been common for a number of years. In fine crushing, some manufacturers are proposing 10 ft rather than 7 ft cone crushers to get more throughput for a given machine, thus reducing capital costs. Regarding energy utilization, current practice is to use the crushers at maximum capacity. Thus, a mineral processing plant should be designed to maximize throughput in the crushing sections regardless of the operating schedule of the concentrator.

4.2.2.6 Roller Mills

Roller mills have three or four vertical wheels, about 3 ft thick and up to 8 or 10 ft in diameter, which turn in a fixed circle. Grinding takes place between the rollers and the horizontal wear table. Roller mills are air-swept and in most instances use waste heated gases from an associated process. The roller mill can be fed a material with 10-12% moisture, provided the material is free-flowing. The rollers turn freely on their shafts, and the table is powered to turn.

A typical use for a roller mill is grinding raw materials for cement manufacture; limestone and shale plus possibly small proportions of iron oxide and quartz are ground together. In such a case, the kilowatt-hours per ton are less with the roller mill than the Bond Work Index would predict for wet grinding in a ball mill. The lower kilowatt-hours per ton with the roller mill is notable because dry grinding in a ball mill calls for a 30% increase in grinding mill energy as an inefficiency factor. Because of high maintenance costs for the tires and table, the roller mill is not particularly suitable for hard ores, such as those containing appreciable amounts of quartz.

Roller-type mills are frequently used in electrical generating plants to grind coal to about 90% - 100 mesh for firing steam boilers. An example is the Raymond bowl mill. The wheels in the Raymond mill are inclined at about 45 degrees and have a profile at the periphery which matches the profile at the circumference of the bowl in which the rolls turn. In this case, the bowl is powered to turn, and the spring-loaded rolls turn freely on their shafts.

4.2.2.7 Hammer Mills

Hammer mills probably are used most frequently in the size reduction of more friable materials such as coal. A typical application is crushing metallurgical coals to about 80% - 1/8 in. for use in making coke. Another application is grinding coal for steam boilers in electrical generating plants. In this case, the hammer mill is described as an attrition mill which pulverizes the coal to about 90% - 100 mesh.

4.2.2.8 Grinding Mill Liners and Lifters

The replaceable wearing surfaces within grinding mills prevent wear and corrosion of the structural shell. That part of the liner wear surface that projects by design from about 4 in. to about 12 in. above the lining is referred to as the lifter. As the mill rotates, the lifters raise the charge of ore plus grinding media and impart the cataracting and cascading motions to the charge. Within the moving charge the actions taking place can be described as: Impact, compression, breakage, abrasion, fracture, attrition, material transport, and screening--the net result of which is size reduction.

In rod mills, liners and lifters are usually cast in one section with one "lifter wave" per section. In ball mills, integral liner-lifter sections are common, with either a double-wave lifter or a single-wave lifter for mills charged with balls about 2 1/2 in. in diameter or larger. Mill manufacturers have devised the criteria used to relate the number of lifts in the circumference to the diameter of the mill.

For autogenous and semiautogenous mills, liners and lifters can be integral or separate sections. Since lifters wear faster than liners, the common practice is to use separate lifters, but in some semi-autogenous mills integral liners and lifters are used.

Alloy steels or wear-resistant cast irons continue to be commonly used for liners and lifters. Within the past two decades, rubber has been used very successfully in ball mills and pebble mills. In a practice originating in Scandinavia, rubber liners and lifters have been used in autogenous mills with very good success and have been installed in the three 27 ft x 8 ft primary mills at Sherman Mine. Nilsson (1979) describes the basic design parameters for rubber linings regarding spacing, height, and profile of the lifters. Apparently the use of rubber linings in rod mills initially was limited to units of 7-8 ft diameter, but Hedlund (1979) claims such linings have been used in rod mills up to 12 1/2 ft in diameter. Hedlund (1979) also described the first application of rubber linings in the 22 ft x 7 ft autogenous mills at Boliden's Vassbo Mine. The mill was first completely rubber lined in 1967; subsequent improvements in rubber quality and lifter bar design resulted in an improvement in operating time of about 5% over the old practice with steel liners.

If rubber has become cost competitive with ferrous wear materials for grinding mill liners, one would expect that energy associated with a pound of rubber liners to be less than the 1.7 kWh/lb associated with ferrous materials. However, Bollinger (1980) has estimated the energy consumed in ball-mill rubber liner compounds as follows:

Raw Materials	36,400 Btu/lb
Curing	12,000
Fuel value	14,000
TOTAL	62,400 Btu/lb

--that is, 18.3 kWh/lb fossil fuel equivalent or 6.1 kWh/lb electrical equivalent.

The cost advantage of rubber has to be derived from superior wear life and greater ease of replacement, both of which would decrease maintenance time and increase mill availability. Other benefits of rubber linings are less noise in the immediate vicinity of the grinding mills and, as with the 27 ft x 8 ft mills at Sherman, about 100 tons less weight on the mill bearings.

Other polymeric wear materials, such as polyurethane, have been considered as liner material but without success to date. Polyurethane has served well as a wear material in the discharge section of autogenous mills and as screen deck material for vibrating screens.

4.2.2.9 Novel Liner Configuration

San Manuel has installed square spiral liners in its grate discharge ball mills, in both the 10 ft x 10 ft and the 12 1/2 ft x 14 ft units. The liners sections have a generally square appearance and rounded corners. Successive pairs of circumferential sections along the length of the mill are progressively offset by about 15°, giving a spiral effect. The square liner concept is illustrated in Figure 4-1. The spiral liner concept is not new and is intended to keep the coarsest grinding media at the feed end of the mill.

Korpi and Dopson (1979) described the results obtained in a 10 ft x 10 ft mill equipped with spiral liners. Apparently about 15% of the mill volume is taken up by structural steel fillers between the mill shell and the spiral liners. Energy consumption was stated to be

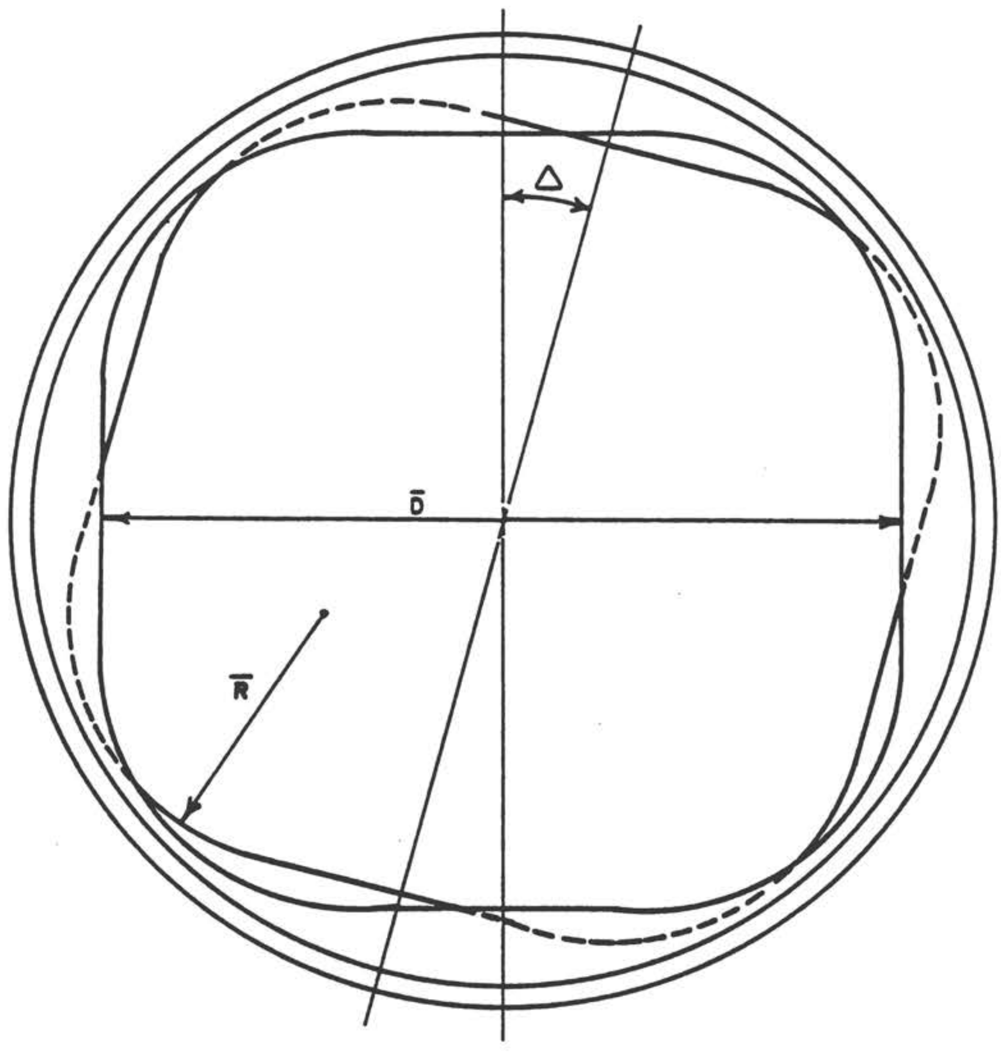


FIGURE 4-1 RELATIONSHIP BETWEEN $\overline{R}$ AND $\overline{D}$, AND THE OFFSET ANGLE, Δ, IN ANGULAR SPIRAL LINER SYSTEM.

reduced by about 17% and ball consumption by about 18%. A drop in throughput of about 1.7% was indicated, but improvements were expected to be made. With a decrease in mill volume of about 15% and only a slight reduction in throughput, it appears that more useful grinding work is obtained with this liner configuration. The ball mills at San Manuel have grate discharges, and it will be interesting to see if the promise indicated by the San Manuel testing can be realized with overflow ball mills. One such installation has been reported to be scheduled for a 16 1/2 ft x 19 ft overflow ball mill.

4.2.3 INTERGRATED GRINDING - IMPROVED SIZE CLASSIFICATION - MINERAL SEPARATION

One area that bears looking into in any mineral processing plant is the use of multistage grinding coupled with intermediate stages of separation, whether these be to separate economic mineral or waste mineral. In the magnetic taconite industry, with size reductions in rod mills, primary ball mills, and regrind ball mills and with magnetic separation following each stage of grinding, studies are under way to see if a dry cobbing of the rod mill feed could further reduce grinding energy. In copper operations, a bulk sulfide flotation frequently is followed by regrind prior to the flotation to make final mineral separations.

A particular problem exists in iron ore processing with cyclone classification in the grinding circuits. Too much of the already liberated particles of iron mineral in the ball mill discharge tend to report to the cyclone underflow and go back into the mill. The specific gravity of the iron mineral-rich underflow tends to direct to the overflow true middling particles which should be reground. The net result is that it is necessary to overgrind in order to achieve the desired liberation. The Larox classifier is employed to remove finish size material from cyclone underflows. Screening devices capable of screening down to about 270 to 325 mesh at acceptable feed rates have been developed; examples are the Rapifine Screen from Dorr-Oliver and the Derrick Multifeed Screen. These two types of screens have been used effectively to screen out middling particles for further grinding.

4.2.4. MECHANICAL CONSIDERATIONS IN LARGE MILLS

Many of the large autogenous and semiautogenous mills have gone through periods with less than expected operating availability. Betts (1979) referred to problems with the mill shells at Palabora which the manufacturer was going to correct. A more universal problem has been the gear trains, either gear reducers and/or ring gears. Since the ring gear is fixed to the mill shell, the ring gears are more than 36 ft in diameter on the 36 ft diameter mills at HibTac. McManus (1979) stated that ring gear problems were the major cause of unscheduled downtime at Lornex.

Obviously the gear manufacturers are working to improve gear reliability. An alternative to the gear drive train driven by, say, two - 6000 hp motors, is the wraparound motor. In Europe, gearless wraparound drives have been in common use in dry cement grinding, but the North American introduction was at St. Lawrence Cement (Fenton et al., 1977). An 8750 hp wraparound motor drives a cement finishing mill 17 ft in diameter by 56 ft long. The drive is cantilevered over the discharge end of the mill.

The earliest application of these drives has been in dry grinding. Mill manufacturers apparently are considering wraparound motors for wet, semiautogenous mills. The capital cost of the mill plus drive evidently is higher, but improved operating availability might be a substantial plus for the gearless units.

4.2.5 SCALE-UP AND MILL SELECTION

Scale-up necessarily includes selection of the grinding system. It follows that the comminution units in each system will have to be scaled up to provide comparisons of capital and operating costs for each system.

4.2.5.1 Rod and Ball Mills

Rowland (1972) and Rowland and Kjos (1978) have provided detailed sample calculations for sizing conventional rod or ball mills. They are based on laboratory determinations of the Bond Work Index, which gives an estimate of the energy required for rod mill or ball mill grinding. Efficiency factors are listed that relate to, for example, the proposed diameter of the mill, and these factors are used to correct the Bond Work Index. The grinding energy per ton as obtained from the corrected Bond Work Index is then used to size the mill for the required tonnage. An Abrasion Index derived from a separate laboratory test is used to predict liner wear and grinding media consumption.

Rowland (1972) published the results of an exhaustive comparison of operating work indices with laboratory Bond Work Indices. Good correlations were obtained, and some of the guidelines for rod and ball mill selection were modified by Rowland (1976). For conventional rod or ball mill grinding, sufficient confidence exists to size mills on the basis of the laboratory Bond Work Index. Thus, with drill core samples representing the proposed ore body, Bond Grindability Tests can be used to provide a range of possible work indices on which to base a judgment of the grinding energy required.

4.2.5.2 Mill and Circuit Design

Current design procedures are based largely on empiricism using correlations of mill performance with some kind of grindability test. The latter can be classified into two general types: standard grindability tests, and similitude tests. The standard tests, of which the Hardgrove Grindability Test (Hardgrove, 1932) is a well-known example, usually involve a simple grindng test in a standard laboratory machine. The results, generally in the form of a grindability index, are compared with standard correlations, based on accumulations of empirical data, to select the appropriately sized mill for the desired feed and product specifications and production rate. Similitude tests such as the Bond Test (Bond, 1952) are based on the scale-up of laboratory tests carried out in a standard mill of the same type as the production unit. Again, the scale-up criteria are determined, for the most part, from accumulations of empirical data.

More advanced design methods based on the population balance models have yet to find general acceptance, largely because of the lack of an adequate data base and incomplete knowledge of the appropriate scale-up laws. In general, current design procedures do not take machine dynamics into account directly. Mass transport considerations are essentially ignored.

One of the principal attractions of the population balance models of grinding processes is their direct applicability to computer simulation of the process (Herbst et al., 1973; Luckie and Austin, 1972; Lynch, 1977; Kelsall et al., 1968). By incorporating the appropriate mass transport and classifier models, complete grinding circuits can be simulated for analysis of process behavior, optimization, and automatic control.

4.2.5.3 Autogenous and Semiautogenous Mills

In autogenous or semiautogenous grinding, the general consensus, summarized by Shoemaker and Gould (1978), is that pilot plant testing is required to provide information for scale-up. Rowland (1976) stated that work indices obtained from standard Bond Grindability Tests cannot be used to determine the grinding energy for autogenous milling. Pina (1979) has described the procedure used by Koppers-Hardinge based on pilot plant testing. Attention was called to the need to measure accurately the power draft of the test mill, including the use of a Prony brake to provide information on losses in power transmission to the test mill. A formula was presented relating the power draft of the commercial mill to that of the test mill as follows:

$$\frac{\text{Power com}}{\text{Power test}} = \frac{\text{Length com}}{\text{Length test}} \times \left(\frac{\text{Diameter com}}{\text{Diameter test}}\right)^{2.84} \times \frac{\text{Critical speed com}}{\text{Critical speed test}}$$

The above formula is at variance with that usually employed in scaling up grinding mills with power varying as the Length X $(\text{Diameter})^{2.5}$.

In determining the suitability of an ore for autogenous grinding, a media competency test is run on as many crude ore types from the ore body as are available. The media competency test assesses the ability of the coarse pieces of ore in the feed to survive as grinding media. Without suitable competency as grinding media, an ore charge tends to build up within the mill, reducing feed rates and increasing power consumption.

Pilot plant testing for autogenous or semiautogenous mills may present problems with ore availability and the availability of the possible different ore types. In some cases, it has been necessary to sink a shallow shaft and drive an adit into the ore body. Careful test work should be done on a sufficient number of typical crude ore types and blends thereof to minimize risk in scale-up. Furthermore, it is essential that the data obtained from careful test work be used with sound engineering design judgment in choosing a system. The nature of the ore as determined in testing may rule out or favor one system or another.

On the basis of both media competency tests and pilot plant autogenous grinding tests, the decision on whether autogenous grinding is feasible can be made. Crushing of the excess pebbles, referred to earlier, made it possible to use autogenous grinding at Tilden. If the ore is not suitable for autogenous grinding, pilot plant tests with a ball charge can be used to determine grinding energy requirements in the semiautogenous case. Semiautogenous grinding will have more universal application because the large balls assure breakage of the harder ore pieces to prevent build-up of the ore charge in the mill. Semiautogenous grinding offers the possibility of conversion of older mineral processing plants so that the crude ore can be taken directly from a primary crusher to the semiautogenous mill to avoid the construction of a new fine crusher plants.

The choice of the grinding system is based on the estimates of capital and operating costs. Freitag (1979) has presented comparisons of autogenous versus conventional grinding for three separate operations--an expansion and two new operations--processing, respectively, 5000, 3300, and 1200 short tons per day. In each case, both capital and operating costs were estimated to be lower for the autogenous circuits. Estimates by Barratt (1979) of direct capital and operating costs favored semiautogenous over conventional grinding for porphyry copper operations. The advantage becomes significant with tonnages greater than 30,000 short tons per day.

Marcus Digre (1980) presented to the committee "An Energy Accounting in Comminution," comparing conventional, autogenous, and semiautogenous grinding. Energy consumed in operating crushing and grinding units and energy associated with wear materials were totaled, giving an energy equivalent of material ground from a feed size (F_{80} = 150 mm) to a product size (P_{80} = 0.06 mm). Two hypothetical cases--a hard ore and a soft ore--were considered. Metal wear comprised of media and of crusher and tumbling mill liners was based on the Allis-Chalmers abrasion index (A.I.) using formulas listed in

Rowland and Kjos (1978). Digre used an energy requirement for ordinary steel of 0.009 kWh/gm (4.1 kWh/lb) and double that for manganese steel. Digre's data have been summarized in the following table showing the ratio of the operating work index (OWI) to the Bond Work Index (BWI) for autogenous and semiautogenous grinding at the total energy equivalent to that for conventional grinding. For conventional grinding the OWI was assumed to be the same as the BWI. For values of the OWI/BWI less than those indicated in the tabulation, the total energy consumed in autogenous or semiautogenous grinding is less than for conventional grinding.

	Ratio of OWI/BWI	
	Autogenous	Semiautogenous
Hard Ore: BWI = 15 kWh/ton, Allis-Chalmers A.I. = 0.5, 46.6 kWh/ton total energy in conventional grinding.	1.9	1.7
Soft Ore: BWI = 7.5 kWh/ton, Allis-Chalmers A.I. = 0.1, 18.1 kWh/ton total energy in conventional grinding.	1.5	1.4

At present, all design procedures are oriented toward selecting a grinding device to produce a given product size. In mineral processing, however, it is usually the extent of liberation of valuable minerals, not size, which is important in a grinding operation. Although there is generally a reasonable correlation between product size and liberation, the development of design procedures that involve liberation directly is a worthwhile goal. To implement such procedures, additional work is needed in the area of quantifying liberation (Sections 3.2 and 3.3).

In the two cases cited earlier (Aitik and Palabora), the ratios of autogenous to conventional grinding energy (excludes wear materials) were 1.25 and 1.2 respectively.

Another aspect of scale-up in autogenous grinding is the wide disparity between North American and Scandinavian practices with regard to length-to-width ratios. The 36 ft x 15 ft mills at HibTac have an L/D ratio of 0.4, whereas the 20 ft x 34.5 ft mills at Aitik have an L/D ratio of about 1.7. Russell (1967) described the reasons for the large diameter-short length tumbling mills. Digre (1977) discussed the advantages of both the long and the short mills and pointed to the fact that the question could be best answered by having one mill of each L/D design in the same plant rated at the same power draw. Capping the discussion, Digre claimed more stable operation for the longer mill.

4.2.6 RECOMMENDATIONS & RESEARCH INCENTIVES

The incentives for research in this area of comminution are as follows:

- A reduction of even 10% in the energy used in comminution in mineral processing plants would conserve about 3 billion kWh per year. Higher energy costs should spur energy conservation.

- As comminution becomes more energy-efficient, fewer grinding units should be required, reducing capital as well as operating costs.

- The ability to scale up accurately from bench tests for all grinding systems would reduce developmental costs for a given flowsheet and provide opportunities to minimize capital costs. Where pilot plant testing is still required to test the complete flowsheet, the operating work indices obtained should be sufficiently accurate for scale-up.

Based on the above discussion and incentives, the following recommendations are made:

- Given the apparent drop in grinding efficiency in the largest diameter ball mills, fundamental research related to motion of the ball in the tumbling charge is recommended.

- Relative to the limitations in the sizes of both rod mills and ball mills, there is a requirement for fundamental research related to material transport through the tumbling charge of ore and grinding media.

- The Bond Work Index is based on determining the 80% passing size from a curve of size distribution. Now that computers are quite universally used, a computed index based on the total size distribution might provide improved scale-up procedures.

- In autogenous and semiautogenous grinding, research is needed to establish the diameter, length, and energy consumption relationships necessary for the development of the most energy-efficient configuration.

- The reported success with the square spiral lining indicates that liner configurations present potentially fruitful areas for research.

- An assessment of the limits to the size of a fully autogenous or semiautogenous mill should be made. Also, investigations are needed to evaluate whether the installation of a primary crusher could be avoided with the addition of gearless drives so that it would it be possible to feed carefully blasted pit run material over a grizzly and into such a mill.

- Rubber has been used successfully as a wear material in grinding mills. Other polymeric wear materials could conceivably be developed. Chemical properties of these polymeric materials could be scrutinized, recognizing the demands of the environment in the grinding mill.

- Continued development of improved mineral separation techniques could reduce grinding energy by removal of waste or liberation of valuable minerals from the circuit as soon as possible, using the concept of stage grind-stage separation.

4.3 Classification

Although classification is not a unit operation in which size reduction occurs, classifiers are an integral part of closed circuit grinding systems, and the ability to specify and operate classification devices is vital for the achievement of good overall grinding efficiency.

This section of the report examines the role of classifiers in closed circuit grinding, discusses the various types of classification devices used in dry and wet grinding circuits, and identifies areas for further research.

4.3.1 CLOSED CIRCUIT GRINDING

Classifiers are devices that can be used to create lower-energy-consuming fine grinding systems. The classifier and the grinding device are integrated so that the output from the grinding device becomes the input to the classifier. The classifier, in turn, separates the input particle stream into a stream of fine particles and a stream of coarse particles. The coarse stream is returned to the grinding device, and the fine stream is discharged as circuit product. This integrated arrangement, termed closed circuit grinding, allows the mill to operate in a lower-energy-consuming coarse grinding mode while producing a fine product.

Because the size distribution of the fine particles is normally characterized by a single control size--e.g., 80% less than 200 mesh (75 microns)--closed circuit grinding leads to two very important comminution principles:

- o The quantity of product is increased only when a steeper size distribution (one with less fine material but the same control size) is produced.

- o A size distribution with a minimum proportion of fines (a steep size distribution) is produced by achieving a high ratio of coarse material mass to fine material mass with a very selective classifier.

The obvious advantage of the first principle is that the grinding specific energy

$$E = \frac{\text{power draw of the mill}}{\text{product production rate}} \qquad 4.3\text{-}1$$

should be reduced, since the power draw of the grinding device should remain essentially constant while the power draw of the classifer and of incremental transportation should be very small (say, less than 10% of the power draw of the mill). As a rule of thumb, the output is increased 30% or more. Therefore, the closed circuit specific energy should be 85% or less of the open circuit specific energy.

The two foregoing comminution principles would indicate that there is no limit to increasing the capacity and eliminating the fine sizes in the circuit product, perhaps even producing monosized material at very low specific energies. However, there are constraints on these principles, both practical and theoretical.

An example of a practical constraint would be the maximum feed rate to the grinding device, since, as the ratio of coarse material to fine material (C) increases, the feed rate (F) to the grinding device increases much faster than the production rate (Q) of the circuit; i.e.,

$$F = (1 + C)\, Q \qquad 4.3\text{-}2$$

Thus, increasing the production ratio from 100 to 130 tons per hour by closing the grinding circuit so that the circulation ratio (C) goes from zero to two means that the feed rate to the grinding device will increase from 100 to 390 tons per hour.

There is also a very important theoretical constraint--a limiting condition that is approached since, when particles of a certain size fracture, they produce a suite of particles smaller than the original particles. Therefore, there is a limiting size distribution and consequently a limiting production rate. Nevertheless, reductions in the specific energy on the order of 50% should be achievable. However, achieving such a reduction is obviously a function of the comminutor and the classifier.

4.3.2 CLASSIFICATION PERFORMANCE

The second comminution principle pinpoints the need for very selective classification in order to reduce the grinding specific energy. Classification is the term assigned to nonscreening processes that partition particles in a fluid medium--usually air or water--into two streams on the basis of their size. One stream contains coarse particles; the other contains fine particles. Now, it is assumed that size is the only characteristic that varies among the particles and hence only size influences their trajectories. Obviously, other characteristics, such as specific gravity or shape, affect the trajectory of the particle, but only in a minor manner, if they are essentially the same for all particles in the feed. If there is a characteristic, such as specific gravity, among the particles that varies and influences the trajectory, the partitioning is termed sorting, not classifying. Thus, if the feed is a mixture of particles with different specific gravities, each group of particles must be treated individually when analyzing or predicting classifier performance.

The perfect classifer would send all particles in the classifier feed larger than the designated "cut size" back to the grinding device, while all particles smaller than the designated cut size would be removed as circuit product. The fine recovery value--the ratio of the quantity of material smaller than a particular size in the fine stream to the quantity of material smaller than the same size in the feed stream--would be 100% for all sizes less than the cut size. Such a classifier is termed ideal.

Unfortunately, real classifiers suffer from two types of nonideal behavior. One type occurs because the trajectory taken by a particle of particular size in a real classifier varies, hence the exiting stream it enters also varies. The probability that a particle smaller that the cut size will report to the coarse stream is not zero. Instead, the probability increases monotonically from 0 for the smaller particles to 1 for the larger particles, resulting in misplaced material and loss of the identity of the cut size. In addition, the fine recovery values will change for each particle size. The probability values change for most classifiers as a function of the feed material's p-operties and the operating conditions. Some types of classifiers are more sensitive to these factors than others.

Since the identity of the cut size is lost with real classifiers, a substitute is the d_{50} value, or equiprobable size--the size of the particles whose probability of entering either stream is 0.5. A measure of misplacement or dispersion is the Sharpness Index (κ), which is the ratio of size of the particles whose probability of entering the coarse stream is 0.25 to the size of the particles whose probablity of entering the coarse stream is 0.75. An ideal classifier would have a κ value of 1; real classifiers have κ values less than 1. Good industrial classifiers should have κ values in the range of 0.6 to 0.8.

The other type of nonideal behavior is the Apparent Bypass. If, because of mutual interference or other reasons, some of the particles of each size in the classifier feed stream end up in the coarse stream, a certain portion of the coarse stream will have the same size distribution as the classifier feed stream. Therefore, it appears that a certain percentage of the classifier feed stream bypasses the classifier and reports directly to the coarse stream (a splitting action). This type of nonideal behavior is particularly common to wet classifiers, where the bypassed particles are termed "void filling" material, but is also observed for dry classifiers.

These three parameters--the Sharpness Index (κ), the Cut Size (d_{50}), and the Apparent Bypass (a)--determine the classifier's performance (AIChE, 1980). The performance, along with the classifier feed size distribution, dictate the size distribution of the coarse and fine streams. The effects of the classifier performance parameters are shown in Table 4-1. Here a product defined by a single control point (95% less than 150 microns) is being made from a fixed feed (50% less than 150 microns). The first result is for an ideal classifier ($\kappa = 1$). The second is for partially ideal classifier--i.e., an ideal classifier with 30% Apparent By-pass ($a = 0.3$). The fine size distribution is the same; however, the recovery value for 150 microns (R_{150}) decreases, and the ratio of the coarse stream mass to the fine stream mass (C) increases. The third set is for an industrial classifier ($\kappa = 0.6$). The Cut Size (d_{50}) must be lowered to achieve the desired control value of 95% less than 150 microns. This trend is typical of classifier behavior. The lower the κ value, the smaller the d_{50} value required to produce the desired single point control value. The last set represents a typical industrial classifier ($a = 0.3$). Again, the fine stream size distribution is not affected by the Apparent Bypass, only the coarse stream size distribution. However, the recovery value is a minimum when both types of nonideal behavior ($\kappa < 1$, $a > 0$) are present.

As noted in the previous example, the Apparent Bypass does not affect the fine stream size distribution. Analysis of various types of industrial classifiers has led to the observation that the Sharpness Index is essentially constant for a classifier with a fixed geometrical configuration over its normal operating range. Therefore, only two things affect the fine stream size distribution--the classifier feed size distribution and the Cut Size. Hence, if the size distribution of the feed to the classifier is constant, only one d_{50} value will produce a fine size distribution containing the desired control value.

4.3.3 CLASSIFIERS

Usually in classification, the fine stream is not defined in terms of 100% passing a particular size, since to guarantee such a specification would require a fine stream much finer than actually needed. Hence, the fine stream requirement is usually specified as 95% passing, 80% passing, etc. Using the 95% passing definition, the type

TABLE 4-1: Comparison of ideal and nonideal classification. Size distribution data. Weight fraction passing stated size.

K		1.0		1.0		0.6		0.6	
d_{50}		160.0		160.0		122.5		122.5	
a		0		0.3		0		0.3	
size	**feed**	**fine**	**coarse**	**fine**	**coarse**	**fine**	**coarse**	**fine**	**coarse**
1180	99.9	100.0	99.8	100.0	99.8	100.0	99.8	100.0	99.85
850	99.75	100.0	99.5	100.0	99.6	100.0	99.6	100.0	99.65
600	98.5	100.0	96.8	100.0	97.6	100.0	97.6	100.0	98.0
425	94.0	100.0	87.3	100.0	90.5	100.0	90.6	100.0	92.0
300	82.5	100.0	63.0	100.0	72.0	99.8	72.6	99.8	76.6
212	66.5	100.0	29.8	100.0	47.1	98.9	48.0	98.9	55.5
150	50.0	95.0	0.0	95.0	23.5	95.0	24.3	95.0	34.75
106	35.0	66.6	0.0	66.5	16.5	82.8	7.7	82.8	18.8
75	24.0	45.5	0.0	45.5	11.3	63.2	1.0	63.2	10.7
53	16.0	30.5	0.0	30.5	7.5	43.5	0.3	43.5	6.7
38	10.5	20.0	0.0	20.0	4.9	28.9	0.0	28.9	4.25
C		0.9		1.7		1.75		2.95	
R_{150}		1.0		0.7		0.7		0.5	

of classification can be defined as presented in Figure 4-2 from Hukki (1973). Industrial classifiers are in the coarse to medium classification range.

The particles are partitioned by sequentially or simultaneously combining gravitational (including centrifugal), inertial, or drag forces. Within the classifier, the fluid is always moving. The fine particles exit with the fluid. Consequently, a solid/fluid separation must follow serially if the particle stream is to be utilized separately. Classifier designers take advantage of one of a number of facts: small particles fall at a slower rate than large particles; larger particles are acted upon with greater force in a cyclonic flow than smaller particles; smaller particles can change their direction of flow much more easily than larger particles; larger particles require a higher conveying velocity than smaller particles; or the probability of collision with a rotating blade is higher for a larger particle than for a smaller particle. The designer then proceeds to design the classifier so that there is a minimum mutual interface among the particles in the classification zone.

The performance of the classifier is a function of the feed material, the operating conditions, and the classifier. Variation in any of these can give results that range from excellent to poor.

Feed material properties that influence performance include, but are not limited to, such things as:

1. Size consistence

    a. cumulative percent passing the designated size of the product stream control point

    b. the narrowness of the size range (the so-called near product material)

2. Specific gravity

3. Rheological properties

Classifier design factors include, but are not limited to, such things as:

1. Type of classification
2. Geometric configuration
3. The setting of adjustable features
4. Materials of construction

Operating conditions include, but are not limited to, such things as:

1. Fluid/solid ratio
2. Energy input
3. Feed rate

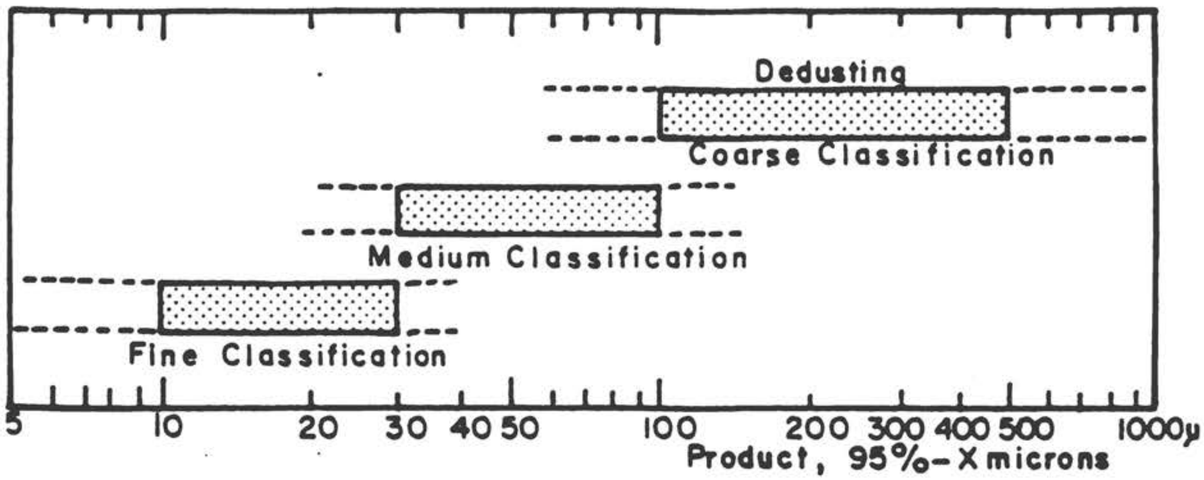


FIGURE 4-2 Classification range. (Hukki, 1973)

Obviously, there are correlations between these factors. For example, the rheological properties of a slurry depend not only on the material, but also on the fluid/solid ratio. Not all of these factors are controllable; in any given system, many are uncontrollable.

There are many different kinds of classifiers, so it is convenient to divide them into common groups. First, classifying devices can be split into two groups on the basis of the conveying fluid--gas or liquid, or dry or wet classification. Each of these groups can be subdivided on the basis of whether the solids and fluid enter the classifier independently (internal mixing) or together (external mixing). Finally classifers can be grouped according to the primary separating mechanism, e.g., centrifugal or intertial. For this paper, only a dry and wet grouping will be employed.

4.3.3.1 Dry Classifiers

The most common type of pneumatic classifier is a modification of the Mumford-Moodie device invented about 100 years ago. The mechanical air separator embodies all of the system aspects of dry classification.

Basically, a mechanical air separator is constructed with an inner shell and an outer shell (Figure 4-3). Material fed by gravity to a rotating plate is dispersed within the inner shell. Air, pulled in from the outer shell, passes upward through the descending curtain of dispersed feed, elutriating intermediate and fine material out of the feed. The elutriated particles enter a section that contains rotating blades, which separate the intermediate from the fine particles, returning the intermediate material to the coarse stream. The fine particles exit from the inner chamber into the outer shell in the air stream, passing through the turbine or the blower, which maintains the forced circulation. The swirling of particles and air in the outer shell is supposed to separate the fluid and the particles, and the air reenters the inner shell. Both the coarse and the fine particles exit via seals. The classifier scales up volumentrically, and units 10 m in diameter and 30 m high have been manufactured. The fineness of the product is dictated by the speed, pitch, and number of secondary fans.

The dry classification system aspects, then, are:

- Classifier
- Fan
- Cyclone
- Seals

The mechanical air separator integrates these elements into one compact package, while other types of dry classifiers must be engineered into such a system. The mechanical air separator also employs gas recirculation, rather than a once-through design.

However, the outer shell of a conventional mechanical air separator generally does not separate the solids from the gas very efficiently. Thus the recirculated gas causes less efficient classification to

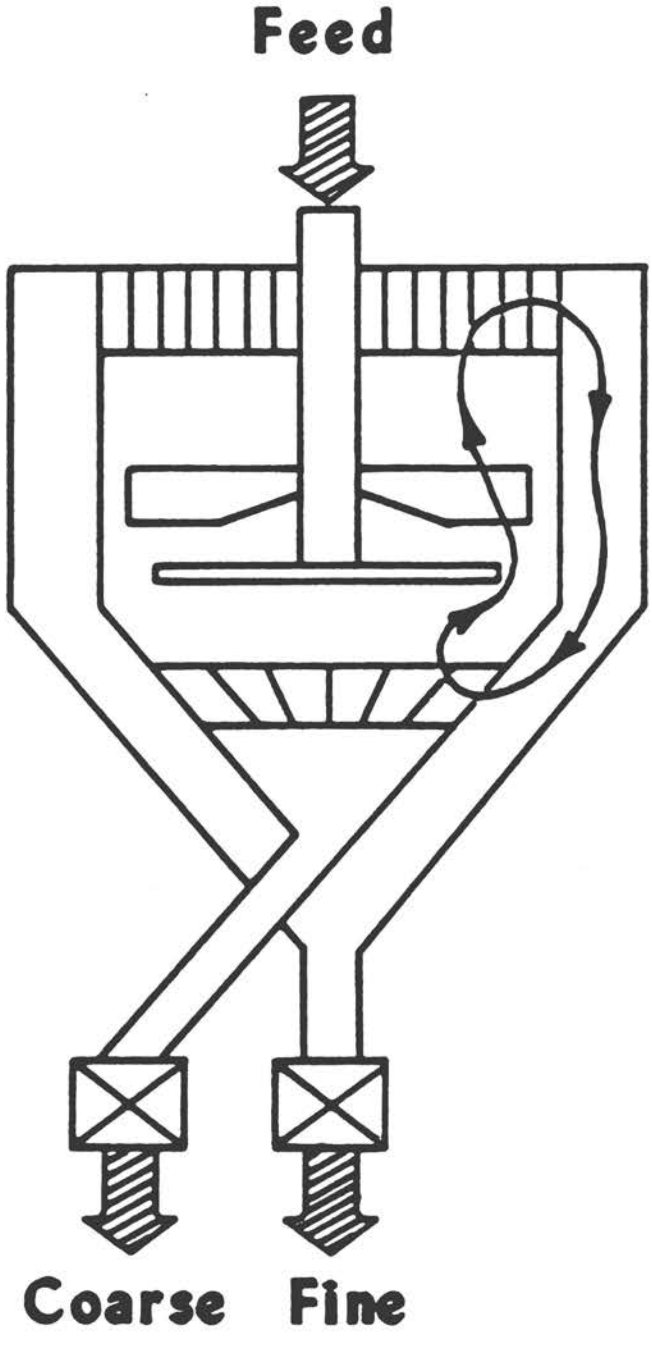


FIGURE 4-3 Mechanical Air Separator.

occur. Consequently, new designs with clusters of more efficient cyclones and an externally mounted fan are being offered. Usually the cyclone is not sufficient to remove all the solid, and a dust collector must be used. However, by recirculating the gas stream only a fraction of the total gas stream needs to exit via the dust collector.

The performance of this type of classifier has been studied by Austin and Luckie (1976) and Luckie and Austin (1975). The secondary fan in the 1.2 diameter test unit could be operated at speeds independent of the primary fan speed. Consequently, variation in the speed of the secondary fan should be equivalent to blade removal or pitch changes in devices that do not incorporate independent secondary fan speeds.

Classifier performance was analyzed by adopting a special two-stage classification arrangement (Figure 4-4). The Sharpness Index for the second classifier--the solid/air separator--was found to be an essentially constant value of 0.75. The Apparent Bypass for the second classifier was also found to be an essentailly constant value of 0.7. In this case, a high value is desirable. The Sharpness Index for the first classifier was found to be an essentially constant value of 0.615. The variation of the Cut Size and Apparent Bypass for various primary and secondary fan speeds and feed rates is summarized in Table 4-2.

Increasing the primary fan speed decreased the Apparent Bypass, everything else being constant. This produced a doubling in the classifier power draw, but also a doubling in the mass product. Hence, the classification specific energy remained constant. Therefore, it would make sense to operate at the higher primary fan speed. At fixed primary and secondary fan speeds, increasing the feed rate increased the Apparent Bypass. The maximum feed rate for this unit should probably be around 5.5. Therefore, the feed rate of 1 represents a major underloading of the classifier. Consequently, it is appropriate to state that the Cut Size does not change with feed rate. Underloading the classifier to reduce the Apparent Bypass produced an increase in the Cut Size. The increase in the circulation ratio (C) with increasing feed rate but a constant Cut Size can be attributed solely to the increase in the Apparent Bypass. The specific energy for this type of classifier is estimated at 2.0 kWh/ton for Portland cement.

A new type of pneumatic classifier developed in Finland (Hukki and Airaksinen, 1980) is the centrifugal classifier shown in Figure 4-5. This classifier has no internal moving parts. The solids are fed by gravity into the classifier where they are dispersed by a high velocity primary gas stream. The coarse material continues to fall to the bottom of the classifier where a secondary gas stream recleans the fraction before it exits through a seal. The primary and secondary suspensions are directed upward into a centrifugal field. The fine particles exit in the gas stream via an eccentrically placed channel. The middling fraction separated in the centrifugal field is returned to the primary air stream.

The fineness of the product is controlled by the amount of gas, the ratio of primary to secondary gas, and the location of the exiting channel marker. This classifier scales up volumetrically. The

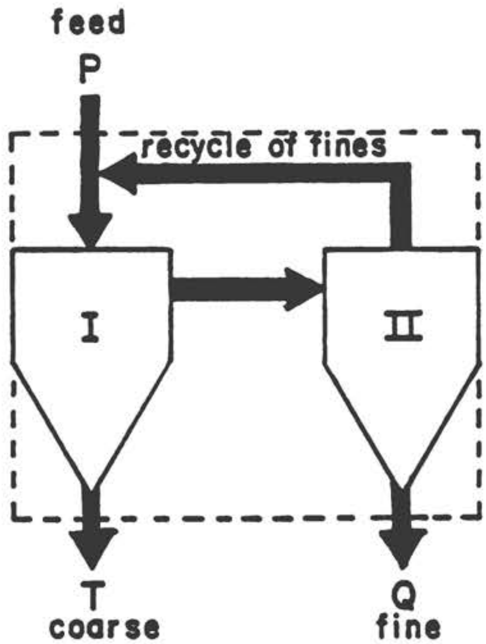


FIGURE 4-4 Two-stage classification arrangement.

TABLE 4-2: Salient classification parameters for a mechanical air separator at various operating conditions

fan speed, rpm primary	fan speed, rpm secondary	relative feed rate	d_{50}	a	c
1400	600	1	62.5	0.05	0.75
		4	40	0.50	4.1
		7	40	0.75	8.35
	1000	1	32.5	0.10	2.0
		4	30	0.75	10.3
		7	30	0.80	13.6
1850	800	1	60	0.05	0.8
		4	40	0.30	2.4
		7	40	0.40	3.3
	1000	1	45	0.05	1.15
		4	35	0.30	2.95
		7	35	0.50	4.4
	1400	1	35	0.05	1.5
		4	30	0.40	4.15
		7	30	0.60	6.75

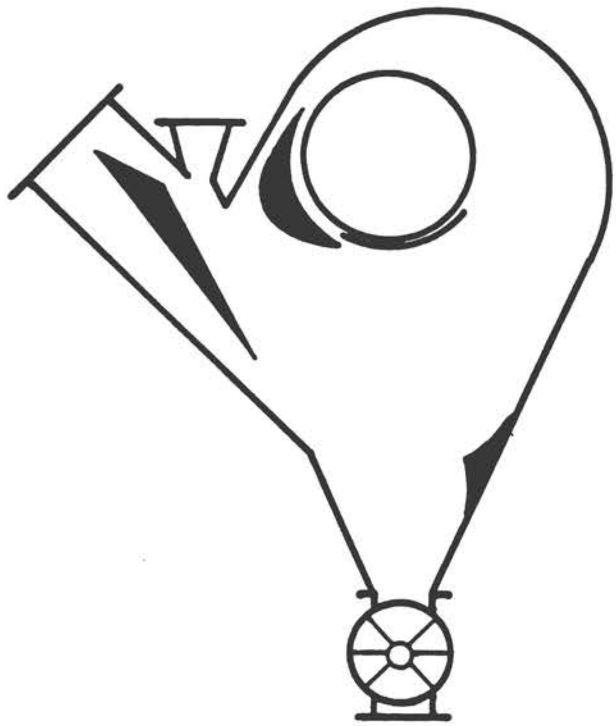


FIGURE 4-5 Centrifugal Pneumatic Classifier.

specific energy for this classifier is estimated at 2.0 kWh/ton for Portland cement.

Other dry industrial classifiers used in closed circuit grinding receive their feed in a gas conveyed stream. This is achieved either by air sweeping the grinding device or by air lifting the discharge of the grinding device. Simultaneous drying of the solids prior to classification to reduce agglomeration is also possible.

Two dry classifiers shown in Figure 4-6 are similar except that the one on the left scales up by utilizing clusters of units (multiunit scaling), while the one on the right scales up volumetrically.

In the classifier on the left in Figure 4-6 the solids enter the main expansion chamber in the gas stream. The air exits through adjustable vanes, carrying the surviving fine particles. The middling and coarse particles continue downward where they are recleaned by a seondary gas stream entering from the left. The coarse particles continue to fall to the bottom and exit through a seal, while the middlings are swept upward in the secondary gas stream into the main expansion chamber. Adjusting the vanes and the secondary gas controls the fineness of the product.

In the classifier on the right in Figure 4-6 the solids are brought to the classifier in a gas stream flowing through a volute. The centrifugal action of the volute concentrates the solids to the outside as they leave the elbow. The gas is directed via adjustable gates down the right side where it recontacts the solids at several points, causing the particles to be swept outward and upward into the main expansion chamber. Particles that continue to fall or drop out in the expansion chamber exit through a seal at the bottom of the classifier.

The main expansion chamber is adjustable, so that product fineness can be varied. In addition, part of the incoming gas can be diverted by adjusting the port to the left of the feed entrance.

Figure 4-7 shows the evolution of the internal configuration of a basic type of dry classifier--the cone classifier--in which the following basic geometric configuration is maintained:

- o Solids and gas enter through a center feed pipe.
- o The larger particles exit through the bottom of the chamber.
- o Fine particles and gas exit through a central product pipe in the top of the chamber.

The evolutionary process involved incorporating additional geometrical obstacles to improve the selectivity of middlings removal.

Sketch A of Figure 4-7 represents the classical expansion classifier. Here, solids are brought into the classifier in the airstream flowing up through the center feed pipe. As the gas/solid stream exits from the pipe into the classifying chamber, the sudden expansion reduces the velocity of the stream to the point where it cannot convey the larger particles, which drop to the bottom of the chamber and exit as coarse rejects through a seal. As the gas stream continues up the chamber, the volume continues to expand, reducing the stream's velocity

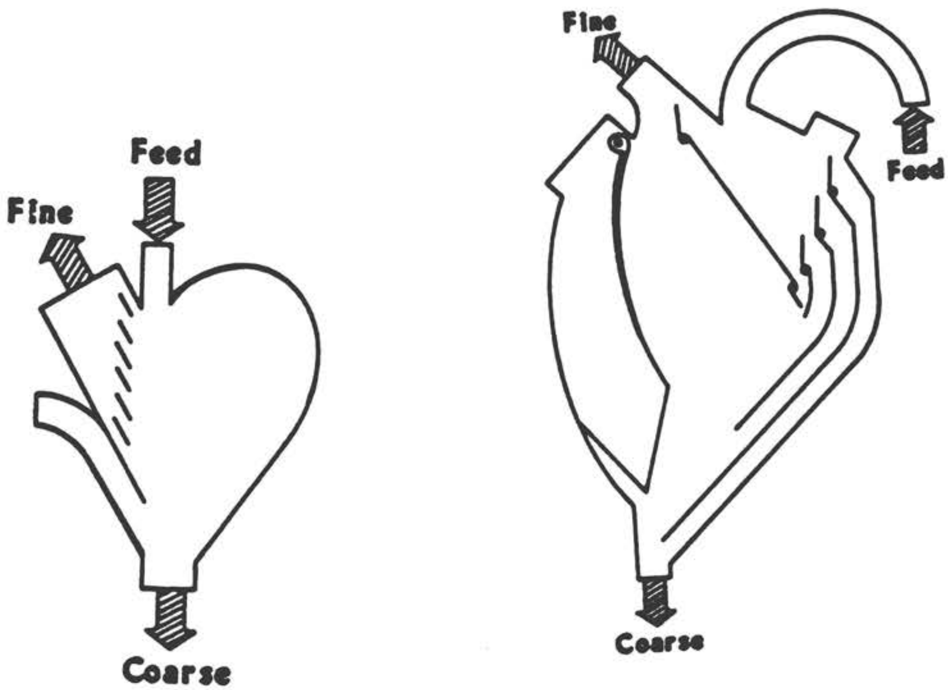
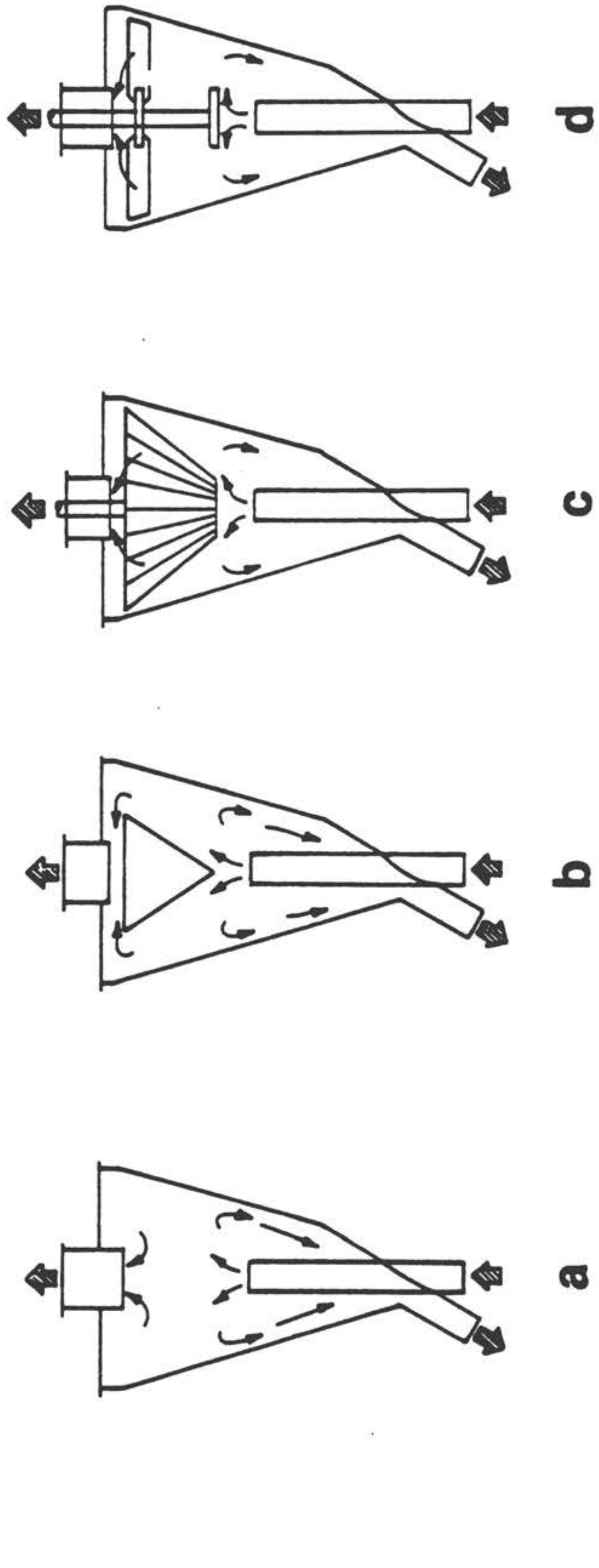


FIGURE 4-6 Dry Classifiers with Gas Conveyed Feed.



a b c d

FIGURE 4-7 Evolution of Dry Classifier Design.

even more. Middling particles escape the airstream, falling to the wall and then to the bottom of the chamber. The fine particles exit from the classifier, since the velocity of the conveying stream is dramatically increased in the product pipe.

In Sketch B of Figure 4-7, a solid cone is used to force the incoming gas/solid stream into a radial flow, which increases the likelihood of particle/wall collision. Sometimes the product pipe is extended down into the chamber, increasing the tortuous path. The cone can be slotted, requiring the gas/solid stream to pass through it to reach the exit. The angle of the slots can be set to vary the degree of difficulty for a particular size particle to pass through without collision or loss of conveyance. The slotted screen on such an inertial type classifier can be rotated as depicted in Sketch C, thereby subjecting the particles to a centrifugal force. The speed of rotation controls the product size. The use of a rotating geometric obstacle leads to the adaptation of the mechanical air classifier in which the dispersed gas/particle stream passes through a rotating, pitched-blade assembly. The blades force the gas to move radially and, of course, collide with the larger particles, causing loss of momentum. Such an adaptation is shown in Sketch D.

The twin cone classifier shown in Figure 4-8 is probably the most popular type of air classifier design for air-swept coal grinding. The feed stream enters the outer cone and passes through adjustable vanes into the inner cone. When the vanes are fully opened, the inner cone acts as an expansion chamber, increasing the amount of product while decreasing the overall fineness of the product. Turning the vanes toward the closed position causes the inner cone to act as a cyclone. The centrifugal action decreases the product output while increasing the overall fineness of the product. The coarse particles are directed to the bottom of the classifier and exit through a seal.

Sketch A of Figure 4-8 shows a twin cone classifier design using the basic geometric design presented earlier. A double solid cone arrangement is used to direct the feed stream to the outer cone and the coarse particles from the inner cone to the bottom of the classifier. Inefficient classification can occur in the double cone area. This difficulty is eliminated in design B, where the feed stream enters the outer cone directly while the coarse particles exit through a seal at the bottom of the inner cone. A slight modification of this arrangement is shown in design C. This arrangement is particularly adaptable as an integral classifier design by eliminating the need for a seal on the coarse stream.

The performance of this type of classifier was evaluated as part of an overall coal grinding study conducted by the Kennedy Van Saun Corporation for the Department of Energy (Luckie and Austin, 1979). The classifier performance parameters for the 0.7 m diameter test unit are shown in Table 4-3. Although the concentration of solids in the airstream could be changed for a fixed air rate, there was no noticeable change in performance parameters over the loading range investigated. However, the performance parameters did change with operating condition. The Sharpness Index changed only with vane setting. The Index increased with decreasing vane setting and then

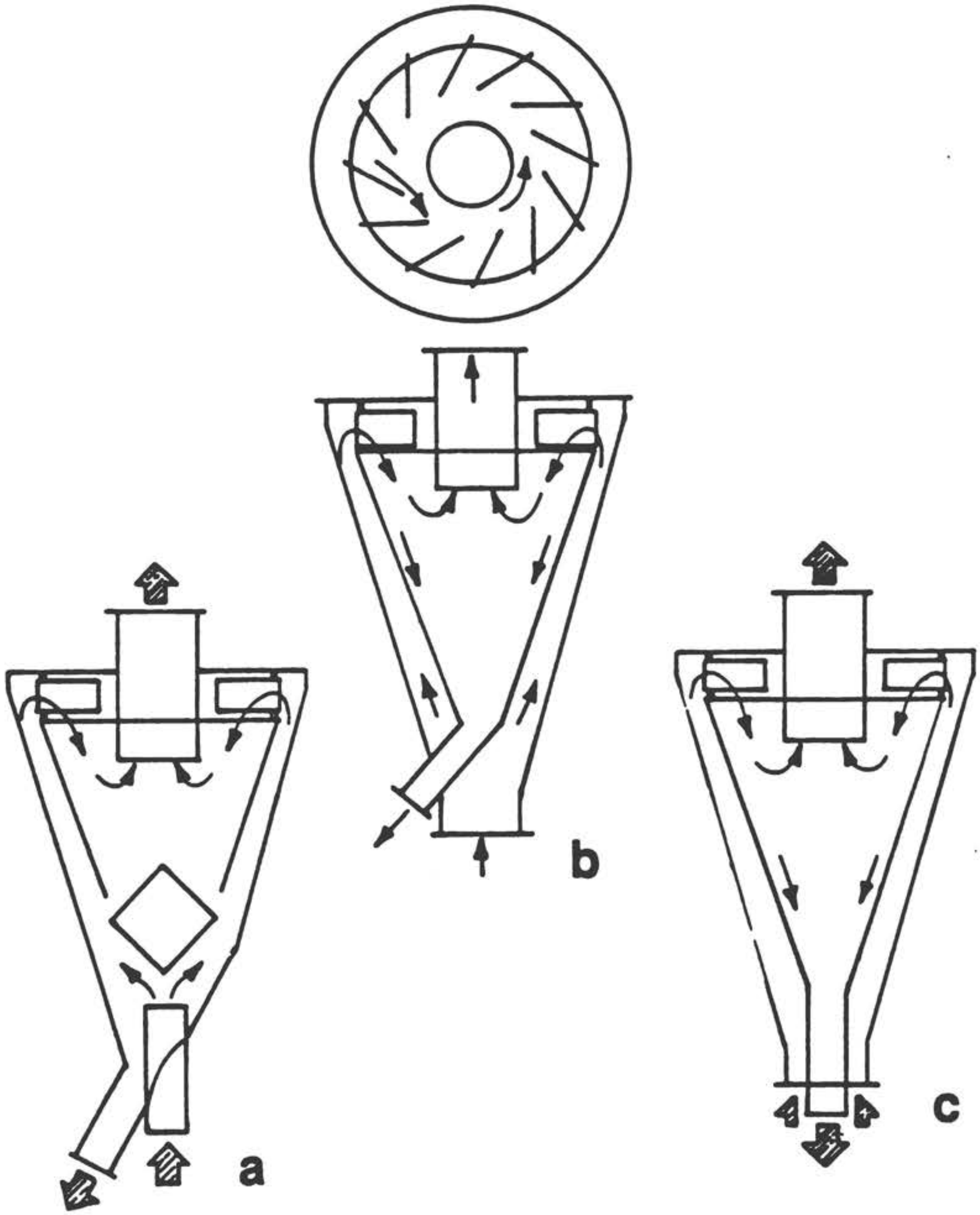


FIGURE 4-8 Twin Cone Classifier Designs.

TABLE 4-3: Salient classifier parameters for a twin cone pneumatic classifier at various operating conditions

vane setting -	100%	50%	25%
Sharpness Index (κ)	0.325	0.51	0.29
Cut Size (d_{50})			
1000 ACFM	120	56	22
2000 ACFM	164	105	82
3000 ACFM	197	151	178
Apparent By-pass (a)			
1000 ACFM	0.05	0.35	0.55
2000 ACFM	0.05	0.20	0.30
3000 ACFM	0.05	0.02	0.05

decreased, indicating that there is probably an optimal vane setting that maximizes κ. For all three vane settings studied, the Cut Size decreased with decreasing air rate. The range and rate of decrease was different for each vane setting. For air rates below 2500 actual cubic feet per minute (acfm), the Cut Size decreased with decreasing vane setting, although the ranges varied. At the 100% vane setting, the Apparent Bypass did not appear to change with air rate. At the 50% and 25% vane settings, the Apparent Bypass increased with decreasing air rate, being more pronounced at the 25% vane setting.

A new classifier design employed on an air swept coal grinding system in Finland is shown in Figure 4-9. It is an adaptation of the gravity fed centrifugal classifier discussed earlier. The feed stream enters through the volute, concentrating the solids to the outside as they leave the elbow. The stream is split as it enters the classifier into an upper particle stream and a lower gas stream. The particle stream is forced downward as it enters the classifier. As the velocity decreases, the coarse particles fall to the bottom of the classifier. The gas stream passes through the coarse stream before it exits via a seal to remove any fine particles. The particle/gas stream is then directed upward into a centrifugal field. The middling fraction is separated to the outer wall and returned to the incoming particle stream. The fine stream is removed via the eccentrically placed exit pipe.

4.3.3.1.1. Dry, Fine Classification

A classifier used in the production of fine powders should be capable of separations which are:

- Predictable and repeatable
- Adjustable
- Sharp
- Accomplished at high solids loading

Solids loading is the ratio of powder mass flow rate to classifying fluid mass flow rate and is a basis for comparing the relative capacities of units having different geometries. To achieve the desired performance, it is necessary to design the classifier system so that:

- All particles enter the classification zone under identical conditions

- The classification zone is well defined with stable flow conditions

- Effective particle dispersion is achieved.

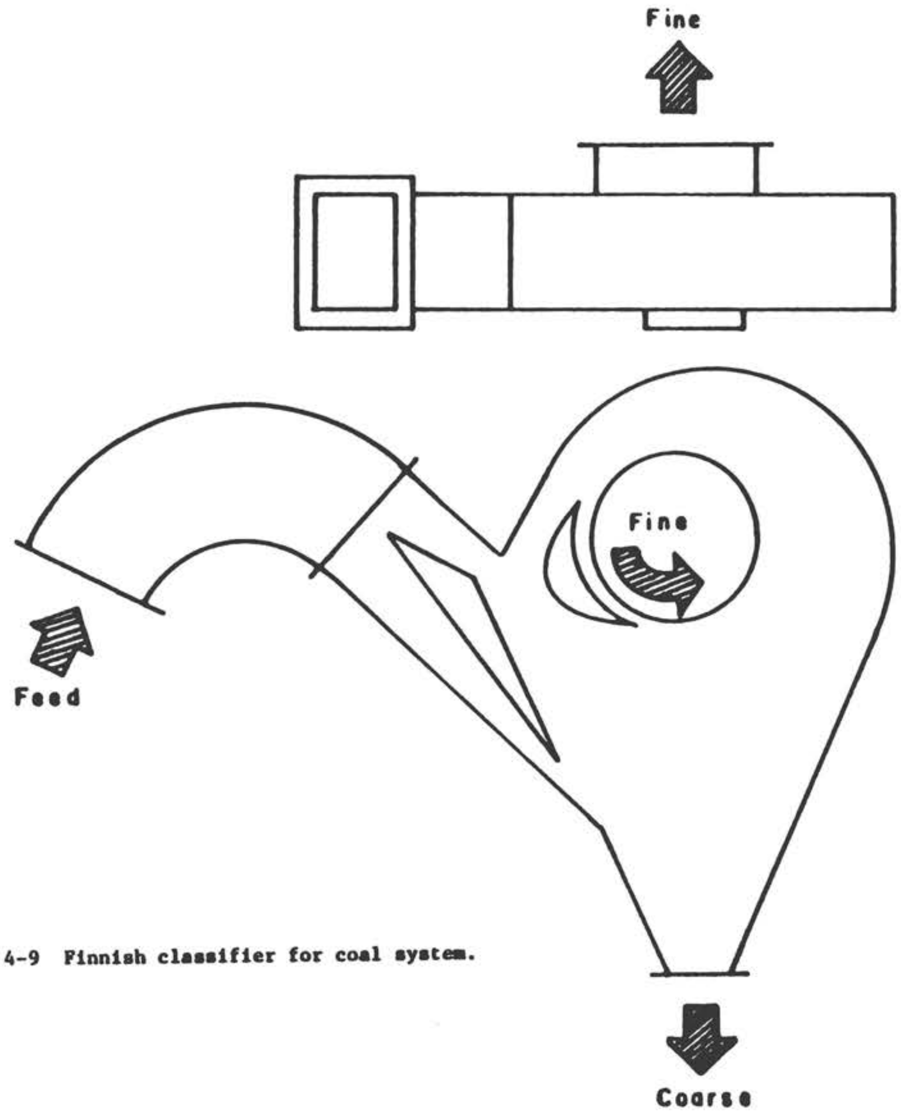


FIGURE 4-9 Finnish classifier for coal system.

In fine classification, dispersion is especially important to performance. Agglomerates behave as large particles in the classification zone and, if not dispersed, are carried to the coarse stream. Since fine particles have a relatively high surface to mass ratio, their behavior is dominated by surface forces. As a result, particle dispersion becomes increasingly difficult as particle size decreases. Typically, the classifier capacity is reduced, and the energy required to disperse the particles increases as finer powders are produced.

At this point it is perhaps appropriate to distinguish between fine classification and dedusting. Dedusting is the removal of dust-producing fine particles below a stated size from the bulk of the feed material, and usually represents removal of a minor portion of the feed stream.

Most fine classifiers operate under negative pressure. The feed material enters through some type of pressure lock. The coarse material usually is discharged through a pressure lock, normally at the bottom of the classifier. The fine material is air conveyed to a cyclone/dust collector system for removal from the air. The air can be exhausted or recycled.

It has been suggested that classifiers designed for fine classification can be characterized as either counterflow, transverse flow, or a combination of the two (Leschonski, 1977). The distinguishing difference between counterflow and transverse flow is that the coarse particles flow either essentially opposite or transversely to the main classifying fluid stream, respectively. The counterflow classifier depends on a force balance between particle drag and field forces (usually centrifugal for fine classification). The transverse flow classifier is an inertial device in which particle trajectories depend on the relationship between particle drag and inertia.

Counterflow classifiers are grouped as forced and free vortex according to the flow field generated within the classification zone. A free vortex classifier typically uses stationary guide vanes, while forced vortex types use rotating vanes. Free vortex classifiers do not always scale up and can be more sensitive to solids loading than forced vortex classifiers.

Particles introduced into the vortex-sink flow field are acted upon by centrifugal force and an inwardly directed drag force. The fine particles, having relatively large drag per unit mass, penetrate the centrifugal field and are removed from the classification zone through a central outlet. An example of a free vortex classifier is the Apline Mikroplex Spiral Type MV Classifier. An example of a forced vortex classifier is the Donaldson Acucut Classifier.

With a transverse flow classifier, feed particles are introduced transversely by a rotating plate into an annular jet of air. Upon entering the classification zone, the particles are fanned out into trajectories according to their size. Large particles tend to traverse the classification zone, with small particles following the air flow. Coarse and fine particles are separated by strategically located partitions. An example of a transverse flow classifier is the

Bauer Centi-Sonic Classifier. Another is the Jet Classifier, the commercial version of the cross flow transverse flow classifier principle first described by Rumpf and Leschonski (1972). Leschonski (1977) has described a new version of the cross flow classifier (not yet commercially available) in which the classifying air is directed around a rounded elbow to increase the inertial effect and produce a finer classification.

Two types of combination flow classifiers are the tank through flow and the recirculating flow devices. The tank through flow classifier contains a rotor within a relatively large enclosure. An initial classification takes place as the feed particles enter a rising, rotating airstream. However, the airstream then passes through the central rotor, where the primary classification occurs. The fine particles are discharged from the classifier with the classifying air. Examples of the tank through flow classifier are the Majac Air Classifier and the Hosokawa/Mikropul Classifier.

The recirculating flow devices also introduce the feed particles into a rising airstream. The air-conveyed particles also pass through a central rotor. However, the airstream exiting from the rotor enters an integral cyclone collector and is recirculated back to the classifier. It has been shown (Luckie and Austin, 1975) that two separate classifying actions take place in the recirculating flow, increasing the chances of poorer classifier performance. Most classifier systems can be operated as recirculating flow devices. Examples of the recirculating flow classifier are the Georgia Marble Air Sifter or the Apline Mikroplex Spiral Type MVP Classifier.

Another type of combination flow classifier is the Static Microclassifier (Hukki and Airaksinen, 1980). This classifier design is particularly adaptable to series construction.

The specific energy for pneumatic classification should be strictly a function of fan power, since the seal requirements are relatively small. Therefore, the pneumatic classification specific energy consumption can be estimated from a fan power formula. This gives:

$$\frac{\text{kWh}}{T_{\text{product}}} = 0.068245\ \Delta Px \qquad 4.3\text{-}3$$

where ΔP = static pressure drop in inches of water, and x = ratio of pounds of air to pounds solid. Thus, for 10 in. H_2O and 3 $\text{lbs}_{\text{air}}/\text{lb}_{\text{solid}}$, the specific energy would be 2 kWh/ton. The specific energy increases with increasing fineness of product.

4.3.3.2 Wet Classifiers

4.3.3.2.1 Mechanical Classifiers

Mechanical wet classifiers are of a type that is basically a settling pool that uses simple gravitational forces to separate particles dispersed in a liquid. The feed slurry enters the pool at a rate such that only the coarser particles have time to settle out. The remainder are carried out of the pool as effluent. The devices in this group of wet classifiers differ in the way the coarse particles are removed. They may be removed by gravity or mechanically. Mechanical methods include drag, spirals, or reciprocating rakes.

The classifier in Figure 4-10 comprises a tank, one or more rakes, and a mechanism for actuating the rakes. The tank has parallel vertical side walls, a substantially vertical wall at one end, and a sloping bottom of such length that its upper end rises above the level of the top of the end wall. The rakes consist of a number of parallel blades set perpendicularly to the tank bottom and to the longitudinal axis of the tank. They are carried on a frame consisting of two parallel longitudinal members, suitably cross-braced, and with heavy by-plates from the driving mechanism. In operation the rakes move forward along the tank bottom a fixed distance, rise, move back the same distance, descend, and move forward once again. Thus, they move in rectangular pattern.

Feed slurry is introduced to this classifier and flows over a distributing apron toward the high end of the tank. The larger and heavier particles settle into the zone of the rakes and are raked up the slope and out of the tank. The smaller particles overflow the rear wall with the effluent. By proper adjustment of the height of the rear wall, a cut can be made at a definite size limit. When the coarse particles are being raked up the sloped wall, drainage is taking place, producing dewatered rejects. However, some portion of the fine particles, known as void filling particles, is also removed with the coarse particles.

These classifiers scale up volumetrically.

4.3.3.2.2 Hydrocyclone

The hydrocyclone shown in Figure 4-11 is a geometrically simple wet classifier that uses centrifugal force to separate particles dispersed in a liquid. The feed slurry, under pressure, enters a cylindrical chamber tangentially via a volute, causing the stream to rotate. The centrifugal forces thus produced cause the heavier particles to move to the outer wall. These particles then migrate downward in a spiral pattern into the cone, where they continue to spiral along the wall to the apex of the cone and are discharged to the atmosphere as underflow. The lighter particles migrate toward the center and spiral upward and out as overflow through a central pipe, the vortex finder, whose function is to prevent short circuiting of large particles in the feed to the fine particle outlet. The spiraling patterns are depicted in Figure 4-11(c), and the general nature of the particle trajectories was established by Kelsall (1952).

Since the forces driving the larger particles to the walls are greater than those driving the fine particles to the center, some portion of the fine particles do not have time to escape and pass down the wall with the underflow. These fine particles are referred to as void filling material.

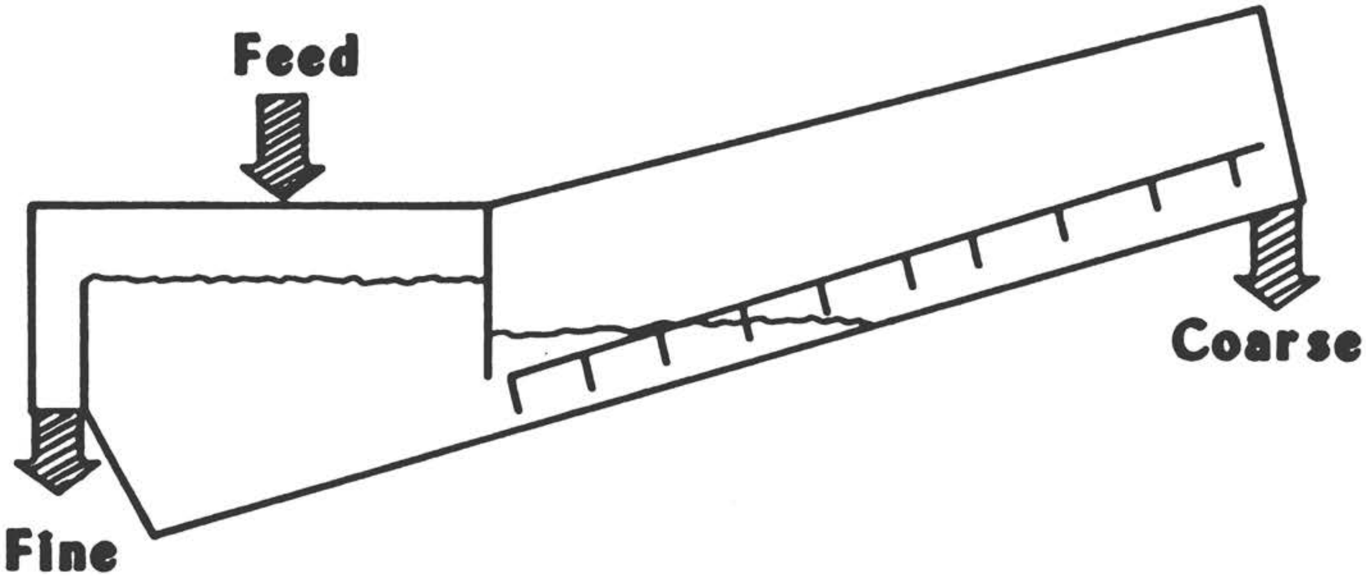


FIGURE 4-10 Rake Classifier.

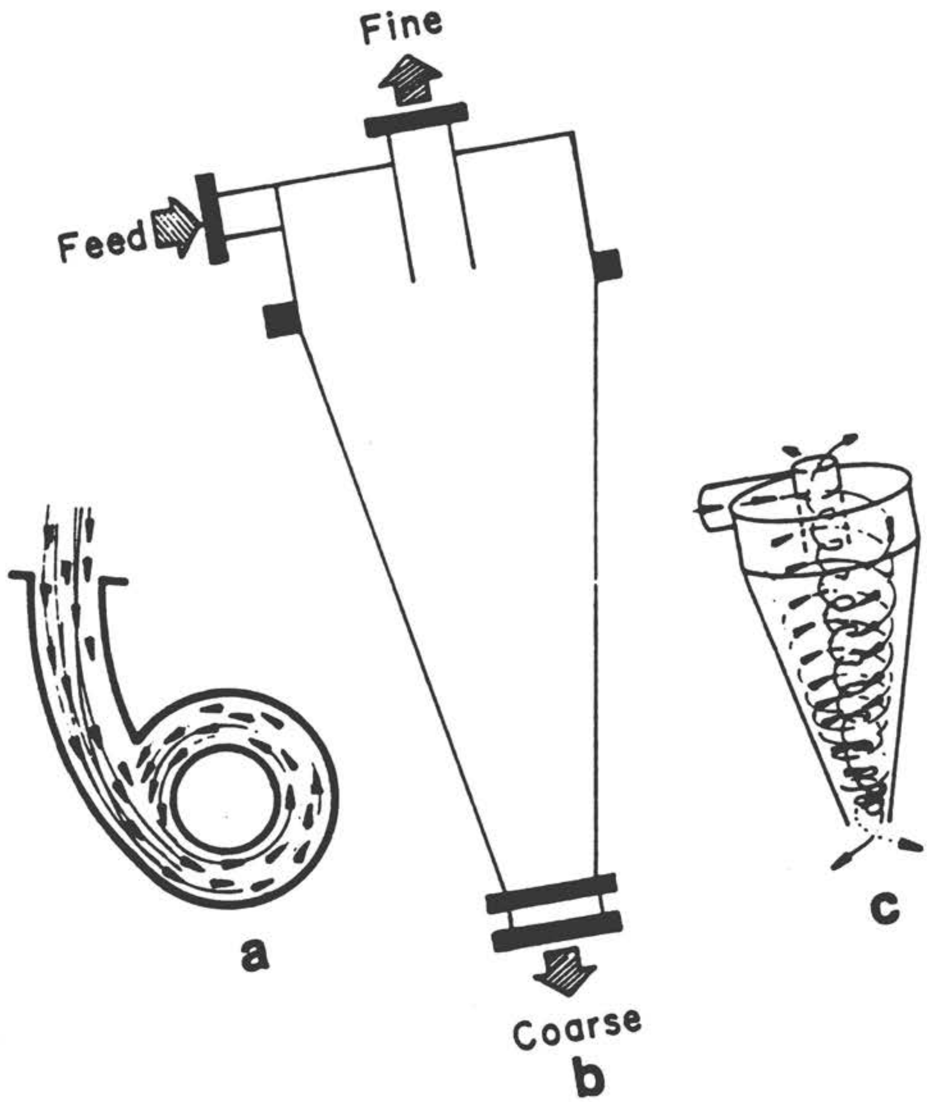


FIGURE 4-11 The Hydrocyclone.

Different particle size separations can be achieved by changing the diameter of the cylindrical chamber, the percent solids of the feed, and/or the feed rate. Bradley (1965) has reviewed the early hydrocyclone literature. Inoue and Imaizumi (1980) report a range of κ values from 0.25 to 0.625. However, Lynch (1977) reports that the κ value does not change with operating condition. This is not true for the Cut Size (d_{50}) and the Apparent Bypass (a). Both Lynch (1977) and Inoue and Imaizumi (1980) have found the a value to change with throughput and feed size distribution, while Lynch has confirmed that the d_{50} value changes with feed rate and solids concentration. Scale-up is via multiple units. Typical specific energy values rang from 0.4 to 0.6 kWh/ton (Arterburn, 1979).

The role of the hydrocyclone in wet closed circuit grinding systems is very important, and scale-up problems have been observed in larger plants (Rowlands, 1979). Although the mineral processing engineer has sought to eliminate finished product from classifier coarse product since the beginning of closed circuit grinding, it is probable that elimination of fines would seriously affect slurry transport characteristics and lead to decreased grinding efficiency. It would be of considerable value to establish methods for identifying the optimal fine recycle requirements for minimizing overgrinding without adversely changing slurry transport behavior. Methods of washing out the fines have been developed (Kelsall and Holmes, 1960), but in limited tests with such a Cyclowash unit, no increase in throughput was observed (Arterburn, 1979).

4.3.3.2.3 Sieve Bend

The sieve bend wet classifier (Figure 4-12) is a special type of screening device. The screen is a slotted deck made of stainless steel wedge bar oriented at right angles to the direction in which the feed slurry streams across the screen. The feed slurry is fed evenly across the entire width of the deck, tangentially to the screen. The full stream of a slurry flowing over the sieve bend decreases in depth in increments of about one quarter of the slot width each time it passes a slot. The result is a separation of the feed solids at a size considerably smaller than the opening in the sieve bend. The curved surface is only important to insure that the slurry layer stays in contact with the screen surface. The specific gravity of the particles has no influence on the fineness at which they are screened.

The effluent of the sieve bend is collected in the effluent chamber from where it is piped for further processing. The cake is discharged over the lip of the sieve bend and returned to the mill. This cake will contain some portion of the fine particles, or void filling material. Different slot widths in the sieve bend produce various particle separations. In addition, the rounding of the bars from wear will result in a decrease in the d_{50} value. This condition is controlled by reversing the direction of flow of the slurry across the screen surface. The a value will increase with increasing feed rate. These classifiers are scaled up linearly by increasing the width.

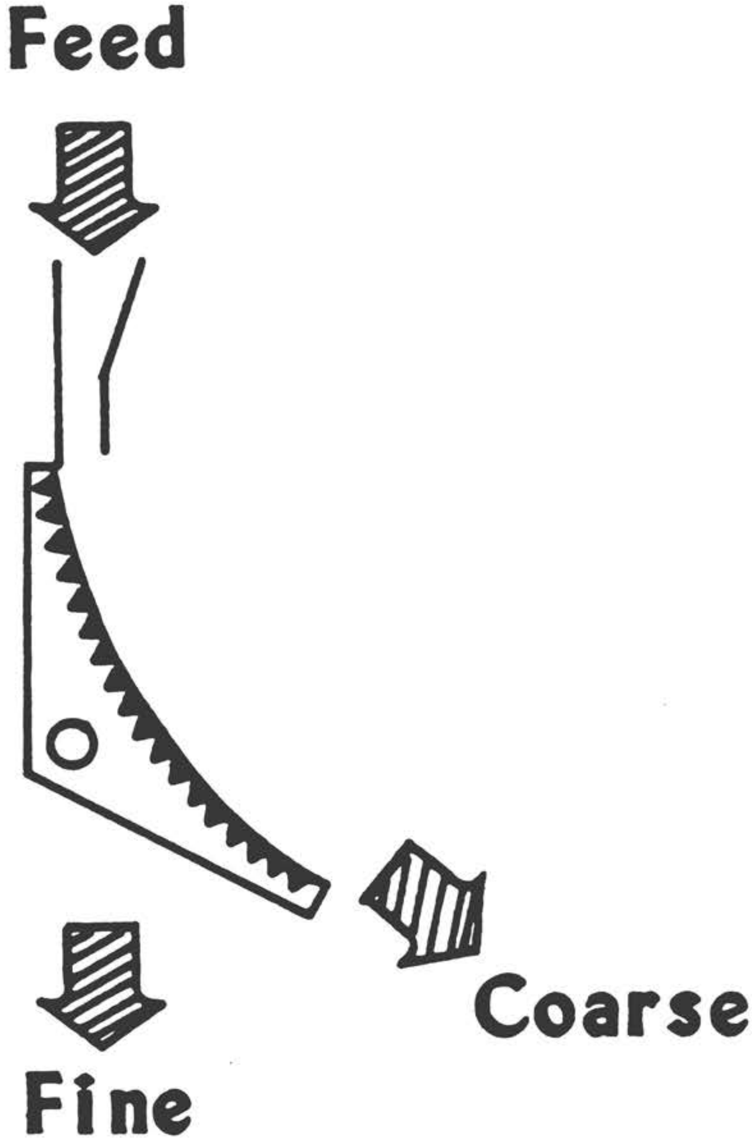


FIGURE 4-12 Sieve Bend Classifier.

4.3.3.2.4 Dilution

Wet classification has one inherent disadvantage--the dilution effect. Basically, the effect stems from two facts:

- o Particles must be dispersed to achieve good classification.
- o The fine particles go with the fluid.

Therefore, even if a wet grinding mill can be operated at concentrations approaching 50% solids by volume, this does not mean that the classifier can accept such concentrations. This is one of the reasons for including a sump between the ball mill and classifier. For example, the feed stream to a hydrocyclone is typically 30% solids by volume; the coarse stream would be a maximum of 45% solids by volume, and the fine stream would be around 15% solids by volume. For mechanical classifiers, the finer the product, the greater the dilution and hence the lower the percent solids in the effluent. It is this fact that makes closed circuit wet grinding unattractive for operations that need downstream concentrations of 40% to 50% solids by volume.

Dilution can become an even greater problem when attempting to improve the classification by removing the fines that fill the voids in the coarse rejects. Attempts to repulp and reclassify result in excessive dilution of the fines. Counter-current staging overcomes excessive dilution, but requires large operating areas. The hydraulic cone classifier (Figure 4-13) is a new device developed in Finland (Hukki, 1977) to remove void filling fines in the coarse stream without excessive dilution.

Feed material enters through the feed tube of this device to distribution discs. The upwardly directed laminar flow takes the fine fraction along. Fines are discharged as overflow. Coarse particles settle downward. Below the distribution discs this coarse material is kept in forced continuous motion to prevent sand layers from accumulating on the inside wall of the cone. Wash water is introduced under pressure through the wash water ring to the falling sand layer. The fine particles carried by sand fraction are separated and rise with the wash water through the central flow tube back to the classification zone. The washed coarse fraction is discharged through the apex valve as underflow. However, the device can only achieve a coarse classification.

4.3.4 TWO STAGE CLASSIFICATION

The probability of developing single stage classifiers whose performance is far superior to those currently employed in industrial practice is remote. An alternative is to accept the limitations of one stage classification, but correct them by reclassifying the impure coarse product in a separate, independent step. This approach is being pursued in both wet and dry classifications. The use of the hydraulic

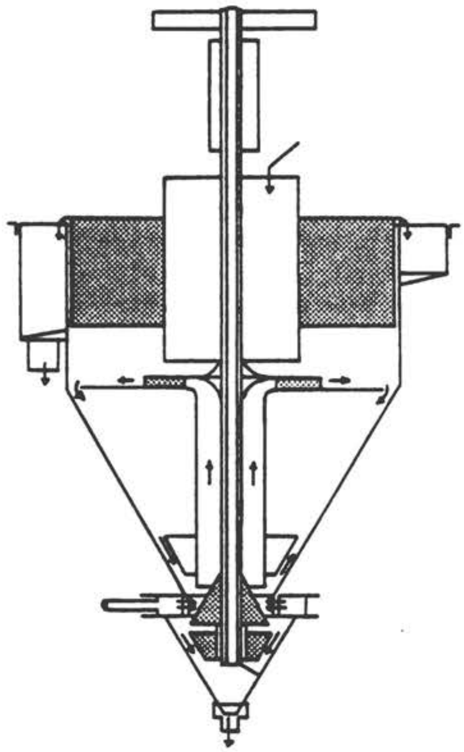


FIGURE 4-13 Hydraulic Cone Classifier.

cone classifier to reclassify hydrocyclone rejects is an example of two stage classification.

There are various arrangements of two classifiers in series, but the most common configuration is one where the coarse stream from the first classifier is fed to the second classifier and both fine streams are combined. The overall fine recovery for such a two stage classification is the sum of the individual classifier fine recoveries minus their product. Consequently, a claim for improved recovery must be met even if the second stage recovery is very small.

Examination of the overall selectivity values provides more insight into this type of operation. The chance of a particle's appearing in the coarse stream is the product of the chance of a particle's reporting to the first classifier coarse stream and the chance of a particle's reporting to the second classifier coarse stream. If both classifiers have the same size selectivity values without Apparent Bypass, and if both equiprobable sizes are 1.0 and the Sharpness Indices are 0.6, the overall equiprobable size should be 1.175 and the overall Sharpness Index should be 0.714. Therefore, two stage classification has improved the Sharpness Index, but at the expense of increasing the equiprobable size. If there is an Apparent Bypass, examination of the size selectivity values where they are equal to the Apparent Bypass shows that if each classifier has a 30% Apparent Bypass, then the overall Apparent Bypass should be only 9%. Consequently, a claim that the circulation ratio--the ratio of the coarse stream particle mass flow rate to the fine stream particle mass flow rate--is reduced must be met.

Another serial configuration is the case where the fine stream from the first classifier is fed to the second classifier, and both coarse streams are combined. The overall fine recovery for this arrangement is the product of the individual classifier fine recoveries. The overall chance of a particle's appearing in the fine stream is the product of its not appearing in the coarse streams of the first and second classifiers. In this case, the overall equiprobable size should be 0.8 while the overall Sharpness Index should be 0.55. Here, the equiprobable size has been reduced, but at the expense of decreasing the Sharpness Index. In addition, the overall Apparent Bypass should be 51%.

There are two other serial configurations. One is where the coarse stream from the first classifier is fed to the second classifier, but the fine stream of the second classifier is fed back to the first classifier. Here, the overall equiprobable size is increased to 1.1 while the overall Sharpness Index is increased to 0.725. The overall Apparent Bypass should be 11%. The overall recovery is the product of the individual recoveries divided by the quantity: the product of the individual recoveries plus one minus the second stage recovery. In the other case, the fine stream from the first classifier is fed to the second classifier, but the coarse stream of the second classifier is fed back to the first classifier. Here, the overall equiprobable size is decreased to 0.875 while the overall Sharpness Index is increased to 0.643. The overall Apparent Bypass should be 38%. The overall recovery is the product of the individual recoveries divided by the quantity: the product of the individual recoveries plus one minus the first stage recovery.

Obviously, the way to operate a two stage classification circuit is to optimize the type of classifier and the operating conditions for each stage in order to maximize the assets and minimize the liabilities. In addition, if the classifying system is an integral subsystem of a larger system, such as a size reduction circuit, the interaction between the two must be checked before claims of improved overall operation can be made. Further practical points to be noted in exploring two stage classification are the additional pumping power required and maintenance costs associated with higher wear rates.

4.3.5 PERFORMANCE OF CLOSED CIRCUIT GRINDING

Although reasonable characterization of the classifier is possible, predicting the results of closed circuit grinding is not as simple as it might appear. Interactions between the classifier and the grinding device make such a prediction very difficult. For example, the Apparent Bypass does not directly affect the size distribution of the fine stream. However, it will affect the fine stream size distribution indirectly, since the size distribution of the coarse stream contributes to the size distribution of the feed to the grinding device and hence the size distribution of the feed to the classifier. In addition, the mass flow rate of the feed to the classifier will change, which, in turn, affects both the Cut Size and the Apparent Bypass. Prediction is further confused because the size distribution of the feed material to the grinding device can affect the transport through the grinding device and the amount of holdup material in the device. Both of these affect the rate of grinding and hence the size distribution of the feed to the classifier.

One way to investigate the impact of these interactions between the grinding device and the classifier is to test actual systems. However, it would be very expensive and difficult to test all possible arrangements and operating conditions. A workable alternative is to develop a mechanistic model of the grinding device and simulate closed circuit grinding arrangements using the classifier performance relationships. Thus, the model becomes the testing facility.

Investigations of this type have been conducted for two conditions. The first is where comparisons are made among grinding circuit product size distributions with a common single control point. The second is where comparisons are made among grinding product size distributions with two common control points. The latter condition is essentially equivalent to producing the same size distribution.

For the first condition--a single control point--the following trends have been observed. As the Sharpness Index of the classifier increases, the fineness of the product size distribution decreases while the circuit output rate increases. As the Apparent Bypass increases, the fineness of the product size distribution increases while the circuit output rate decreases. As the Cut Size decreases, the fineness of the product size distribution decreases while the circuit output rate increases. Consequently, selecting a classifier on the basis of one of these operating parameters may not produce what it apparently should.

For example, while studying the twin cone classifier, Luckie and Austin (1979) determined that, for a specific air to coal mass ratio, the production rate declined with an increasing percentage passing 75 microns criterion. However, the vane setting that consistently gave the highest production rate was 50%. This condition was predicted by the model and observed in the testing facility. Thus, the modeling concept can be used to investigate circuit design as well as predict existing circuit operation.

The findings for the second condition--two control points--were very different. As the Sharpness Index increases, the Cut Size required to achieve the fit increases. The Apparent Bypass has no effect on the product size distribution or the circuit output rate. Therefore, the poorer performing classifier can be used, if it can achieve the necessary Cut Size, without loss of output--but only if the grinding device can handle the higher throughput rates. Consequently, there is relatively no change in the specific grinding energy.

The modeling approach can be extended to investigate closed circuit grinding with two stage classification. For example, a currently popular arrangement is one that takes the coarse stream from the first stage and reclassifies it, combining the second stage fine stream with the first stage fine stream and returning the second stage coarse stream to the grinding device.

Analyzing a single stage classifier by the single control point condition gives a relative production rate of 143 for $\kappa = 0.40$, $d_{50} = 37$, and $a = 0.30$. Adding a second stage classifer as in the described configuration with $\kappa = 0.5$, $d_{50} = 37$, and $a = 0.0$ increases the relative production rate to 149. Now the value of the increased production must be balanced against such things as the increased energy consumption, control requirements, and operating stability. Another example gives a single stage relative production rate of 131 for $\kappa = 0.4$, $d_{50} = 53$, and $a = 0.3$. Adding a second stage classifier with $\kappa = 0.4$, $d_{50} = 37$, and $a = 0$ decreases the relative production rate to 122. Thus the two stage classification reduces the production rate. The message here is that two stage classification arrangements for closed grinding circuits should be thoroughly analyzed before being installed.

4.3.6 RECOMMENDATIONS

Closed circuit grinding is a proven way to reduce grinding specific energy. However, several aspects of this procedure need to be studied further. For example, more information should be gathered on the energy consumption of classifiers, which obviously changes with the type of material and the fineness of the product. A performance evaluation procedure should be developed so that users of closed circuit grinding could also systematically assess the performance of their grinding circuits. Such data could be used to identify any pattern of poor performance of industrial classifiers because of Apparent Bypass and misplacement of particles. With such a pattern, methods of improving performance could be advanced.

Since the two stage approach potentially can improve classification performance, it should be thoroughly analyzed to determine the potential benefits and studied to determine actual benefits, difficulty of operation, and energy consumption. However, the major contribution to the improvement of classification would come from investigations into the fundamentals of classification. Then it would be possible to predict directly the effects of fluid/solid ratios, specific gravity, shape, etc. Such an approach is much more desirable than the indirect method of assembling several thousand evaluations and attempting to develop a pattern from the data.

4.4 Instrumentation, Control, Optimization

4.4.1 INTRODUCTION

After 20 years of development in comminution optimization and control, it is appropriate to review the achievements, to assess the impact on current practice, and to identify potential areas for progress. Although the motivation for this entire study was the current concern over energy at the national level, the industry has always been interested in this area because in most mineral processing operations the major energy consumption is in comminution, and energy has always been an important operating cost.

Traditionally, more attention has been given to tumbling mill operation, again because of the major energy demand of these units compared with other comminution unit operations. However, for many years any significant progress was frustrated by the difficulty of measuring the operating variables and observing the process performance. This in turn hindered the development of a fundamental understanding of the dynamic characteristics of grinding operations and the rational design of control systems based on such knowledge.

During the first half of this century, manual control was dominant; the typical operator's duty was to maintain a steady feed tonnage and avoid pump sump overflows. Periodically, product density would be checked by direct sampling and weighing, and compensating changes would be made in water flow or solids feed rate if required.

Slow changes in water additions, circulating loads, or feed rate requirements were observed during a normal eight hour shift, and early thoughts on control systems consequently were aimed at compensating for low frequency--i.e., slowly changing--input disturbances. In the past decade, with the advent of on-stream size sensing devices, a significant high frequency--i.e., rapidly changing--component has been identified and has led to more comprehensive control strategies.

In considering control strategies it is vital to define control objectives and operating constraints clearly. Failure to do so will lead to confusion and suboptimal control. Grinding control is generally a balance between product size requirements and feed tonnage requirements, constrained by mill capacity, classifier capacity, or slurry pumping capacity.

In modern practice, the dominant factor in mill requirements is maximum tonnage. Under these circumstances, full utilization of grinding capacity is required, and any change in feed grinding properties will result in a change in product size distribution.

Grinding systems have two distinct objectives:

- Meeting product size specifications, as in cement and coal operations
- Meeting liberation requirements in operations where a subsequent mineral separation process is involved.

In all cases the ultimate objective is economic.

Since variations in mineralogy and dissemination markedly influence the separation process, it follows that for optimal control the objectives of the grinding control system in terms of tonnage and/or size distribution will not necessarily be constant. Instead they will need to be adjusted in response to variations in mineralogical properties and their influence on the tonnage and value of the final product. Although this total control concept is simple to comprehend, few if any concentrators have such a system in operation.

The motivation for all control systems is improved performance by some economic measure, and this should be clearly understood by all staff associated with projects in process analysis and control.

This section of the report is meant to serve as an introduction to the current status of instrumentation, control, and optimization technology relevant to comminution in the mineral industry and to provide guidelines for further reading for those interested in obtaining more detail.

Historically, the mineral industry has been at least a decade behind the chemical and petrochemical industries in instrumentation and process control. The rapid progress achieved by these latter industries has been due largely to their aggressive pursuit of new techniques. The slow progress of the mineral industry has been related to several factors, including: (a) high demand for metals in the 1940's and 1950's, during which period close control and high recoveries were not a prerequisite for profitable operations; (b) technical problems relating to the harsh nature of mineral processing operations, sampling problems in slurry systems, and the inherent difficulty of measuring variables identifying properties of the solids; and (c) lack of knowledge of the basic process dynamics.

During the 1960's, the mineral industry began a serious attack on the problems of applying control and instrumentation to its particular needs. In the 1970's, several operating plants have quantified the benefits achievable with the instrumentation and control technology available today, particularly with the application of on-line computer control.

There are some areas in which new instrumentation--particularly primary sensors (listed later)--could have a significant impact. But the main task the industry faces in significantly improving current operating efficiencies is to understand the available technology, do the necessary economic evaluations, and initiate the appropriate action.

From work reported to date, it is not unreasonable to anticipate throughput improvements in the range of at least 5% to 10% by the application of known control and optimization strategies. In some cases, improvement in excess of 15% may result.

4.4.2 INSTRUMENTATION

The term, instrumentation, covers three types of equipment:

- Sensors, which provide a signal related to observed process variables

- Control modules, which manipulate the measured signal according to a predefined control algorithm to produce a control signal

- Final control elements, which receive the control signal and adjust manipulated process variables accordingly

This equipment can be either pneumatic or electronic. It can operate in a time continuous mode (analog) or in a time discrete mode (digital).

Control modules have developed historically from pneumatic devices to analog electronic and then to digital electronic systems. All of these types of modules have uses in modern comminution plants. The choice of the basic type depends on questions of reliability, price, and company standard policies. With the declining prices for digital microelectronic systems and their demonstrated reliability, the majority of installations in the 1980's are expected to be of this type. A major concern related to microprocessor developments is the natural desire of instrument manufacturers to maintain a competitive edge over other companies and to do so, in some instances, by designing hardware and software that is compatible only with their own lines of equipment.

Very few instrumentation companies produce the complete range of equipment, from sensors to final control elements, that normally will be required in mineral industry installations. It is important to the mineral industry, therefore, that mutually compatible data transfer systems be available to permit effective communication between microprocessor-based systems made by different companies. To some extent this is accomplished by standardized digital communication procedures (Steele and Matson, 1973), but communication between different microcomputer systems remains an area of concern to the industrial user.

In the general area of process control modules, the mineral industry is fortunate at this time to be able to use reliable equipment developed for other industries. By contrast, there are primary sensing problems peculiar to the mineral industry for which particular solutions have to be developed.

The most common process variables that must be measured for comminution control can be divided into two categories--those for which adequate sensors are available from other industries, and those for which sensors have had to be developed or need to be developed. Table 4-4 lists these variables, and Table 4-5 lists supplies of particular instrumentation and control equipment.

Table 4-4 COMMINUTION CONTROL VARIABLES

Sensors readily available from other processes/ industries	Sensors requiring special development for particulate systems
Weight (static or dynamic)	Solids mass flow
Liquid flow	Solids level
Pressure	Slurry level
Temperature	Slurry flow
Power	Slurry viscosity
Speed of rotation	Pulp density
Sound	Particle size
	Moisture content of solids
	Composition of solids
	Liberation

In all cases, control equipment must be selected with care because operating conditions in the typical concentrator are quite different from those in the general process industries, for which most of the instrumentation was intially developed. The need to select robust, reliable equipment is well known among the leading large contractors and units that do not meet the required standards have been eliminated as a result of in-house experience. Major new projects with modern control systems designed into them can be expected, therefore, to have suitable equipment specified.

A study of operator acceptance of some standard mineral industry instrumentation (White, 1974) shows areas in which instrument performance can be improved and is a first step toward solving the communication problem. The natural tendency to introduce new equipment into industry as rapidly as possible can lead to premature failures and industry resistance to new units. Thus it is important to insure that new instrumentation undergoes an appropriately long prototype trial in the industrial environment.

TABLE 4-5: Instrumentation for Mineral Processing

Company	Level	Tempera-ture	Samplers	Belt scales	On-stream composition analyzers	Hydrogen-ation concen-tration	Particle-size analyzers	Pulp density	Flow meters	Metal detectors	Re-corders	Con-trollers	Computer control systems
1	x	x				x			x		x	x	x
2	x			x	x			x		x		x	
3	x						x						
4	x												
5	x			x				x					
6			x										
7			x										
8	x												
9			x		x					x			
10			x		x					x			x
11					x								
12					x								
13					x	x							
14		x				x	x						x
15							x						
16								x					
17				x				x					
18		x				x			x		x	x	x
19										x			
20		x									x	x	x
21											x	x	
22		x										x	
23				x									
24	x	x									x	x	
25	x	x									x	x	x

1. Foxboro Company
2. Ramsey Engineering
3. Milltronics, Inc.
4. Western Marine Electronics
5. Nuclear Chicago
6. Galigher Company
7. Denver Equipment
8. Industrial Nucleonics
9. Harrison R. Cooper Assoc.
10. Outokumpu Oy
11. Applied Research Laboratory
12. Perkins Elmer Company
13. Beckman Instruments
14. Leeds & Northrup
15. Armco-Autometrics
16. Halliburton Company
17. Ohmart Corporation
18. Kent Instruments
19. Omicraft
20. Honeywell
21. Texas Instruments
22. General Electric Company
23. Merrick Scale
24. Taylor
25. Fisher Controls

Areas in which research is needed for new primary sensing elements include: cheaper particle size analyzers, including devices for sensing coarser size distributions in slurry flows; slurry viscosity meter; methods for sensing the degree of mineral liberation; measurement of particle size in dry solids; and on-stream measurement of moisture in filter cake.

In all cases, the need for continuing interaction between industry and organizations doing equipment development cannot be overstressed. Some German work was reported to the committee for developing an on-line particle size analyzer for dry systems based on the cross flow classifier principle (Leschonski, 1977). Although there is a considerable potential for the application of such a unit in the cement and coal grinding industry, it is important not to lose sight of the ultimate objectives, i.e., the properties of concrete and the combustion characteristics of coal, and of the fact that changes in the relationship between size and final properties invariably occur with natural changes in mineralization. The German group is also working on the use of the dry on-stream size analyzer to control dry classifier performance.

The most common forms of final control element are the valve for liquid and slurry flow control and electrical or electronic devices for adjusting the speed of pumps for fluid flow or electric motors driving conveyors or feeders for solids. In the majority of cases in comminution systems the required accuracy and sensitivity of final control can be achieved with the well established, currently used, final control elements. One area in which improvement might be made is in reducing the wear rate on valves controlling slurry flows.

The three principal variables controlled in grinding circuits are solid feed rate, sump water flow rate, and slurry flow rate in the classifier feed stream. The slurry flow rate is regulated using a variable speed pump. Because of high cost of variable speed drives, this option is sacrificed in many installations and the slurry flow is not directly controlled.

Two kinds of studies would be useful in relation to variable speed pumps. One would be theoretical studies to identify the conditions under which direct control of the slurry flow rate would be of significant value and to identify the general sensitivity of grinding circuit operation to variations in classifier feed flow. The second would be to seek methods of reducing the cost of variable speed drives for slurry pumps.

There is also a secondary optimization to be done in selecting pump size and using variable speed pumps. The wear rate on pumps can increase rapidly with increasing flow. The economic aspects of pump maintenance must therefore be included in overall optimization.

4.4.3 PROCESS CONTROL

The basic theory for modern control systems has been known for many years. Several texts cover both the fundamental and applied aspects of the subject (Takahashi et al., 1970; Shinsky 1967).

The simplest unit in a control system is the single feedback control loop, in which a measure of a single process variable is compared with the desired value (the set point) for that variable. An error signal proportional to the difference is used to regulate the variable.

A common control loop of this kind is for water flow control (Figure 4-14). Any change in upstream pressure caused, for example, by the opening of another offtake from the water main will cause a change in flow through the valve C. This change will be detected by the flow measuring device A, typically an orifice plate flow meter. The controller B determines the error signal $e = Q_s - Q_m$ and generates a control signal U which is a function of e. The control signal is applied to the control valve C with the control function defined to reduce the error e. The ability of this type of system to minimize flow disturbance is a function of the physical installation (basic engineering design), the response characteristics of the individual units in the control loop (A, B, and C), the type of control algorithm selected, and the tuning of the controller B.

The term process control is generally applied to a much larger system than the simple case given above--for example, grinding control, in which there are more than one measured and controlled variable. An obvious prerequisite for the development of the broader process control system is that all controllable inputs are individually controlled and operating smoothly. This prerequisite has not always been attended to in industrial attempts to develop grinding control systems. The majority of older concentrators could realize significant improvements at this simplest level of regulatory control, and this work has to be successfully completed before overall control of the grinding process can be attempted. Areas that must be examined in this initial phase are listed in Table 4-6.

Table 4-6 SOME FACTORS INVOLVED IN BASIC REGULATION AS A PREREQUISITE FOR GRINDING CONTROL

Fine Ore Feed	Mill Water	Slurry Flow
Bin design and flow behavior	Water supply system	Sump design
Bin discharge system	Individual flow control loop	Pump characteristics
Solids level alarm system	Head water systems	Slurry level control
Solids transport system	Sump water systems	Flow measurement
Solids weighing system	Auxiliary water flows	Density measurement
Solids flow control system	Return of floor washing	

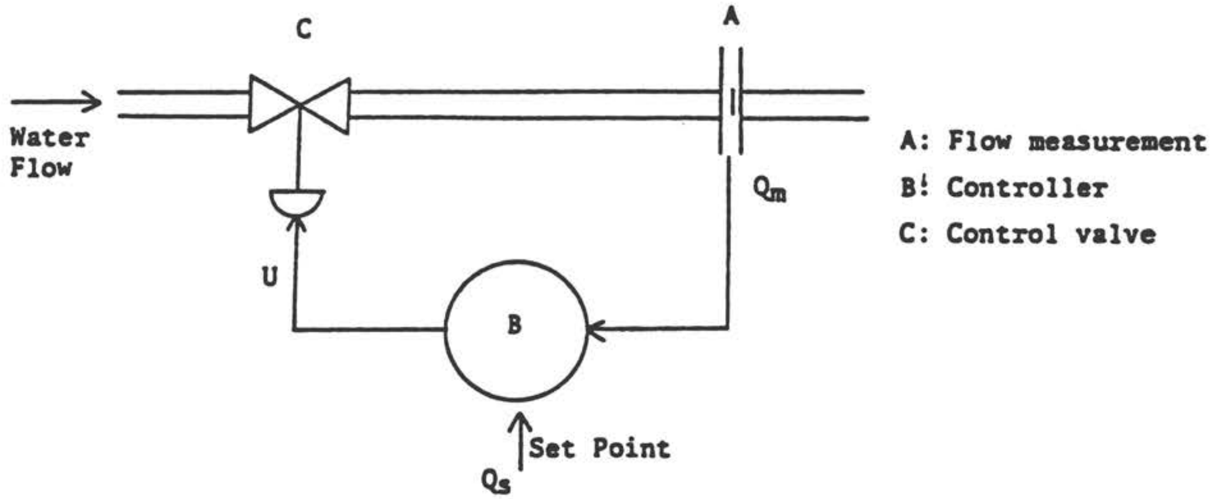


FIGURE 4-14 Water Flow Control Loop.

It should be noted that basic optimization of design variables should be carried out before beginning serious attempts to develop control systems. Such variables include: pump capacity, classifier geometry (e.g., orifice sizes for cyclones), ball loading, mill operating slurry density, etc. It should also be noted that basic design characteristics such as sump geometry, piping layout, and positioning of sensors and control elements can seriously influence control system performance.

Two approaches to crusher control have been developed. The first originated in Australia (Fewings and Whiten, 1973; Whiten and White, 1977) and is based on the extensive process characterization studies to develop accurate process models. Possible control system configurations can be designed from steady state model results and further refined by examining the overall dynamic characteristics. Systems of this type typically involve bin level measurement, bin discharge flow control, and crusher power control. Practical problems with filtering of crusher current signals need to be resolved for particular applications, and the system needs to be able to respond automatically to the different constraints possible in crushing plants with two or more crushing stages. A similar system has been successfully implemented in Canada (Hatch and Mular, 1978).

A second approach to crushing control has been developed in Sweden (Borrison and Syding, 1976). It is based on adaptive control theory (Astrom et al, 1974; Astrom, 1970), in which self-tuning regulators can eliminate much of the work required in plant experimentation and model development. This type of adaptive control system is in the very early stages of application in the mineral industry. Further work is needed on the inherent computational procedures, and plant application studies are needed to extend the range of approach.

Much time and effort has been put into developing different control strategies for grinding control. A recent review (Herbst and Rajamani, 1980) covers a representative selection of strategies in current use for conventional wet grinding circuits. Reviews by Bailey and Carson (1974) and Krogh (1979) cover autogenous systems. One of the problems associated with grinding control has been the difficulty of directly measuring the variables to be controlled. This problem has led to numerous schemes based on the measurement of secondary variables and an intriguing array of ways to use this information to regulate the limited number of controlled variables.

The simplest view of a wet grinding process is the black box approach illustrated in Figure 4-15.

External control can be effected only by regulating the three input flow rates, and changes in any one will influence the product. Grinding is immediately identifiable as an interactive control system, and for optimal solution of the control problem modern multivariable control theory (Athans and Falb, 1966; MacFarlane, 1972) will be required. In many practical instances extremely good control can be achieved by very simple classifical control loops. It should always be borne in mind, however, that multivariable control will give better results and that the magnitude of these benefits should be estimated before rejecting multivariable techniques.

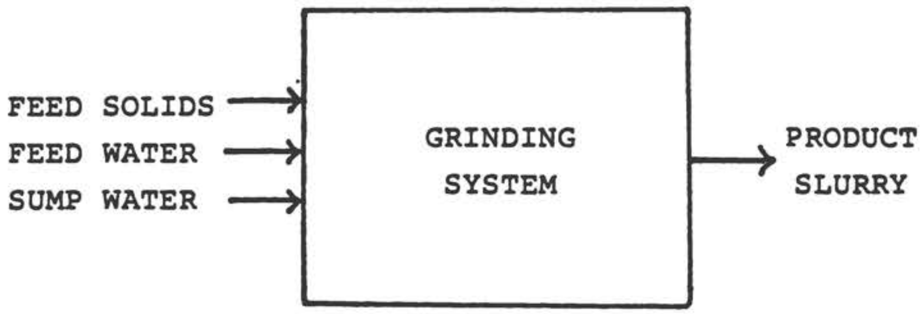


FIGURE 4-15 Black Box view of grinding control.

At the simplest practical level is should be noted that many of the older grinding installations in the United States do not have the basic ability to regulate the three input variables shown in Figure 4-15. Significant improvements in plant efficiency could be gained by correcting this situation.

The development of a good grinding control system is an interactive process involving four distinct activities (Figure 4-16). The potential gain to be achieved by progressing through these development steps can vary considerably, depending on (1) the age of plant, (2) the current level of instrumentation and control, and (3) the inherent variability of the ore. Ranges are given in Table 4-7.

Table 4-7 EXPECTED MARGINAL GAIN IN STEPWISE DEVELOPMENT OF CONTROL SYSTEMS

Stage	Range of Expected Marginal Gain	Notes
1. Process Analysis	0-10%	Low for well tuned plant. High for neglected plant.
2. Input Regulation	0- 5%	Low for regulated plant. High for unregulated plant.
3. Process Control	0-10%	Low for well controlled plant. Low for previously uncontrolled plant with homogeneous ore. High for previously uncontrolled plant with major feed disturbances.
4. Process Optimization	0- 5%	Low for inherently stable plants in a stable market. High for plants with major input disturbances. High if objective function changes frequently.

4.4.4 OPTIMIZATION

Optimization is the final level of activity in the logical progression toward improved process performance. There are four steps: (a) process analysis--finding out what a process is doing;

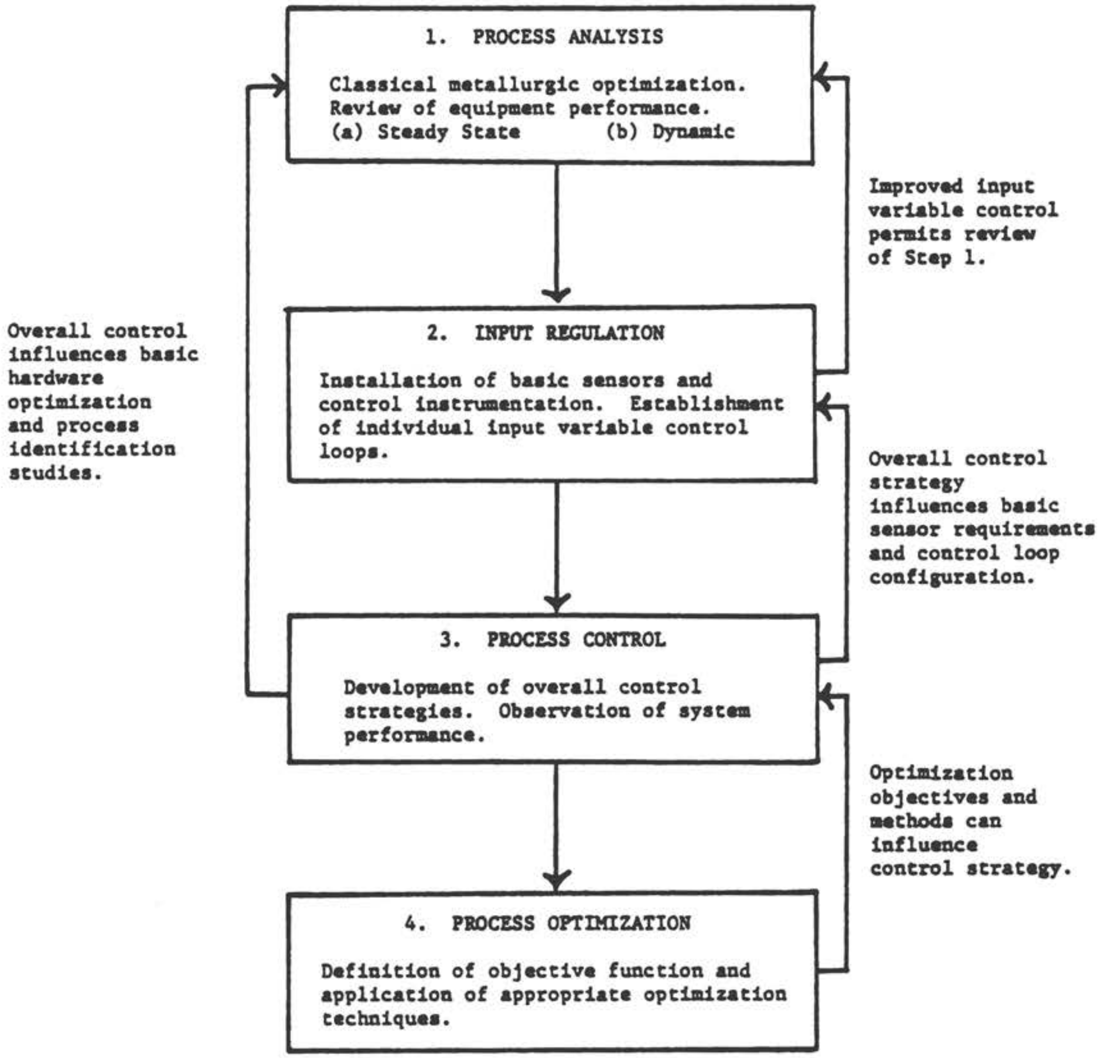


FIGURE 4-16 Iterative procedures in development of grinding control systems.

(b) input control--basic regulation of individual input variables; (c) process control--regulating the total process to achieve stable performance; and (d) process optimization--manipulating control of the process to achieve a defined optimal objective.

Process analysis preferably is carried out on an existing plant and involves a study of the dynamic response to imposed input disturbances. These studies help to identify the dynamic characteristics of the plant and aid in the selection and location of primary sensors and final control elements. For new projects in which direct testing on existing plants is not possible, a significant amount of process analysis can be carried out on dynamic models of the entire plant by using computer simulation techiques and a general knowledge of the dynamics of individual unit operations.

The process analysis stage also involves a review of equipment performance and modification of hardware to optimize the static performance of each unit, e.g., cyclone orifice sizes, mill ball load, etc. These are also equipment operating decisions to be made with respect to two or more units--for example, selection of the sizing of the feed to grinding to achieve an optimal balance between crushing energy (costs) and grinding energy (costs).

The full benefits of subsequent steps will not be achieved if basic process analysis, including equipment optimization, is not done effectively.

Process control involves the design and operation of closed loop systems to stabilize measured variables by the controlled regulation of selected process variables by the final control elements. In real process situations there is generally a significant interaction between control loops (Pitts et al., 1974), i.e., adjustment of one controlled variable will influence several sensors, which will cause changes in other control loops and ultimately the adjustment of other controlled variables. In these cases, selection and tuning of control strategies is an evolutionary procedure requiring considerable observation and interaction with the plant. The procedure is considerably facilitiated by the use of modern multivariable control techniques.

The process can be optimized after it is stabilized by adjusting set points within individual control loops to move overall process performance toward some defined overall objective. This objective can be static, i.e., maximum tonnage at all times, or dynamic, i.e., maximum profitability with allowance for changes with time in metal prices and cost of supplies, etc.

Optimization theory, like control theory, is well developed (Wilde and Beightler, 1967; Lee et al., 1968; Beveridge and Schechter, 1970; Ray and Szekeley, 1973) and its use well established in other industries. In recent years the mineral industry has shown the benefits to be derived from this technology particularly in mineral processing plants (Mular 1976; Boughan and Richardson, 1973; Paakkinen et al., 1973; Amsden et al., 1973; Herbst and Rajamani, 1980).

Optimization can be applied at many different levels. Once a system has been defined with appropriated boundaries, constraints, and objectives, it can be optimized. However, it is important to note that most processes yield a product which is an input to a subsequent operation and that suboptimization of the smaller unit in general will yield an operation strategy different from that obtained by optimizing the larger system.

As mentioned earlier, for overall optimization of a mineral processing plant, grinding control objectives will be influenced both by feed properties and by the performance of the separation process. The state of the art in mineral process optimization does not yet include optimization of the larger system, but significant progress has been made in the suboptimization of each section separately.

Several approaches are available for optimization strategies, which can be either on-line or off-line, and, in each case, can be computer based or not.

On-line strategies involve observing the process response to both natural and applied disturbances and systematically moving performance in the required direction. The overall optimal control is thus based on a measured response. Many different on-line strategies are available. Significant input to selection of the best one for a given system can be gained from plant staff, who have their own mental mode of system performance gained by invaluable direct operating experience.

An ultimate objective of on-line optimization strategies is to learn enough about the overall response of the system to be able to generate valid mathematical models. Such models permit optimization by off-line methods so as to minimize imposed disturbances and hence nonoptimal performance. In this case, the overall optimization strategy is based on a predicted response. With good models, it may be possible to apply control at an earlier stage than possible with on-line systems requiring direct measurement of the process response.

Although the main worldwide use of optimization in mineral processing has been in flotation, there is increasing awareness of its applicability to grinding. This trend is due both to the potential benefits arising in concentrators, where overall plant optimization requires the integration of comminution and separation processes, and to the potential gains in plants, e.g., coal and cement, where no concentration is required but signficant energy is required for grinding.

Capital costs and implementation costs differ at each stage of development, and the return on investment in optimization is not the same for each stage. The costs and benefits of the initial phase of optimizing basic hardware can vary from zero in a well-tuned plant to high costs and major benefits in older plants that have suffered from shortages of metallurgical staff.

The extent to which the development of control and optimization strategies can yield improved performances depends on the inherent stability of the process and, in particular, the type of feed property disturbances to be expected.

In comminution systems where long periods of uniform feed properties (size, hardness, moisture content, etc.) are standard, simple stabilization of the input variables at the optimal setting will yield a uniform product of the required specification. In this case, overall control schemes and optimization techniques will not yield significant improvement. By contrast, in plants with highly variable feed material it is essential to have good overall control systems, and further benefits can generally be achieved by optimization methods.

In the typical industrial environment, assessment of the benefits achieved in process control development projects is not always simple, and the contribution of the initial plant equipment tuning and close operator attention under test conditions is rarely identified. However, the literature reports many cases where productivity increased 10% to 15% as a result of overall process control development work.

An additional major cost is for staff. For successful process control projects the following key criteria must be satisfied:

- Motivation for the development must exist within the plant system. If plant personnel do not want a project to succeed, it will not succeed.

- A clearly defined project team should be formed. The project leader should be on the operating staff, preferably at the level of plant metallurgist or assistant superintendent.

- Skilled technical staff should be available for the project team, but need not be on the permanent plant operating staff.

A good discussion of project team requirements and project motivation is given by Hales and Burdett (1979).

4.4.5 GEOGRAPHICAL REVIEW

It is of considerable interest to note that the major thrust in the development and application of process control and optimization in the mineral industry has been in countries other than the United States. The leading nations in this effort have been Australia, Canada, and Finland, and there major gains have been made in flotation plants. However, since one of the most important factors in flotation control is regulation of the feed to flotation, optimal control of the overall mineral beneficiation process requires close control of the grinding system. For this reason, as well as the motivation to reduce grinding energy costs per se, there is increasing interest in the industry in the development and application of grinding control systems.

The work in Australia has centered on Mount Isa Mines Ltd., in Queensland, the country's largest copper producer. A research institute, the Julius Kruttschnitt Mineral Research Institute, was funded by the company and set up on the campus of the University of Queensland in Brisbane. The early thrust of this effort was directed at modeling grinding and classification units (Lynch et al., 1967; Fewings, 1971; Lynch, 1977). This work has led to a grinding control system (Lynch, 1977) based essentially on the prediction of product size using statistical models of classifier performance.

This approach has a considerable following in Australia (Anon., 1974; Lees et al., 1972; Whiten and Roberts, 1974; Draper et al., 1969) and has led to signficant improvement in grinding performance. These improvements have resulted not only from the installation of control systems, but also from the direction of attention to the comminution process itself. Most plants can realize signficant improvements by insuring that the existing equipment is operating effectively, and this point has been taught throughout the world since the beginning of mineral processing education. However, the basic equipment tuning or optimization is rarely fully exploited under the day-to-day pressures of the industrial environment. Furthermore, when benefits arise from this activity pursued as a necessary precursor to process control projects, the separate contributions are not always clearly identified.

In Canada the main effort was directed intially toward computer control of flotation processes (Smith and Lewis, 1969; Tait et al., 1970), but since has developed also towards the control and optimization of grinding systems (Webber and Diaz, 1973; Bradburn et al., 1977). Several companies (Kay and Patersen, 1973; Konigsmann et al., 1976; Amsden et al., 1973; Doyle, 1974; McManus et al., 1978) are active in developing or operating computer systems for grinding and/or flotation control and the majority are also interested in exploring optimization techniques.

In Finland all mining activities are operated by Outokumpu Oy. This company operates essentially as a normal commercial company even though it is owned and ultimately controlled by the government.

The Finns were among the pioneers in the development of both on-stream analytical equipment for the mineral industry and on-line computer control for flotation. After 20 years' activity, the level of modern control systems as a fraction of total national mineral processing capacity is probably far greater in Finland than in any other country (Paakkinen et al., 1973; Leskinen and Penttila, 1973; Eerola and Paakkinen, 1969). A clearly defined section within the technical export division of Outokumpu Oy is devoted to the development of basic equipment and control systems specifically for the mineral industry. The products of this effort are well respected internationally, and product sales are a significant portion of the company's revenues. To some extent this activity can be seen as a national program for research and development unequaled anywhere else in the world. It is a product of the combination of size, location, and organizational structure inherent in the Finnish system, but offers an example from which several signficant lessons can be learned, including:

- In several areas in process control technology, direct interaction between operating divisions and research and development groups can be fruitful.

- Major long-term investments in well selected technical areas can pay off.

- Equipment-oriented research and development within the industry can yield salable products.

- Stable research funding, including allowance for imaginative and aggressive promotion by technical personnel, can also be profitable.

On the U.S. scene there has been an increase in awareness and activity in comminution control systems in recent years. Significant impetus has been provided by the availability of an on-stream particle size sensor introduced at the beginning of the 1970's (Bassarear and McQuie, 1971). Control systems based on on-stream particle size measurement are described by Klee (1976), Perkins and Marnewecke (1977), and Bolles et al. (1977).

Because of the high cost of on-line size sensing devices, several workers have concentrated on grinding control schemes that do not use direct size measurements. Pitts et al. (1974) describe a system based on the Australian method of using a mathematical model of the grinding system from which product size can be inferred. Peterson and White (1974) identify methods for developing minicomputer control programs for wet grinding circuits.

There is also a wide interest in instrumentation and control in the USSR (Tikhonov, 1979). In grinding control, development has followed the basic pattern observed throughout the world. Control schemes based on constant feed rate, constant circulating load, and constant total feed to the mill have been explored. Instrumentation for the more difficult on-stream analytical tasks is not available, and all product size estimates are arrived at by operator observation or inference methods. The cost of energy is also of concern in the USSR, and there is strong motivation to reduce overall energy consumption.

One optimizing control strategy that has been implemented in the USSR seeks the maximum point on the output tonnage-circulating load curve (Figure 4-17). Steps of the order of 20% are imposed on the ore feed rate, and the circulating load response is followed for four hours. The point of inflection of the circulating load response gives an estimate of the critical circulating load, from which the optimal feed rate can be derived.

This approach has not been favored by operators because of the large feed steps required. To overcome this objection a dynamic search method is being evaluated which involves feed rate steps of only 2% and time intervals of the order of 15-20 min. If the circulating load is in the stable left-hand region as indicated in Figure 4-17 the response

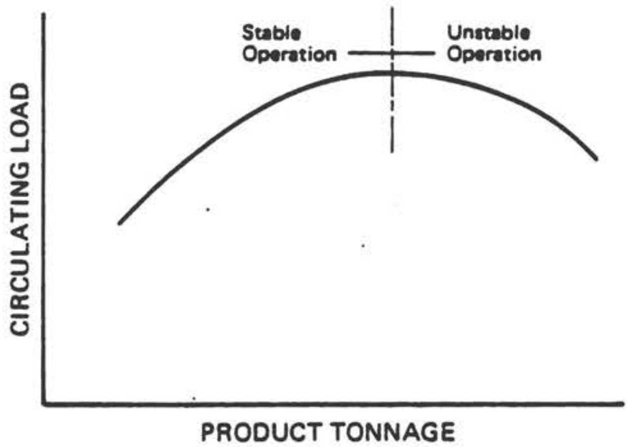


FIGURE 4-17 Product tonnage as a function of circulating load.

to incremental feed rate changes will be a classical step response which will rise rapidly at first and tend toward an asymptotic maximum. Once the critical circulating load has been reached, this response pattern no longer holds and the rate of increase of circulating load becomes larger. Implementation of this technique requires the estimation of derivatives of slow process responses in the presence of noise, and methods for solving this estimation problem have been developed.

As with all optimizing control systems for grinding that seek maximum mill loading, the consequences of an error are very serious. Once the critical circulating load is exceeded, it does not take long to choke the mill. Operator acceptance of these methods will therefore develop slowly and only after well proven, fail-safe procedures are established.

4.4.6 POTENTIAL FOR IMPROVEMENT

The benefits to be obtained from control and optimization of comminution systems are mainly related to the lowering of the energy requirements to achieve the required degree of size reduction. In most practical cases, the gain will take the form of an increase in tonnage, with both product size distribution and net energy consumption being held essentially constant. In some cases a finer grind at the same tonnage and energy consumption will be adopted if significant improvements in final product grade and/or recovery are realized with the finer feed material.

The possibility of maintaining previous tonnage and size performance while using less energy is not often achievable in practice because of the nature of comminution machines. However, in multiunit concentrators, energy consumption can be reduced incrementally by shutting down one or more units. The achievable benefits result from more efficient use of energy and reduced energy input per ton of product.

The relative energy inputs into the three main areas of comminution --blasting, crushing, and grinding--on the average are of the order 1:2:20. However, the ratio can vary considerably with the particular mineral commodity. It is immediately apparent that the greatest potential for improvement lies with grinding, and that energy savings result from maximizing comminution in the crushing stage.

All current comminution methods in large scale use yield product size distributions that include certain amounts of unwanted fines that represent unproductive use of energy. The net energy required to produce a fixed relative reduction in size (i.e., to reduce particles to a fixed fraction of their original size) increases as the initial size becomes smaller. For example, the energy required to reduce a 1-m block to 10-cm size is very much less than the energy required to reduce the same weight of 1-cm particles to 1 mm. Therefore, any improvement in current technology that can reduce the proportion of fines generated will result in improved energy utilization.

In blasting, much progress has been made in reducing dust and fines production, but there is room for further improvement. There are two approaches to this problem: optimal placement of charge with respect to geological conditions (Persson, 1973) and design of new rock breaking methods with controlled application of the explosive force (Bligh, 1974). There is also a need to evaluate the potential benefits arising from more uniform blasting and closer blasting control to produce a more uniform run-of-mine product. It is readily apparent that excessive fines and oversize rocks represent inefficient operation. Research is also needed to determine the extent to which narrowing of blasting product size distribution represents (a) more effective use of the blasting energy, and (b) a more favorable feed to the crushing operation from either energy or operating veiwpoints. See Section 4.8 for a discussion of current blasting research.

In a presentation to the committee (Wells, 1979), it was pointed out that a critical objective in blasting control is to achieve a product size appropriate for the subsequent materials handling requirement, i.e., bucket capacity, haulage capacity, etc. General practice, therefore, is to use a safety margin in designing blasting patterns to avoid the costly loss of time that results when excessively large boulders are produced and secondary fragmentation is necessary.

The majority of crushing systems today have minimal instrumentation and spend a significant portion of their running time in seriously underloaded conditions. A key factor in all crushing and grinding systems is uniform and optimal loading of the units. The rate of feed to the grinding circuits is more easily regulated than that to crushers, and a major area for improvement in crushing operation is in feeding systems and their control.

Some interesting work in crusher control has been done in Australia (Fewings and Whiten, 1974), Canada (Hatch and Mular, 1978), and Sweden (Borrison and Syding, 1976). In the Swedish case, an adaptive control system run on a computer 1800 km from the plant improved performance remarkably and at the same time verified the feasibility of using remote computing power and of adaptive control of a crushing plant.

The extent to which generation of fines can be minimized in grinding systems is strongly related to classifier performance (See Section 4.3). Benefits will result both from closer control of existing classification devices and from the development of new ones with sharper classification performance characteristics.

In older plants there is considerable scope for improving crushing and grinding performance by installing the required instrumentation and developing control schemes. In crushing plants, where it is conventional to run less than three shifts, improved operation will lead to net energy saving since total operating time will be reduced. As mentioned earlier, net energy for grinding systems is unlikely to be reduced, but the gains in efficiency will be used to obtain either increased tonnage or finer product for the same energy consumption.

The major obstacle to applying known control systems to older plants is the high cost of installing new instrumentation in old plants. In more modern plants, the level of instrumentation is much better, and the obstacle listed above for older plants is not applicable. However, there is often a lack of appropriate information systems for management and a poor availability of skilled technical support. Problems have been created in some modern plants by installing more instrumentation than was required. At several concentrators the arrays of unused instruments illustrate this point.

For mineral plants of the immediate future, enough is known to permit rational specification of primary sensing units, basic control equipment configuration, and final control elements. This can be done to permit the evolutionary creation of sound operating control schemes to accommodate the practical idiosyncrasies of the operating plant, idiosyncrasies that can never be known at the drawing board. With current microelectronics, these basic control systems can be simply included in an overall computer control system with provision for the development of both on-line and off-line optimization.

The incentives for improved control are the economic benefits of more efficient operation. As the cost of energy used in mineral processing increases, as it surely will in the 1980's, it is hoped that industry will respond by tackling the educational problems involved in achieving the necessary levels of management support and awareness and the technical training required to implement the available technology.

4.4.7 RESEARCH NEEDS

There is a need to encourage academic research into advanced control and optimization technology, particularly as an aid to fulfilling training needs. The potential impact of such work in the short term, however, is less than the results known to be achievable by the application of available technology.

The main area requiring work as we enter the 1980's is communication. There is an urgent need for easily accessible information tailored for industrial personnel at all levels. There is a particular need for methods of making available to management simple estimates of control project costs and potential benefits so that this information can be included in corporate decision making procedures.

Corporate decision making will continue to be based on direct economic estimates, and projects will be geared to maximizing company profits. As energy costs increase, all avenues available to decrease the impact of this trend must be explored.

There are, however, other factors to be considered that arise from the present importance of energy on the national scene. For example, if government sponsored measures are to be adopted to encourage more efficient use of energy than purely corporate economics can justify, many new research areas open up as the larger optimization problem is addressed. In this case perhaps the most important initial aim of research will be to meet the first requirement of all control and optimization projects--the clear definition of objectives and constraints.

There is a continuing need for research and development of primary sensors of simple, robust design and reasonable cost. The availability of on-stream particle size analyzers in the past decade has had a major impact on control of grinding circuits. However, the cost of this equipment is still sufficiently high to inhibit rapid installation throughout the industry. Both units currently available are based on relatively sophisticated technology and may yield more information than is needed for effective control of grinding circuits. Research is needed for the development of a simpler unit to give, for example, a single estimate of size once every minute at a price about one third that of particle sizers currently available, i.e., less than $10,000 (1979 prices).

For many years, slurry flows in pipes have been measured to acceptable accuracy by magnetic flow tubes. More recently, ultrasonic flow devices have become available; with further development, improved accuracy and lower cost are possiblies.

In control system and instrument development there is a natural tendency to refine performance to a level far beyond that pertinent to the dynamic response of the total plant. It is important, therefore, to maintain a clear view of total system objectives and capabilities and insure that individual instrument sensitivity is appropriately matched to overall system performance.

As mentioned above, comminution plants yet to be built should be able to take advantage of the benefits of control and optimization strategies currently available. However, the major potential for improving energy efficiency is in existing plants, and research aimed at simplifying the introduction of instrumentation into older plants should therefore be encouraged.

Recent studies in control of interactive systems are applicable to comminution processes. Further work in these areas, including development of practical adaptive control algorithms, is of considerable relevance.

Research in modern control theory in relation to comminution systems should be aimed at demonstrating the applicability of existing new concepts to real plant situations rather than seeking the development of more advanced control systems.

One problem facing recent engineering graduates is the lack of familiarity with real control systems in a real industrial environment. A very worthwhile research effort would be the development of fully instrumented, computer controlled pilot plant facilities at university locations for detailed student training and for special training of groups of engineers from industry. Training could also be achieved through dynamic simulation techniques using a computer.

As outlined above, there are many different approaches to grinding control, with different degrees of instrumentation and of dependence on directly measured or inferred values for key process variables. Research to quantify the benefits achievable by various control systems on the same grinding unit, or preferably on parallel units, would be very useful in identifying the conditions under which specific systems would have particular advantages.

Operating companies have their own constraints on the time that can be spent on the evolutionary work necessary to develop an optimal control system. Research to develop a clear understanding of the conditions under which different schemes are to be preferred and the marginal gains that are achievable by further modifications would be very useful to industrial staff responsible for the most economic operation of mineral processing plants.

4.5 Corrosion and Wear

The principal direct operating costs in commercial comminution operations (crushing and grinding) are the energy consumed and metal wear. This section deals with metal consumption in comminution. The metal wear in crushing and grinding systems is an expensive item in the mineral industry because of the cost of grinding media and worn equipment parts, decrease in production, and downtime for maintenance (Bombled, 1972). In addition, metal wear is a source of pollution and contamination.

The term metal consumption usually includes the metal worn away by abrasion, removed by corrosion, and lost as scrap. Approximate values for average scrap losses are 60% of the total metal consumption for crusher liners, 35% for rod and ball mill liners, and 20% for crushing roll shells (Bond, 1964).

4.5.1 MECHANISMS AND FACTORS AFFECTING METAL WEAR

Metal wear loss in wet grinding is 8 to 10 times that in dry grinding and in crushing. This may be due to actual dissolution of iron from the highly active, nascent fresh metal surfaces produced in grinding. Bond (1964) has stated that metal consumption in wet grinding circuits is increased further by grinding in an acid pulp, especially when the pH is lower than 5.5.

Metal wear in wet rod mills is 15% to 40% higher than in wet ball mills when based on the weight of steel used per unit of comminution energy consumed. In wet ball mills, the grinding media wear is about 13 times that of the mill liner, and in dry ball mills it is about 10 times. In wet rod mills, the rod wear is 11 to 12 times that of the liner wear (Bond, 1964). The factors affecting metal wear in comminution, in general, include:

(a) Metal composition: Most of the grinding media and liners are made of ferrous alloys. The chemical composition governs the ability to develop particular wear resistant structures [(b) below] and to resist chemical corrosion.

(b) Manufacturing and treatment (balls, rods, or liner pieces): The manufacturing technique (melting, casting, and cooling) controls the final metal structure, which has an important effect on the rate of wear of the metal. In addition, the heat treatment (surface hardening) affects the hardness of the grinding element (balls, rods, or liner) as a function of the depth from the surface where the surface becomes harder than the core. In general, hardness is inversely proportional to the rate of wear within the same element. Figure 4-18 presents an example of the effect of heat treatment on the hardness across a grinding rod and a grinding ball.

(c) Design: It has been found that the liner design affects both the grinding efficiency and the rate of metal wear. Since maximum grinding efficiency combined with minimum liner wear rate and maximum liner life guarantee lower grinding cost, the efforts that achieve this goal are justified (Dunn, 1976).

(d) Mineral properties: The properties of the material being ground, namely hardness (Bombled, 1972; Voigt and Clement, 1974) and particle size (Bond, 1964; Clement and Voigt, 1972) also have notable effects on the rate of metal wear in comminution devices. Table 4-8, reported by Bond (1964), shows the effect of the mineral hardness on the metal abrasion index. The particle size of the material to be ground influences liner wear, which can be uneven along the mill length, decreasing from the feed end to the discharge end of the mill (Tsypin et al., 1976). This means that the coarser size material is more abrasive to the liner than the finer size. Laboratory tests designed to quantify the rate of metal wear under specified conditions revealed that the metal wear rate decreases with decreasing particle size, reaches a minimum, and then increases in the finest sizes (Uetz and Fohl, 1972). This may be due to macroscratches made by the coarse feed compared with the polishing of metal by the fine feed. It has also been observed that the rougher the metal surface, the higher the wear rate (Abouzeid et al., 1978).

(e) The pulp properties: The solid/liquid ratio, pH, and ionic concentration also influence the rate of metal wear. The major effect may be in changing the electrical potential difference between the pulp and the metal parts, which will influence the rate of metal dissolution (Bombled, 1972; Voigt and Clement, 1974).

The metal losses in comminution systems have been attributed mainly to abrasive wear and/or corrosion. The rate of abrasion depends mainly on the physical properties of the material being ground, particularly hardness and size; operating variables such as mill speed, load and solid/liquid ratio; and the type of steel used. Abrasion is a combination of high-stress wear in which the crushing strength of the ore is exceeded, and low-stress (scratching) abrasion where the stresses created are below the ore crushing strength (Nass, 1974). Spalling and breakage have also been reported to contribute to the metal losses in comminution (Wick et al., 1979).

In wet grinding, the rate of metal loss by corrosion is proportional to the potential difference between the pulp and the surface of the metal as well as the physical properties of the ore and type of steel. This potential is developed as a result of the dissolved ions and the pH of the pulp. Some measurements have indicated that the potential of the pulp in wet grinding may be positive to the mill shell by as much as 0.5 V and is probably also positive to the grinding rods and balls (Bond, 1964).

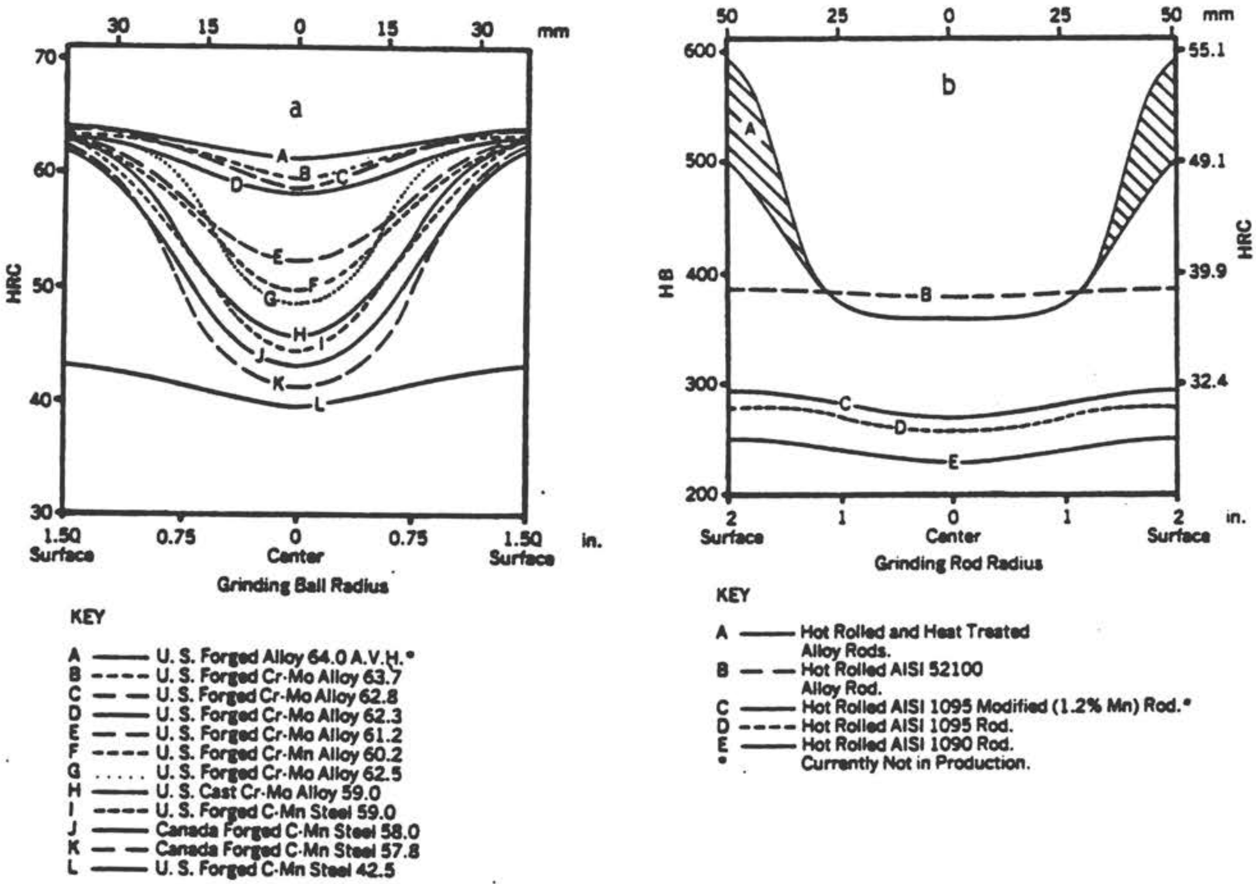


FIGURE 4-18 Cross sectional hardness profiles of commercially available 75 mm diameter forged and cast-steel grinding balls (a), and 100 mm diameter steel grinding rods (b). (After Nass, 1974)

TABLE 4-8: Abrasion Averages

No.	Material	Ave.	Sg	Wi	P	Ai
1.	Dolomite	5	2.7	-	-	.0160
2.	Shale	5	2.62	9.9	11,700	.0209
3.	L.S. for Cement	14	2.7	12.7	12,830	.0238
4.	Limestone	9	2.7	11.7	-	.0320
5.	Cement Clinker	8	3.15	13.5	13,070	.0713
6.	Magnesite	3	3.0	-	-	.0783
7.	Heavy Sulfides	10	3.56	11.4	12,000	.1284
8.	Copper Ore	24	2.95	11.7	12,700	.1472
9.	Hematite	7	4.17	8.5	13,450	.1647
10.	Magnetite	2	3.7	13.0	-	.2217
11.	Gravel	4	2.68	15.4	12,950	.2879
12.	Trap Rock	20	2.80	17.8	14,400	.3640
13.	Granite	11	2.72	16.6	14,630	.3880
14.	Taconite	7	3.37	16.3	-	.6237
15.	Quartzite	3	2.7	17.4	-	.7751
16.	Alumina	7	3.9	17.5	15,800	.8911

Sg = Specific Gravity
Wi = Bond Work Index
P = Product 80% passing size, microns
Ai = Abrasion Index

The influence of metal structure can be very important, and work to design alloy composition and structure for particular abrasion resistance has been done (Thomas, 1979a). In most cases, structures favoring impact resistance do not have good abrasion resistance and vice versa. However, by careful heat treatment steps it is possible to generate a fine metallic crystalline structure that is particularly resistant to both impact and abrasion. The cost benefits of these materials have not yet been fully evaluated, but the approach appears to offer a potentially fruitful area of research for reducing the high costs associated with corrosion and wear in comminution.

4.5.2 QUANTIFYING METAL CONSUMPTION

Some attempts have been made to quantify the rate of metal wear in comminution (Bombled, 1972; Bond 1964; Uetz and Fohl, 1972). Among these attempts, Bond (1964) proposed an experimental procedure for standardizing the evaluation of the rate of metal loss so that it could be related to the physical properties of the ore and the operating conditions of the comminution process. He designed a system for determining the weight loss of a rotating paddle (632 rpm) inside a rotating drum (70 rpm) in the presence of 1.6 kg of 18 x 12 mm feed material for one hour under specific conditions. He defined an abrasion index, Ai, as the loss from the weight of the paddle in grams in a single test. Full details of the procedure are given in the original paper.

Using steel consumption data from various mineral processing plants, Bond attempted to relate the abrasion index, Ai, to the metal consumption under various working conditions. The data collected include wet rod mills (rods and liners), wet ball mills (balls and liners) for overflow and grate discharge openings, dry ball mills (balls and liners) for grate-discharge openings, and gyratory, jaw, and cone crushers (liners).

No single relationship relates the rate of metal consumption and the abrasion index in all of these comminution processes. Figure 4-19 presents the rate of metal consumption (kg/kWh) and the abrasion index for balls, rods, and liners under different operating conditions (wet and dry) (Bond, 1964). It is obvious that Figure 4-19 shows only an average trend of the rate of wear of metal under different operating conditions with various types of mills or crushers as a function of the abrasion index. Table 4-8 gives the average work index, Wi, and the average abrasion index, Ai, for tests on various materials, and Bond pointed out that the correlation between the abrasion index and the work index is very poor. However, Figure 4-20, which includes most of the data from Table 4-8, together with Figure 4-21, presents the abrasion index as a function of work index and hardness of materials (Mohs' scale), respectively, and shows that general trends can be observed.

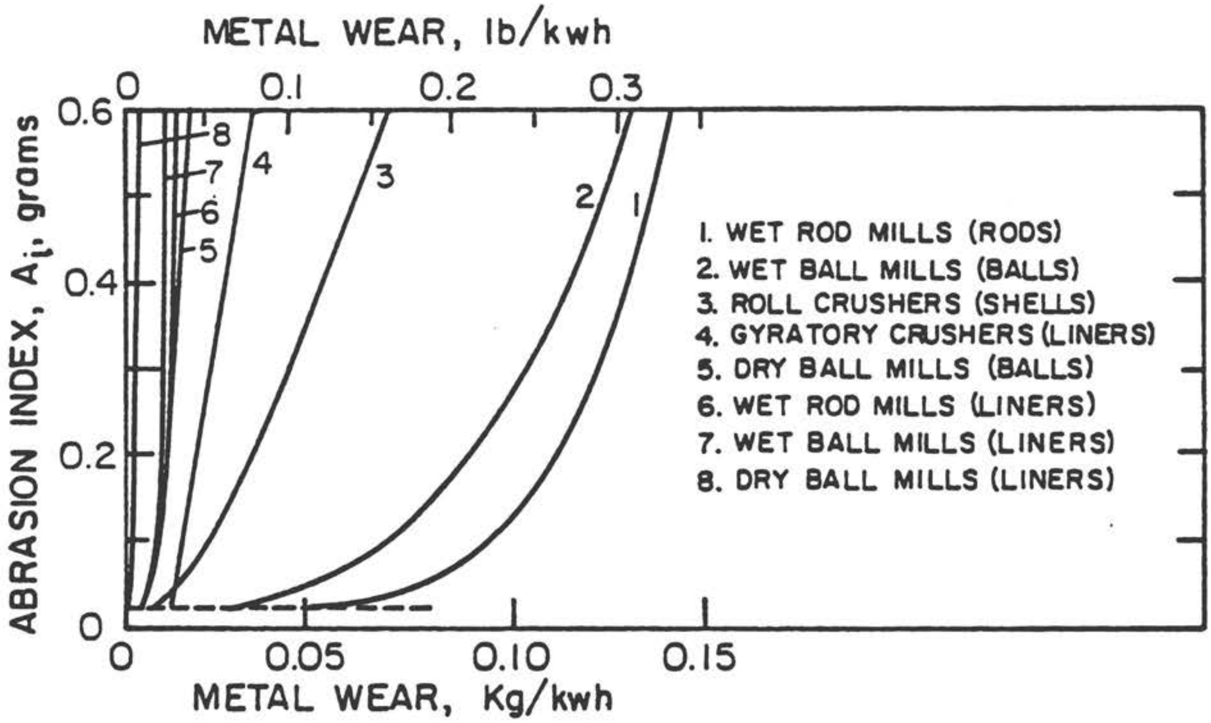


FIGURE 4-19 Metal wear (kg/kWh) as a function of abrasion index, Ai, in various comminution systems under different operating conditions (wet and dry). (After Bond, 1964)

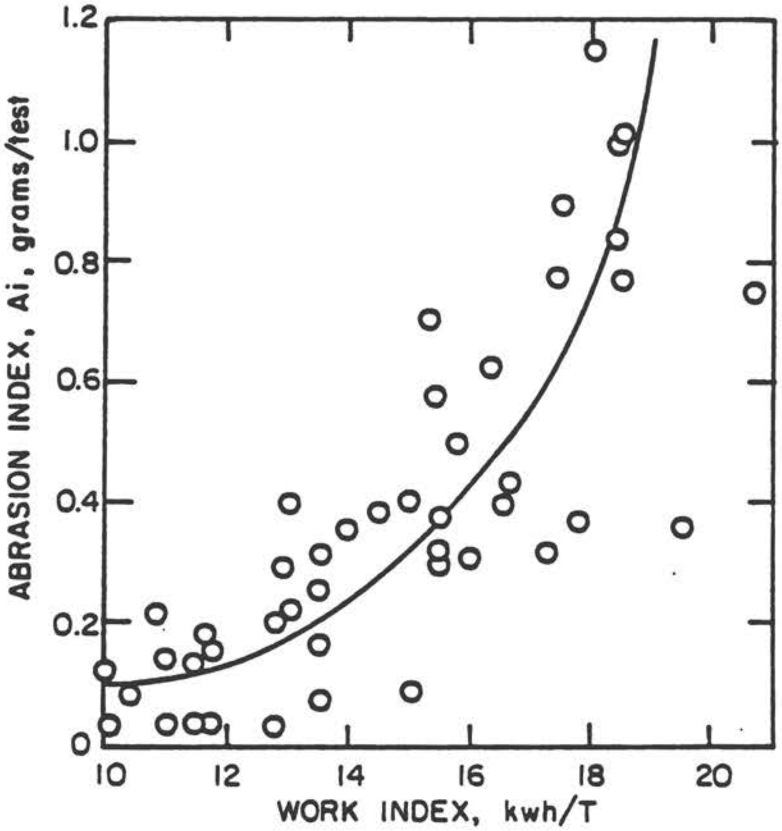


FIGURE 4-20 Work index, Wi, plotted as a function of abrasion index, Ai, for a variety of materials using the data of Bond (1964).

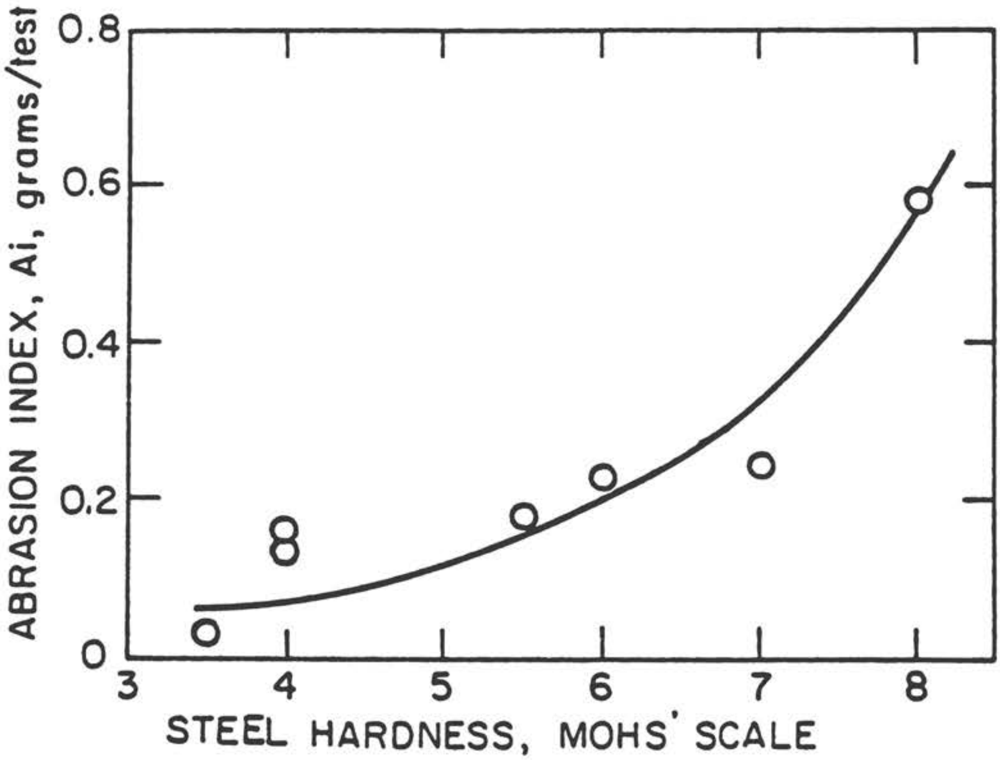


FIGURE 4-21 Abrasion index, Ai, plotted as a function of the mineral hardness in Mohs' scale using the data of Bond (1964) and standard mineral hardness values.

4.5.2.1 Liner Wear

Steel consumption in comminution processes in the mineral industry in the United States and Canada in the form of mill liners amounts to about 150,000 tons annually. The steels and iron used in grinding mill liners may be classified into several general types. The composition of each type is listed in Table 4-9. Table 4-10 gives hardness, wear rate, and merits and extent of usage of each type (Norman, 1974). The rate of wear of metal liners appears to depend to a large extent on the metal hardness (see Figure 4-22), although it may not be the only criterion.

Rubber can now be used as mill liners successfully and economically in a wide range of fine grinding conditions and to a limited extent in coarse grinding operations in ball and rod mills. However, it is not a suitable material for liner in cement clinker mills, where the high operating temperatures quickly destroy wearing qualities. Rubber tends to provide the best service when the abrasive particles are small (as in fine grinding) because the feed particles are free of hard or sharp corners or edges, and the mill speed is not very high (less than 75% of critical speed) (Norman, 1974). Figure 4-23 shows the trend of rubber liner life as a function of the fraction of critical speed and feed size. One can see that the life of the rubber liner is shorter with coarse feed than with fine feed and shorter at higher speeds than at lower speeds.

Ceramic linings have been used for quite a long time in many fine-grinding mills, particularly pebble mills. The available high alumina ceramic shapes have good abrasion resistance, resistance to spalling, relatively high strength and toughness, resistance to heat and chemical attack, and adaptability to economic production and installation under present-day conditions (Norman, 1974). In some cases, these ceramic linings can economically replace metal and rubber linings. Where contamination of the ground product by wear of metal or rubber linings must be avoided, ceramic linings may solve the problem.

Ceramic materials are too hard (hardness is 9 on Mohs' scale) to be scratched or worn away, even by abrasive minerals such as quartz. Ceramics probably wear by a microspalling mechanism. If favorable conditions can be maintained in the mill, a ceramic lining may last 10 times longer than metal or rubber linings (Norman, 1974). It seems that rubber and ceramic materials may have potential for replacing steel linings in particular comminution processes. If this happens, the rate of consumption of steel linings might be reduced.

The total cost of lining should include the cost of liner materials, the cost of installation, and the cost of shutdown time for repair or relining, which in turn is a measure in terms of lost production. Since the cost of lost production varies considerably, depending on conditions at each milling operation, it is hard to put down an average value for this item of the lining cost. Table 4-11 presents the average price per kilogram of metal and the installation cost per kilogram for the various lining materials, estimated for the United States in December 1973 (Norman, 1974).

TABLE 4-9: Ferrous Materials for Grinding Mill Liners

Item	Material	Composition Range (%)[a] C	Mn	Si	Cr	Mo	Ni	Cu	Hardness Range[b] HB	Relative Wear Rate[c]
1	Martensitic Cr-Mo White Iron	2.4-3.2	0.5- 1.0	0.5-1.0	14.0-23.0	1.0-3.0	0.8-1.5	0.0-1.2	620-740	88-90
2	Martensitic High-Carbon Cr-Mo Steel	0.7-1.2	0.3- 1.0	0.4-0.9	1.3- 7.0	0.4-1.2	0.0-1.5		500-630	100-111[d]
3	Martensitic High-Cr White Iron	2.3-2.0	0.5- 1.5	0.8-1.2	23.0-28.0	0.0-0.6	0.0-1.2		550-650	98-100
4	Martensitic Ni-Cr White Iron	2.5-3.6	0.3- 8.0	0.3-0.8	1.4- 2.5	0.0-1.0	3.0-5.0		520-650	105-109
5	Martensitic Medium-Carbon Cr-Mo Steel	0.4-0.7	0.6- 1.5	0.6-1.5	0.9- 2.2	0.2-0.7	0.0-1.5		500-620	110-120
6	Austenitic 6Mn-1Mo Steel	1.1-1.3	5.5- 6.7	0.4-0.7	0.5 max	0.9-1.1			190-230	114-120
7	Pearlite High-Carbon Cr-Mo Steel	0.5-1.0	0.6- 0.9	0.3-0.8	1.5- 2.5	0.3-0.5	0.0-1.0		250-420	126-130
8	Austenitic 12Mn Steel	1.1-1.4	11.0-14.0	0.4-1.0	0.0- 2.0	0.0-1.0			180-220	136-142
9	Pearlitic High-Carbon Steel	0.6-1.0	0.3- 1.0	0.2-0.4					240-300	145-160
10	Pearlitic White Iron	2.0-3.5	0.3- 1.0	0.3-0.8	0.0- 3.0				370-530	ND[e]

[a]The composition range from an individual supplier will normally be narrower than the ranges given in the table. Production of certain compositions falling within the above ranges may involve proprietary rights covered by patents or trade marks.

[b]Hardnesses listed are on the unworn surface of the liners. Austenitic and martensitic alloys tend to work harden on wearing surfaces.

[c]Relative wear rate as determined when wet grinding minus 3/8-in. (10-mm) feed to minus 48 mesh on ore containing about 65% quartz, 25-30% feldspar, and 3% pyrite as the principal abrasives, in primary ball mills at Climax, Colorado.

[d]A composition containing about 1.0% carbon, 5 to 6% chromium, and 1.0% molybdenum was used as a comparative standard in all tests and assigned a relative wear rate of 100.

[e]Wear rates in the Climax mills could not be determined on this material due to spalling and breakage.

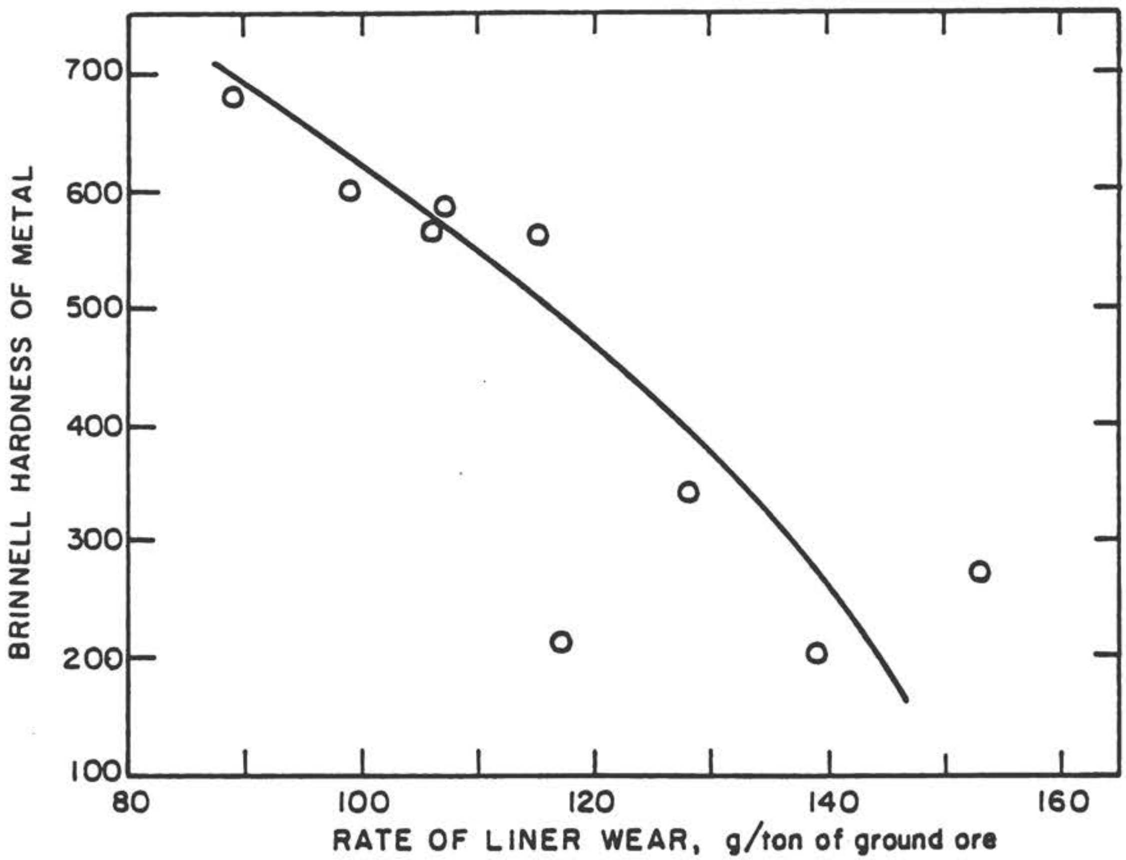


FIGURE 4-22 Relationship between the rate of liner wear and the Brinell hardness of the liner material. (After Norman, 1974)

TABLE 4-10: Hardness, Relative Wear Rate, Properties, and Usage of the Different Types of Steels used in Comminution

		Brinnell Hardness	Relative Wear Rate g/ton	Remarks
1	Martensitic Cr-Mo White Iron	620 - 740	88 - 90	1. Can be hardened by heat treatment. 2. Has the best abrasion resistance. 3. Has good resistance to spalling and breakage. 4. Not suitable under high-impact conditions. 5. High initial cost.
2	Martensitic High-Carbon Cr-Mo Steel	500 - 630	100 - 111	1. High-carbon steel. 2. Can be hardened by heat treatment. 3. Abrasion and spalling resistance. 4. Develop high internal stresses, somewhat prone to cracking during service.
3	Martensitic High-Cr White Iron	550 - 650	98 - 100	1. Heat, corrosion, and abrasion resistance. 2. Little use in grinding mills, widely used in cement mill liners.
4	Martensitic Ni-Cr White Iron	520 - 650	105 - 109	1. Good to excellent abrasion resistance when used in liners. 2. Less resistant to breakage and spalling. 3. Suitable for medium and low impact conditions in ball mills.
5	Martensitic Medium-Carbon Cr-Mo Steel	500 - 620	110 - 120	1. Can be hardened by oil quenching. 2. Used for rod mill liners and liners for high impact ball mills. 3. Good toughness and abrasion resistance.

TABLE 4-10 (Continued)

		Brinnell Hardness	Relative Wear Rate g/ton	Remarks
6	Austenite 6Mn-1 Mo Steel	190 - 230	114 - 120	1. Tough and abrasion resistant. 2. Provides a good liner in ball mills operating under relatively high impact conditions.
7	Pearlitic High-Carbon Cr-Mo Steel	250 - 420	126 - 130	1. The lower carbon grade is used for ball mill grates and for ball mills and rod mills with high impact conditions. 2. Excellent resistance to spalling and breakage. 3. More abrasion resistant than some austenites when used in ball mills. 4. Less predictable in rod mills. 5. Preferred for ball mills and rod mills as lifter bars because of its higher yield strength and greater resistance to flow and plastic deformation. 6. Can be used for liner balls to minimize or eliminate breakage, and provides better service than austenite.
8	Austenitic 12 Mn Steel	180 - 220	136 - 142	1. Can be used for high impact rod mills. 2. Provides high toughness, moderate cost, availability, and fairly good abrasion resistance.
9	Pearlitic High-Carbon Steel	240 - 300	145 - 160	1. Less usable now. 2. If used, will be in the form of as-rolled or rails.
10	Pearlitic White Iron	370 - 530	ND	Disappeared because of inferior abrasion resistance, low toughness, lack of quality control in production, and relatively high price.

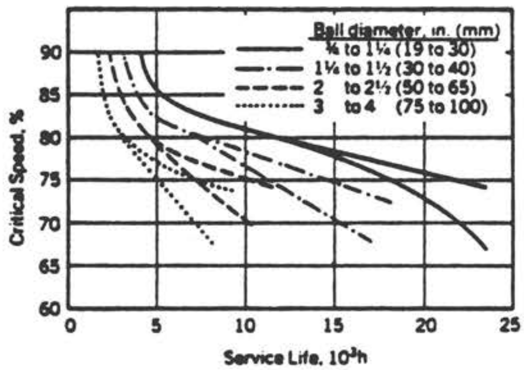


FIGURE 4-23 Service life of rubber liners as a function of mill speed (expressed as a fraction of the critical speed) in the presence of grinding balls of various diameters. (After Norman, 1974)

TABLE 4-11: Typical Relative Costs of Materials Used for Linings in the Western U.S. - December 1973[a]

Item[b]	Material	Cost Per lb	Installation Cost Per lb[c]	Installed Cost Per lb	Relative Weight of Complete Lining	Relative Installed Cost of Lining[f]	Toughness Rating[g]
1	Martensitic Cr-Mo White Iron	0.38	0.04	0.42	100	42	6
4	Martensitic Ni-Cr White Iron	0.30	0.04	0.34	100	34	7
5	Martensitic Cr-Mo Steel	0.36	0.04	0.40	100	40	5
6	Austenitic 6Mn-1Mo Steel	0.32	0.04	0.36	100	36	3
7	Pearlitic Cr-Mo Steel	0.28	0.04	0.32	100	32	4
8	Austenitic 12Mn Steel	0.30	0.04	0.34	100	34	2
10	Pearlitic White Iron	0.22	0.04	0.26	100	26	8
R	Rubber	1.17	0.10	1.27	23[d]	29	1
C	Ceramic	0.37	0.18	0.55	44[e]	24	9

[a]Typical costs and weights are based on averaged data from a number of large ore-milling operations where comparisons of the various materials have been or are being made.
[b]Items 2, 3, and 9 in Table 4-9 are omitted from Table 4-11 due to lack of comparative data.
[c]Installation cost includes cost of bolts plus installation labor, except on rubber, where bolt cost is included in cost per lb, and on ceramic where cement was used in place of bolts.
[d]Relative weight of rubber lining includes weight of bolts and metal reinforcement.
[e]Ceramic assumed to have same lining volume as the ferrous materials.
[f]Relative installed cost of lining does not include cost of lost production during shutdown for relining.
[g]Toughness rating refers to resistance to spalling or breakage as determined by service experience. Highest order of toughness = 1 and lowest = 9.

Consumption of metal mill linings may range from a high of about 0.30 kg per ton of mill feed for the wet ball milling of a coarse abrasive ore to a low of about 0.5 gm per ton in dry grinding soft limestone or cement clinker. In kilogram per kWh (Bond, 1964), the corresponding range may be from abut 23 gm per kWh down to about 0.05 gm per kWh. In fact, the most important situation in which liner wear is economically significant is in primary grinding of low-grade ores that contain high percentages of abrasive minerals (such as quartz). During 1970-1973, liner wear in the mineral industry ranged from 0.05 kg per ton to 0.08 kg per ton. Table 4-12 gives some examples of the rate of liner wear in some plants in this country, together with mill dimensions and feed specifications.

The life of a liner in a mill ranges from 200 to 2000 working days. The time depends on material properties, grinding media (rods or balls), grinding conditions (wet or dry), mill design, liner design, and mill speed, among other affecting variables.

4.5.2.2 Grinding Media

High tonnages of ore ground by the autogenous process in the United States and Canada consume no grinding media since grinding balls are not required in such systems. However, probably more than 1 million tons per year of forged and cast-steel grinding rods and balls are consumed by the mining industry in United States and Canada. Fair quantities of this steel are cast Ni-hard and high-chromium white iron alloy grinding balls.

Grinding media weight loss depends, as mentioned above, on milling and slurry conditions. Abrasion and corrosion, again, are the main mechanisms responsible for metal wear in dry and wet grinding, respectively. Because of acid conditions in the mill slurry in wet mills, steel consumption can be up to 10 times that of dry grinding. For this reason, some milling operations use grinding balls of nonferrous materials, such as ceramics, to achieve acceptable grinding costs. This may also be done if iron contamination of the product is to be avoided. However, a major disadvantage of ceramic material is its low density--almost 3.4 gm/cm^3 for sintered alumina ceramic compared to steel grinding balls at 7.8 gm/cm^3.

In 1973 an extensive survey of the steel consumption in grinding media in the United States and Canada (Nass, 1974) revealed that the three main types of grinding balls are: forged carbon and alloy steel, cast carbon and alloy steel, and Ni-hard and high-chromium cast iron. On the other hand, it was reported that the types of rods produced in the two countries are hot-rolled AISI 1090 to 1095 carbon steel, heat treated alloy steel, hot-rolled alloy steel and hot-rolled AISI 1020 to 1085 carbon steel. Table 4-13 gives the distribution of balls and rods among these types of steel alloys in both countries.

In the United States the major user of grinding media is the iron ore industry, which consumes 56% of the grinding balls and 69% of the grinding rods produced in the country. By contrast, the principal

TABLE 4-12: Wear Rates of Mill Linings in Primary Grinding

Plants Surveyed	Units	Mill Sizes ft (m)	Ore	Feed Size Ranges in. (mm)	Liner Wear, lb/ton (kg/metric ton) max	min	ave
8	Single-Stage Ball Mills	9 x 9 to 13 x 12 (2.7 x 2.7 to 4.0 x 3.7)	Cu-Mo	-3/4 to -3/8 (-19 to -10)	0.150 (0.075)	0.080 (0.040)	0.102 (0.051)
3	Double-Stage, Rod Mill plus Ball Mills	10.5 x 14 to 11.5 x 16 (3.2 x 4.3 to 3.5 x 4.9)	Taconite	-1 to -3/4 (-25 to -19)	0.153 (0.077)	0.070 (0.035)	0.099 (0.050)
5	Double-Stage, Rod Mill plus Ball Mills	10 x 13 to 12.5 x 16 (3.0 x 4.0 to 3.8 x 4.9)	Cu-Mo	-3/4 to -1/2 (-19 to -13)	0.146 (0.073)	0.088 (0.044)	0.106 (0.053)

TABLE 4-13: Types of Steels used in Grinding Media in the United States and Canada

Grinding media	Type of steel	Distribution of steel (%) U.S.	Canada
Balls	Forged carbon and alloy steel	77	81
	Cast carbon and alloy steel	20	--
	Ni-Hard and high-chromium steel	3	19
Rods	Hot-rolled AISI 1090 to 1095 carbon steel	62	89
	Heat-treated alloy steel	21	--
	Hot-rolled alloy steel	17	--
	Hot-rolled AISI 1020 to 1085 carbon steel	--	11

user in Canada is the copper industry, which consumes about 56% of the balls, billets, and slugs and 43% of the grinding rods (Nass, 1974).

4.5.2.2.1 Grinding Balls

Because of production conditions and heat treatment of steel balls, the hardness of the balls produced varies across the ball diameter (see Figure 4-18). Average volumetric hardness values are used by Canadian and U. S. steel companies to describe the hardness of grinding balls. While grinding ball wear rates can generally be minimized by maximizing the average volumetric hardness, this may not always be desirable. Under certain milling conditions, balls of high hardness slip execessively, thereby producing lower mill throughputs. Only testing will determine, however, if the increased cost in ball consumption is offset by the increase in mill tonnage.

To obtain maximum grinding efficiency, uniformity of roundness is desired. Out-of-round balls receive higher than normal stresses which may generate splitting or spalling. Ball cost in grinding systems decreases as ball diameter increases up to balls of diameter 5 cm, where the cost levels off (see Figure 4-24). This can be due to more exposed surface area to be eroded per unit weight of balls in the case of small balls, in addition to less efficient grinding when small balls are used.

4.5.2.2.2 Grinding Rods

Average volumetric hardness may also be used to express the hardness of the rods; however, grinding rod hardness is more commonly specified by an average surface core hardness. Figure 4-25 presents grinding rod consumption rates at Reserve Mining Company over about 17 years compared to rod hardness values. The company has found that it can lower grinding rod consumption by raising the average volumetric hardness of the rods (Nass, 1974). It also found that grinding rod scrap rates could be optimized by varying rod diameters and alloys. Table 4-14 shows the comparison of grinding rod diameter and alloy composition to rod percent scrap weight. For desirable mill operating efficiency, rods should grind to small diameters without bending and break up into pieces of 6 to 18 in. (150 to 450 mm). High impact strength and high toughness are not always desirable in grinding rods since rods having high impact strengths may not break up at desired thicknesses but rather coil up in the mill and hence reduce grinding efficiency. Excessive rod breakage can subject rod mill liners to high impact loads causing shell liners to crack. Shell liner breakage may also result from incorrect grinding rod length. Warped or bent grinding rods will also reduce grinding efficiency. It is possible today to obtain a commercial rod straightness tolerance of 1/4 in.

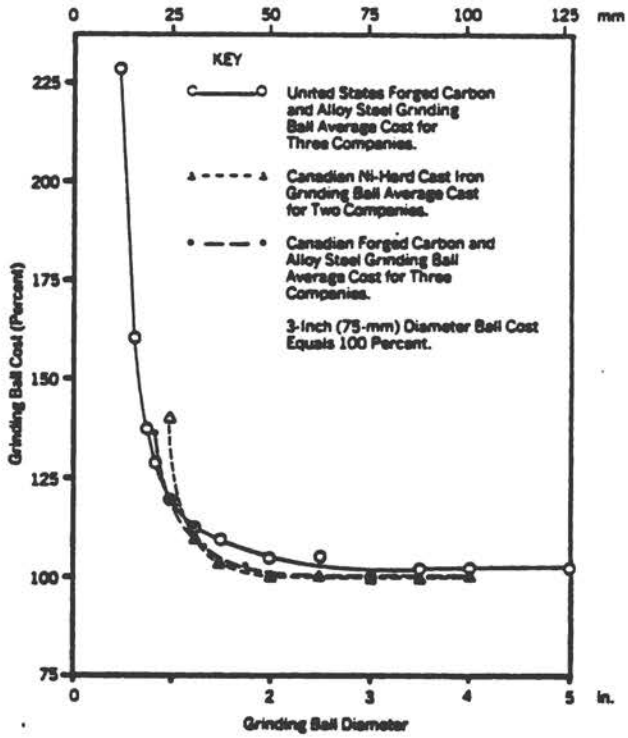


FIGURE 4-24 The relation between grinding ball diameter and grinding ball cost expressed as a percent of the cost of 75 mm diameter balls. (After Nass, 1974)

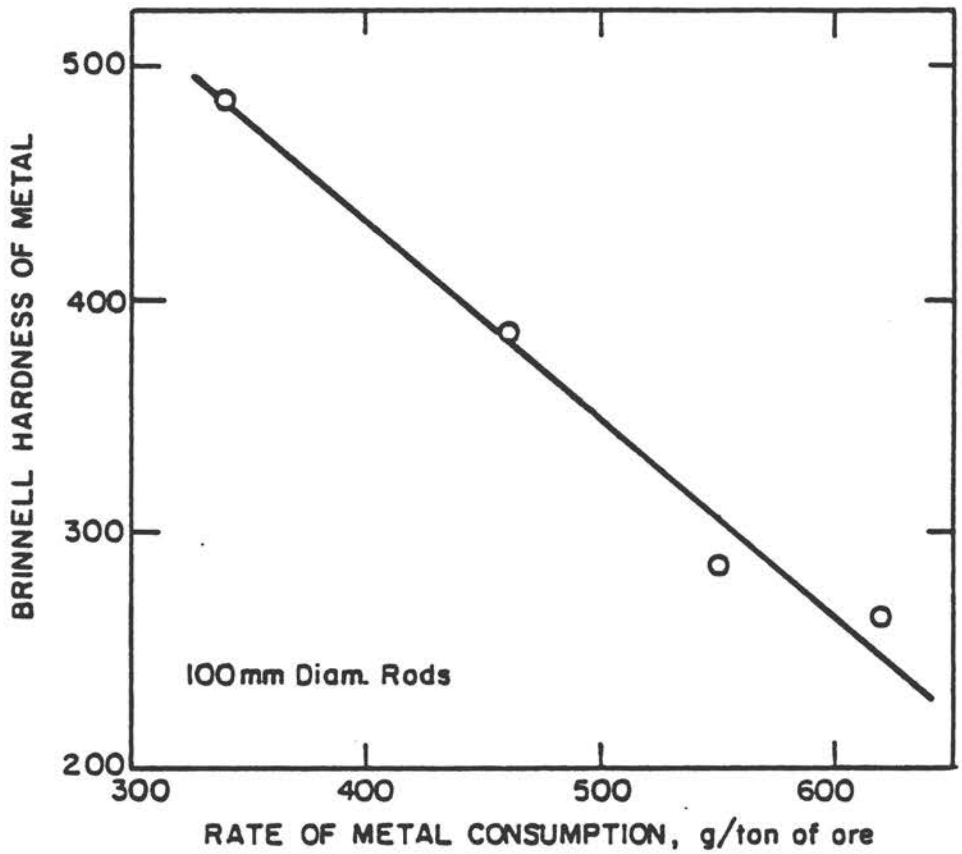


FIGURE 4-25 Rate of metal consumption from 100 mm diameter rods as a function of metal hardness using the data of Nass (1974). The rods were used in wet grinding taconite.

TABLE 4-14: Comparison of Grinding Rod Diameter and Alloy Composition to Rod Scrap Weight for Primary Wet Grinding Taconite

Grinding Rod Diameter in.	Grinding Rod Diameter mm	Grinding Rod Composition	Grinding Rod Scrap Weight (Percent of Original Rod Weight)
4 3/4	120	Hot-rolled AISI 1095 steel (1.2% manganese)	27.0
4 5/3	115	Hot-rolled AISI 1095 steel (1.2% manganese)	28.7
4 1/2	115	Hot-rolled AISI 1095 steel (1.2% manganese)	20.6
4	100	Hot-rolled AISI 1095 steel (1.2% manganese)	17.7
4 3/13	105	Hot-rolled AISI 52100 steel	15.9
4	100	Hot-rolled AISI 52100 steel	12.7
4	100	Heat-treated alloy steel	11-16 estimated

Both types of grinding media (balls and rods) and liners suffer from some metallurgical defects. Foundries and steel mills try always to keep these defects at a minimum. The most serious include the following:

- Shrinkage cavities, sand holes, quench cracks, and alloy element segregation in cast steel
- Shrinkage cavities, cracking, center line pipe, and center bursts in forged grinding media.
- High internal stresses during heat treatment of the alloy steels.

4.5.3 RESEARCH NEEDS

The foregoing discussion indicates that although steel wear in comminution processes in the mineral industry is one of the most important factors in determining process economy, it has not attracted enough attention to reduce metal consumption significantly. Both the fundamental and practical aspects of the topic need intensive research, with cooperation among mineral scientists, metallurgists, and experts in mineral processing, particularly mineral comminution.

The fundamental research should be concerned with the role of each of the basic mechanisms of metal consumption, namely, abrasion and corrosion. There is also a need for standard tests for evaluating wear characteristics to permit percise measurement of the effects of the various variables, such as material properties and operating conditions, on the extent of each of these two mechanisms in metal wear. Steel composition and structure should be among the basic factors to be investigated, together with the effects of heat-treatment variables and techniques.

Investigations should be aimed at establishing the abrasion and corrosion characteristics of various grinding media and the electrochemical interactions between media and minerals. At the same time, it is important to study the effects of abrasion and corrosion products and of additives such as corrosion inhibitors and surfactants on subsequent separation efficiency and recovery of metal values. These studies should be coupled with investigations to develop alloys that will provide reduced consumption of grinding media and maximum metal recovery from the ore, consistent with acceptable production costs. Research on the practical aspects of the problem should include the possibility of applying laboratory findings to field practice, the design of experimental techniques to control the different variables, and the measurement of the effect of the various parameters on the extent of steel consumption in industrial-scale experiments.

4.6 Grinding Aids

4.6.1 INTRODUCTIONS

The unit operation of comminution, which is used extensively in a variety of industries, such as mineral processing, cement, ceramics, chemicals, pharmaceuticals, etc., has long been recognized to be energy-inefficient. Since large tonnages of materials are processed in many of these industries, the high specific energy involved in comminution acquires an even greater significance. In the mineral processing industry alone, the figures are very large. The annual energy consumption at ore concentrating plants in the United States currently exceeds 25 billion kWh, about half of which is attributable to the grinding step, in which it is known that only 1% or less of the energy input goes to the production of new surface area.

With increasing material demands and the obligation to process increasingly larger quantities of ores containing finely disseminated minerals, our limited energy resources and the rising costs of energy present a challenge to the process engineer. One research route that has been explored for about half a century is the development of additives to the grinding mill feed that substantially improve the efficiency of grinding. Such additives are termed grinding aids. This section summarizes and evaluates the results of this effort as well as its implications and points out areas where understanding is lacking and further research is needed.

Some of the distinctive features of the reported results on grinding aids and their proposed mechanism of action must be pointed out in order that the rest of this section be understood in the proper perspective. First, many of the reported results contradict each other. In many cases, the results must be evaluated with caution because of relatively inadequate controls on system variables during experimentation. Incomplete system characterization often makes it very difficult to draw any general conclusions (particularly about the mechanism of the action of grinding aids). Improvement of grinding efficiency by grinding aids appears to be well established, at least on a laboratory scale; the spectrum of hypotheses on their mechanism of action, on the other hand, is rather bewildering. The potential significance of grinding aids in reducing energy consumption in the grinding process is illustrated by the examples in Tables 4-15 and 4-16.

4.6.2 DRY GRINDING SYSTEMS

Considerably more effort has been expended in developing grinding aids for dry grinding systems than for wet grinding systems. Most of this work is oriented toward the cement industry, where huge amounts of cement clinker are dry ground in ball mills. Cement clinker is particularly difficult to grind, and the fine grinding of this material is one of the chief problems of the cement industry.

TABLE 4-15: Examples of the Effect of Grinding Aids in Dry Grinding Systems

Material Ground	Grinding Aid	Effect	Reference
Cement Clinker	Di- or triethanol-amine (0.1%)	22-29% increase in grinding rate	Grachyan and Tavlinova (1973)
Cement Clinker	Propylene Glycol (0.05%)	10% decrease in energy required	Popovic (1971)
Cement Clinker	Organosilicon (0.01-0.05%)	70% reduction in grinding time	Melnik (1969)
Cement Clinker	Glycol	25-50% increase in production rate	Schneider (1969)

TABLE 4-16: Examples of the Effect of Grinding Aids in Wet Grinding Systems

Material Ground	Environment	Grinding Aid	Effect	Reference
Quartzite	Water	Flotigam P (0.03%)	120% Increase in surface area	Szantho (1942)
Alumina	Water	Organosilicon (0.005%)	Grinding time decreased by a factor of four	Kokolev et al. (1968)
Limestone	Water	Flotigam P (0.03%)	70% Increase in surface area	Szantho (1942)
Taconite	Water	XF-42 72 (0.02%)	11% increase in amount -325 mesh	Klimpel and Manfroy (1977)
Taconite	Water	XF-42 72 (0.06%)	18% increase in amount -325 mesh	Klimpel and Manfroy (1977)
Zircon	Water	Triethanol-amine (0.2%)	Grinding time decreased by a factor of four	Orlova, et al. (1977)
Quartz	Water	$AlCl_3$ (0.75 moles/l)	25% increase in relative new surface area	Frangiskos and Smith (1958)
Quartz	Alcohol	---	50% decrease in energy required	Engelhardt (1946)

The first commercial use of grinding aids in the cement industry came about 40 years ago. These grinding aids are usually organic liquids, added in amounts usually not exceeding 0.25% by weight. They are used to increase the product fineness at a given production rate or to increase the production rate at a given product size. In either case, the benefits of any grinding aid must outweigh its cost; furthermore, the grinding aid should have no detrimental effect on downstream processing or the finished product. Given these constraints, the firm industrial acceptance of grinding aids in the cement industry that followed over 10 to 15 years must be considered relatively rapid (Blanks and Kennedy, 1955).

A variety of additives have been shown to have grinding aid characteristics for grinding cement clinker. Typical among these are amines, organosilicones, glycols, resins, cod oil, kojic acid, carbon blacks, wool grease, calcium sulfate, urea, asphaltenes, etc. (Bartell, 1941; Dodson, 1969; Hobday, 1946; Dawley, 1944; Serafin, 1969a; Serafin, 1969b; Swertzer and Craig, 1940; Seebach, 1969; Scherbe, 1970; Evzelman, 1973; Chebukov, 1973; and Dombrowe, 1973.)

Figure 4-26 shows a typical comparison between the grinding of cement clinker with and without a grinding aid. Grachyan and Tavlinova (1973) report that 0.1% of di- or tri-ethanolamine increases the rate of clinker grinding by 22-29%, with glycerol and lower alcohols being less effective. A 25-50% increase in the production rate has been reported with glycol as the grinding aid (Schneider, 1969). The use of 0.05% propylene glycol in pilot plant grinding tests showed that a 10% increase in production accompanied by a 10% decrease in energy costs is possible (Popovic, 1971). The addition of as little as 0.01-0.05% of organosilicones has been found to decrease grinding time by 70% (Melnik, 1969).

The effect of grinding aids on the grinding of materials other than cement clinker has also been reported. These include, among other instances, the use of wool grease in the grinding of gypsum, limestone, and quartz (Hobday, 1946; Voznyuk, 1969); the effect of hydrocarbons on the milling of aluminum powder (Tripathi and Cruszek, 1973); the addition of silicone fluid in the grinding of corundum alumina briquets (Kukolev, 1973); the use of silicones in the ball mill grinding of quartz (Gilbert, 1962); and the use of acetone, carbon tetrachloride, benzene, and nitromethane in the vibratory mill grinding of glass, marble, and quartz (Zegler 1968). In addition, there are many reports in the Russian literature on the beneficial effects of various surfactants of unspecified chemical compositions on the grinding of a variety of materials (Kukolev, 1972; Dzyuba, 1976; Melnik, 1972; Tyagorr, 1971). It is quite clear therefore, that the efficacy of a broad spectrum of chemical additives in improving the efficiency of dry grinding is well demonstrated in the literature.

The results presented thus far indicate the applied nature of the research on grinding aids for dry systems. In other words, these results indicate the potential of a certain additive as a grinding

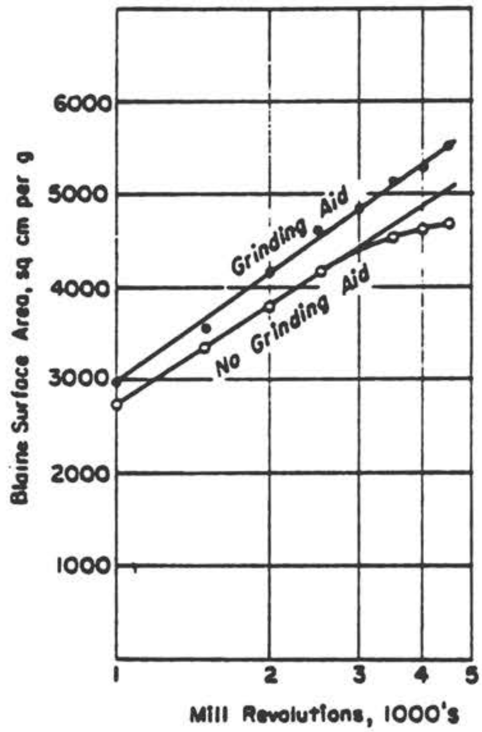


FIGURE 4-26 A typical comparison between grinding of cement clinker with and without a grinding aid in a laboratory mill. (After Mardulier, 1960)

aid for a specific material. However, even though additives have found successful commercial uses, as in the cement industry, the rational development and application of grinding aids requires a clear understanding of their mode of action.

Relative to the effort put into the testing of additives, much less has been expended towards understanding the mechanism of action grinding aids in dry systems. Since comminution involves production of new surface, the energy required to produce a given amount of surface can be reduced by decreasing the solid surface energy. This mechanism was believed intuitively to be responsible for the improved efficiency resulting from additives in rock drilling, according to Rehbinder and Kalinkovaskaya (1932). Rose and Sullivan (1958) extended this theory of adsorption-induced decrease in surface energy to explain the mechanism of grinding aids in tumbling mills.

On the other hand, Westwood and Goldheim (1970) showed that adsorption of surfactants can reduce the strength of materials only when plastic deformation is important in fracture. Since grinding in tumbling mills is typified by brittle fracture, Rehbinder's effect can hardly be responsible for the action of grinding aids in these systems. Furthermore, even with plastically deformable materials, the duration of stress, which must be sufficiently long for adsorbed layers to influence material strength, is too short in a grinding mill, where impact fracture is dominant. Also, the crack propagation velocities in impact breakage in grinding mills are much greater than the spreading velocity (by diffusion into the cracks) of the surfactant molecules (Locher and Seeback, 1972; Somasundaran and Lin, 1972; Graichen and Muhler, 1975; Ocepek and Eberl, 1976). As indicated by the work of Locher and Von Seeback (1972), even vapors of the (liquid) grinding aid adsorbed on the solid surface cannot improve breakage efficiency by decreasing the surface energy of the solid. It is, therefore, very doubtful that adsorption-induced changes in surface energy can be responsible for the action of grinding aids.

According to Westwood (1966, 1968, 1972) and Westwood and Goldheim (1970), the adsorption of molecules of the grinding aid on the solid surface essentially blocks the motion of dislocations near the surface, rendering their motion under stress gradients very difficult. Material plasticity, which is due to dislocation movement, is thus greatly reduced and the solid is rendered brittle. This, these authors claim, is the mechanism of action of surfactants used in drilling. This mechanism applied to tumbling mill grinding is subject to criticism similar to that in the foregoing discussion on Rehbinder's effect. Firstly, Westwood's mechanism has been demonstrated only where plastic flow is important, which is certainly not the case in tumbling mill grinding. Also, the rate of dislocation movement is much lower than crack propagation rates in impact fracture. In fact, it has been demonstrated experimentally that, with single particles, grinding aids have no significant influence on particle fracture by impact (Graichen and Muhler, 1975). Locher and von Seebach (1972) have also given experimental evidence that Westwood's mechanism is probably inoperative. They also conducted abrasion tests, with electron microscopic investigations of the abraded particles. The results indicated that abrasion and re-agglomeration were taking place simultaneously. They found high abrasion rates in organic vapors, which they attribute to the prevention of reagglomeration of the abraded particle because of the adsorption of these vapors on the particles. These results are significant since abrasion is partly responsible for comminution in tumbling mills.

It has been suggested that molecules of the grinding aid might penetrate deep into the tips of preexisting cracks and exert pressure on the crack tips, aiding the fracture process (Lichtman et al., 1958). It is also very doubtful that this mechanism plays a role in impact fracture. Furthermore, as pointed out by Somasundaran and Lin (1972), the mechanism does not explain the effect of long-chain organic surfactants, which would have limited diffusion owing to their large size.

In view of the inability of theories based on adsorption-induced improvement of the breakage subprocess to explain the action of grinding aids, it has been postulated by many researchers that their effect is on the agglomeration and flow characteristic of material in the mill. The idea that grinding aids caused dry dispersion of material was suggested nearly 40 years ago when a considerable increase in dust was noticed in the early commercial uses of grinding aids in cement grinding (Kennedy and Mark, 1938). Observations like these over a period of time suggested that grinding aids coat the cement particles, shielding them from agglomeration forces and preventing the welding of particles back together.

It has been argued that grinding aids readily satisfy the valence forces produced by material fracture (Ocepek and Eberl, 1976; Mardulier, 1961; Ghiki and Rabottino, 1967; Beke and Opoczky, 1967). Since grinding aids usually are polar substances, they are preferentially adsorbed on specific sites where breakage of electrovalent or covalent bonds results in residual electrical forces. By the same token, at least part of the improved efficiency in wet grinding over dry grinding may be attributed to the polar nature of water. Experience has shown that polar grinding aids are more effective than nonpolar aids (Mardulier, 1961). In some cases, the adsorption of grinding aids on the particles might create similarly charged particles, and, therefore, the electrostatic repulsion between the particles could be responsible for their dispersion.

There is considerable evidence and concurrence of opinion on the prevention-of-agglomeration mechanism of grinding aid action in dry systems (Locher and Von Seeback, 1972; Graichen and Muhler, 1975; Ocepek and Eberl, 1976; Mardulier, 1961; Ghigi and Rabottino, 1967; Beke and Opoczky, 1967; Scheibe et al., 1974; Opoczky, 1975). It is believed, therefore, that grinding aids that chemisorb on solid particles would most effectively reduce adhesive tendencies between particles and thereby improve grinding efficiency. Since the adhesive forces are surface-area dependent, the relative improvement in grinding efficiency is greater, the finer the product size (Locher and Von Seeback, 1972; Graichen and Muhler, 1975). Thus, the economic application of grinding aids implies their injection into the mill wherever adhesive forces are obstructive, so that they may be used advantageously even for relatively coarser grinds if the influence of adhesive forces can be confirmed in this case (Locher and Von Seebach, 1972).

At very fine grinds, the tendency of ground particles to coat the grinding media and mill liner becomes very pronounced and has long been considered to limit the effectiveness of the grinding process. There has been some diversity in opinion as to whether grinding aids are effective only in reducing ball and liner coating, but, as Mardulier (1961) points out, this can be considered a special and extreme case of agglomeration. Since media and liner coating occurs only with very fine grinding, it is possible that in certain situations the use of grinding aids is justified only when coating is present initially.

4.6.3 WET GRINDING SYSTEMS

As pointed out earlier, water itself can be considered a grinding aid because of the improved efficiency and mill capacity obtained in wet grinding relative to dry grinding. Figure 4-27 shows the data of Coghill and DeVaney (1937), where it is clear that the material has better grindability over the entire range of mill holdups in wet grinding compared to dry grinding. A variety of reasons have been put forth for this phenomenon.

As discussed earlier, the effect of water is believed by some to be due to its polar nature and consequent ability to satisfy surface residual electrical forces created upon fracture and thus reduce adhesive tendencies between the ground particles (Mardulier, 1961). The cushioning effect of fine particles in wet grinding will be minimal because they become suspended in the water. This would lead to increased grinding action on coarser particles and improved grinding efficiency.

Meloy and Crabtree (1967) proposed that the surface tension of the liquid enables particles to cling to balls and move into the zone of maximum grinding action. Their results (Figure 4-28) indicate that particles are ground more rapidly in liquids of high surface tension. They also found that larger particles are ground more rapidly in a more viscous liquid, while smaller particles show the opposite effects (Figure 4-29). However, with highly viscous liquids such as sorghum, the grinding rate of coarse particles decreased. This they attributed to two competing phenomena: at higher viscosity, the coarser particles tend to be lifted into the grinding zone, while the cushioning effect is enhanced. This implies the existence of an optimum liquid viscosity for grinding. Similar results have been reported in the literature (Schweyer, 1942). Although grinding in organic liquids is economically prohibitive in most cases, these studies imply that to obtain a narrow size distribution in the product, a high viscosity

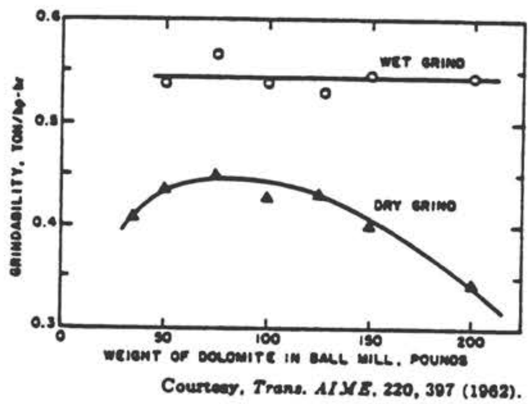


FIGURE 4-27 Comparison between grindability of dolomite when ground under wet and dry conditions as a function of material holdup in the mill. (After Coghill and DeVaney, 1937)

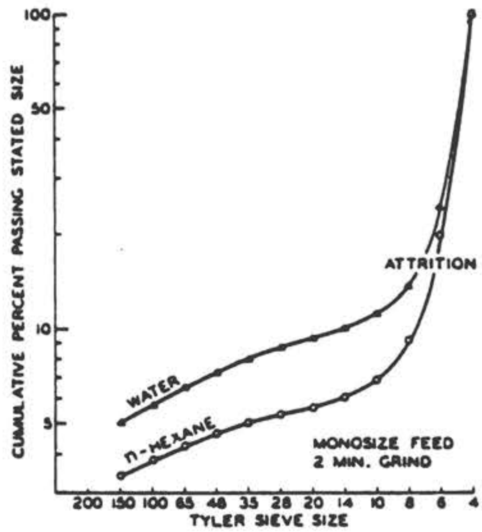


FIGURE 4-28 Effect of liquid surface tension on the grinding of quartz in water and n-hexane. Water and n-hexane have approximately the same visocity and density, but their surface tensions are 72 dynes/cm and 18 dynes/cm, respectively. (After Meloy and Crabtree, 1967)

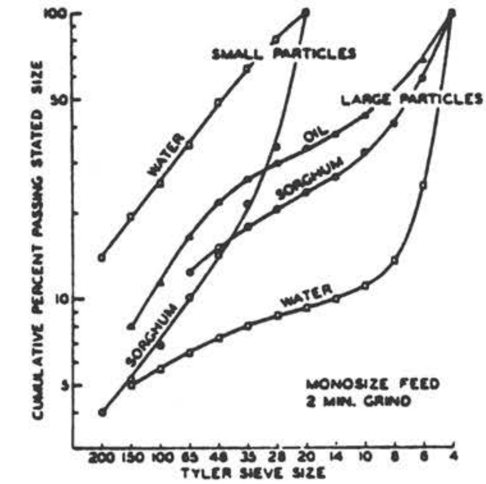


FIGURE 4-29 Effect of liquid viscosity on the grinding of small (20-mesh) and large (4-mesh) particles in various liquids. Water has the lowest viscosity (1 cp), whereas oil and sorghum have viscosities of 2000 and 10,000 cp, respectively. (After Meloy and Crabtree, 1967)

liquid can be used as the grinding environment if the material to be ground has a high unit value. Kapur et al. (1965) showed that the specific gravity of the material with respect to the environment is important; if the material is less dense than the environment (the liquid in which is is ground), it tends to float in the mill, which reduces comminution.

Although organic liquids tend to be economically impractical, it must be mentioned that many of them have been shown on a laboratory scale to provide more efficient grinding environments than water. Englehardt (1946a, 1946b) demonstrated that the energy consumption per unit area of surface created in the grinding of quartz in water was reduced by nearly half when the grinding was done in alcohol. The same work failed to indicate any correlation between the abrasion hardness of quartz and the dielectric constant of the liquid. Kiesskalt's (1949) results indicate a 1200% increase in surface area of material ground in isoamyl alcohol over material ground in water. The data of Lin and Mitzmager (1970) are rather interesting in that grinding in carbon tetrachloride and methylcyclohexane actually decreased the rate of fine production somewhat compared to the corresponding rate in water. However, this decrease was not observed if only a small amount of water was dissolved in the organics.

As with grinding aids in dry grinding systems, a fairly large number of investigations on the effectiveness of chemical additives for wet grinding are reported in the literature. Most of these investigations are on a laboratory scale.

Szantho (1942) demonstrated that Flotigam P in concentrations of up to 0.03% produced a 100% increase in surface area of quartzite and limestone. As can be seen in Figure 4-30 the effectiveness of the grinding aid increases with increasing concentration to a maximum beyond which further increases in concentration decrease the specific area of the products. In one case, the addition of flotation reagents to a wet rod mill improved the production of fine particle only in a certain size range (Malati et al, 1968).

More recently, Klimpel and coworkers have reported the results of extensive laboratory and industrial scale tests on the development of a grinding aid, XFS-4272 (Klimpel and Manfroy, 1977a, 1977b, 1977c; Klimpel and Manfroy, 1978; Katzer et al., 1979; Klimpel, 1980). Figure 4-31 shows a typical result for the grinding of copper ore in an industrial rod mill with and without the grinding aid.

Other examples of the beneficial effects of grinding aids in wet grinding include Khodakov and Rehbinder (1961) on the ball milling of quartz; Pwen and Naloichenko (1968) on the grinding of ultraporcelain and talc with 0.05-0.1% polysiloxane; Kokolev et al. (1968a, 1968b) on the grinding of alumina in the presence of about 0.005% organosilicones; and Orlava et al. (1977) on the grinding of zircon and other minerals. In some cases (Gilbert and Hughes, 1962; Szantho, 1942; Hartley et al, 1978), the addition of organic liquids to a wet grinding system has been reported to decrease grinding efficiency.

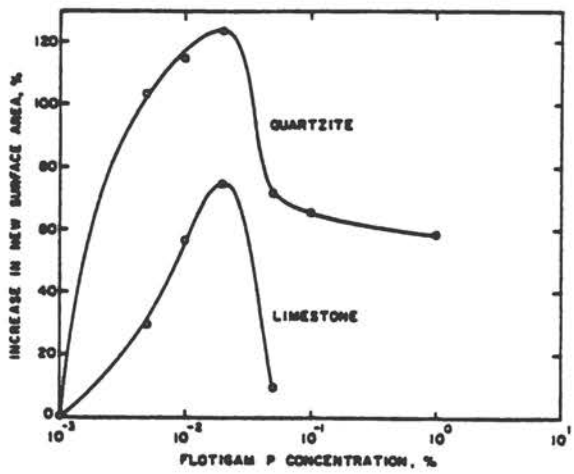


FIGURE 4-30 Effect of a commercial flotation reagent (Flotigam P) on the grinding of quartzite and limestone in a rod mill. In both cases, an optimum Flotigam P concentration of about 0.03% results in the maximum increase in product surface area. (After Szantho, 1942)

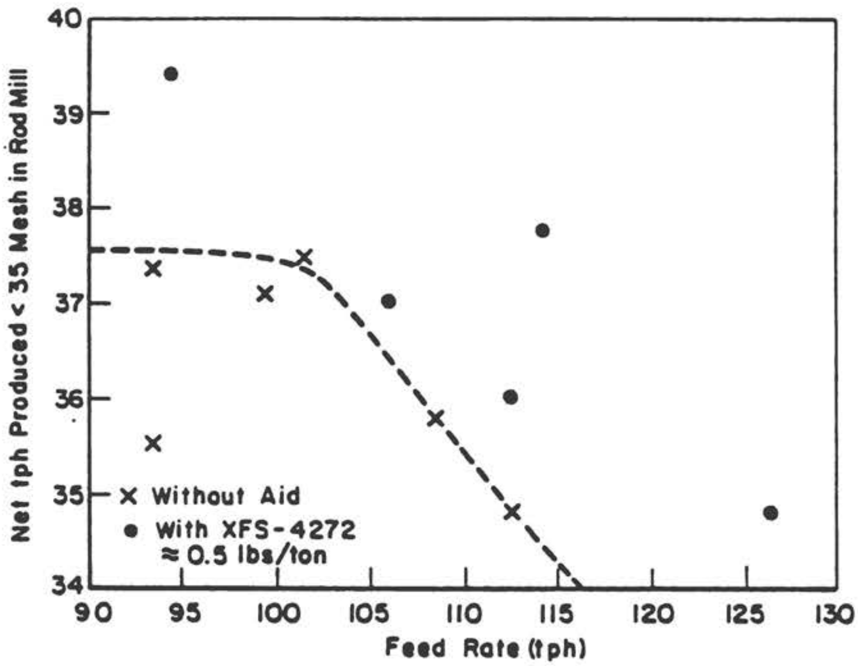


FIGURE 4-31 Effect of Dow Chemical XFS-4272 grinding aid on the production rate of minus 35-mesh material in the grinding of a copper ore in an industrial open-circuit continuous rod mill as a function of the solids feed rate to the mill. In all cases, the water feed rate to the mill was kept constant. (After Klimpel, 1980)

There has also been considerable study of the effects of inorganic electrolytes on grinding performance (Byalkovskii and Kudimov, 1943; Brown, 1955; Beyer, 1964; Stanzyk and Feld, 1968; Mallikarjunan et al, 1965; Hanna and El-Gamal, 1977). Although there are some ambiguities, it might be generally concluded that some improvement in grinding results from the addition of an optimum electrolyte concentration and that dispersing agents improve the comminution of materials.

It is apparent from the foregoing presentation that there is considerable laboratory-scale and occasional pilot-plant-scale evidence of the advantage of using chemical additives in wet grinding systems. But once again, as in dry grinding, the mechanism of action of these additives has not been rigorously studied. Where such studies have been made, the arguments put forward have often been quite controversial and ambiguous. Basically, much of our previous discussion on the mechanism of grinding aids in dry systems is valid here. In other words, it is very doubtful that grinding aids in a wet system affect the breakage characteristics of material under the predominant impact fracture conditions in tumbling mills. In broad terms, therefore, their influence may be on the transport characteristics of these systems and attendant phenomena.

The effect of the physical properties of the environment on grinding kinetics is quite significant. Therefore, the influence of grinding aids on the properties of the environment must be examined thoroughly. In this context, slurry viscosity, liquid specific gravity, etc., are all very important.

In a series of papers Klimpel and Manfroy (1977a, 1977b, 1977c) report experiments in a laboratory ball mill under controlled grinding conditions at various slurry densities without grinding aids. Their results (Figure 4-32) confirm the well known fact that there is an optimum slurry density for grinding (see also Hanna and El-Gamal, 1977). Klimpel and Manfroy (1978) contend that the mechanism of action of the grinding aid in their system is that it permits slurries with a higher solid-to-liquid ratio than the optimum without aids to interact with the grinding media while maintaining the fluidity characteristics of less dense slurries. In other words, the grinding aids control the solids dispersion in the mill and act as viscosity modifiers.

The approach that Klimpel and coworkers have employed in the development of grinding aids, such as XFS-4272, at Dow Chemical Company is based on possible advantages of controlling the slurry dispersion at high solid-to-liquid ratios without the dispersion effect achieved by adding water alone. This is clearly demonstrated in Figure 4-31, which shows a typical result obtained in an industrial rod mill, with and without the grinding aid at constant feed water rate and varying feed rates of solids.

From their extensive test work, Klimpel and coworkers conclude that significant advantages can be realized by using grinding aids if (1) the mill is operated in a region of percentage solids high enough so that a further increase produces a large slurry viscosity increase

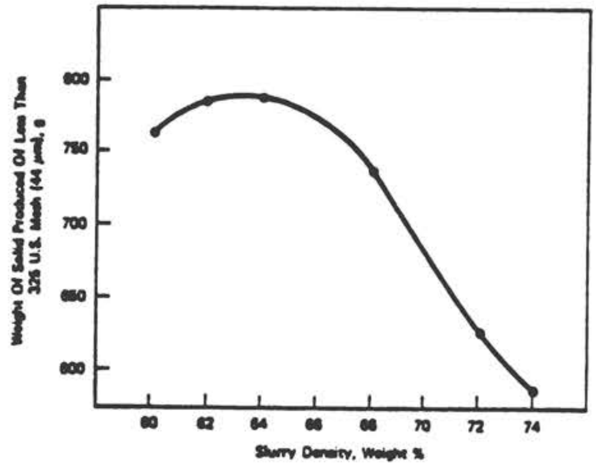


FIGURE 4-32 Effect of slurry density on the weight of minus 325-mesh material produced in the grinding of a copper ore for 40 minutes in a 20.3 cm laboratory batch ball mill. (After Klimpel and Manfroy, 1978)

in the absence of the grinding aid, (2) the solids in the slurry have sufficient adsorption capacity for the grinding aid so that it can improve the slurry dispersion characteristics, and (3) the grinding aid has consistently good dispersion characteristics over the range of physicochemical conditions (such as slurry pH, intensity of mixing, etc.) encountered in the practical operation of the mill.

An important aspect of this work is the quantification of the effect of grinding aids through the parameters of kinetic models of grinding (i.e., the breakage rate and the breakage distribution function). Of course, any grinding aid must have no adverse effect on downstream processing. Many potential grinding aids of the viscosity modifier type break down under high shear conditions, such as those prevailing in industrial grinding mills. It is important to recognize this when using bench-scale tests for evaluating the potential of a dispersant as a grinding aid. Apparently, most commonly used dispersants do not meet the conditions just mentioned and have no applications as grinding aids. Some, such as sodium silicate in taconite ball milling, may even be detrimental to grinding (Figure 4-33).

4.6.4 AIDS FOR CEMENT GRINDING IN THE USSR

Ur'ev (1979) has reported that considerable research on grinding in the Soviet Union is centered on the pioneering work of Rehbinder on chemical additives (Polferov and Kotelnikov, 1975).

The most widely used method is based on application of hydrophilic surface-active substances, primarily based on calcium lignosulfonates. The addition of surface-active substances during the comminution of cement clinker allows production of highly effective plasticized cements. By an analogous method, it is possible to obtain hydrophobe cements by introducing hydrophobic-type additives (acidol-naphthenate soap, sodium abietate, and other types of additives) during comminution. These additives comprise about 0.1% of the mass of the pulverized cement. In recent years, various types of silicoorganic compounds have begun to be used, although so far in limited amounts.

The Soviets also have considerable experience in using grinding intensifiers in the form of solid dispersed particles, chiefly quartz sand. By comminuting quartz and cement together it is possible to produce high-activity sand cements. Work in the USSR on the fine comminution of cement and grinding intensifiers is concentrated in the Scientific Research Institute for Cement of the Ministry of Building Materials Industry of the USSR. Significant Eastern-Bloc research in intensification of the processes of fine comminution of cement is being done by Dr. B. Hoffman (GDR, Forschungsinstitut für Aufbereitung der AW DDR, Freiberg). This work is devoted to a detailed study of the effectiveness of comminution of cement in relation to its chemical nature, structure, and class of surface-active substance.

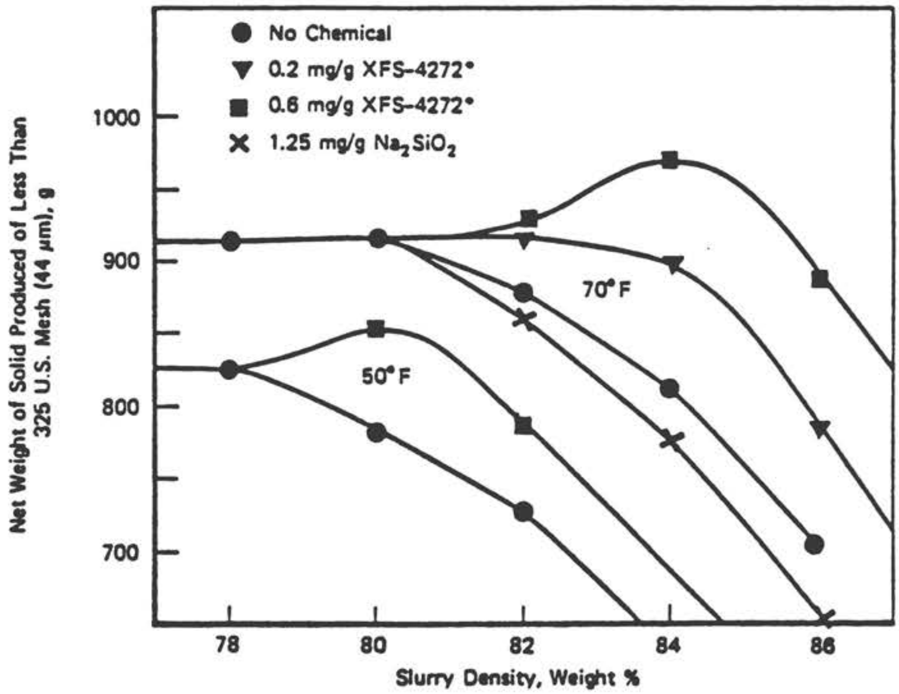


FIGURE 4-33 Effect of Dow Chemical XFS-4272 grinding aid and sodium silicate on the grinding of a taconite ore in a laboratory batch ball mill as a function of slurry density at 50° and 70°F. (After Klimpel, 1980)

4.6.5 RESEARCH NEEDS

It is apparent that although a considerable effort has been put into examining the potential and observed benefits of grinding aids in both dry and wet systems, there is little understanding of the mechanisms involved. Only in recent years have test results been evaluated systematically in terms of basic grinding parameters (Klimpel and Manfroy, 1977a, 1977b, 1977c) or has the need to identify the relevant fundamental properties of both solids and liquids been established (Somasundaran and Lin, 1972). It cannot be denied that benefits can be gained by the use of grinding aids, particularly in dry grinding systems; however, the benefits appear to arise more from modification of flow and agglomeration than from any effect on the inherent mineral strength and fracture mechanism.

To explore the commercial potential of grinding aids, research is needed on the basic mechanisms of grinding processes and the way in which grinding aids can alter these mechanisms to improve grinding efficiency.

More precise evaluation of available grinding aids in terms of basic grinding parameters and fundamental solid and liquid properties is needed, both to quantify current benefits and to lead to better understanding of the underlying mechanisms so that improved aids can be designed.

4.7 Novel Techniques

In this section, novel methods of comminution are briefly reviewed. The real potential of novel methods of comminution requires careful independent assessment, but they are discussed here for historical purposes and for completeness of the report.

4.7.1 MECHANICAL METHODS

Most conventional methods of crushing and grinding are mechanical, e.g., rod and ball mills. Newer mechanical methods are emerging. One that has seen much increased use in recent years is the roller mill. This has been due largely to scale-up in the design of roller mills so that large tonnages now can be effectively comminuted. The roller mill is used to grind coal, phosphate ores, cement raw mixes, and similar materials which are not too abrasive or hard. Recently, attempts have been made to use the roller mill to grind cement clinker, and one such plant is now operating.

The mechanical power required by the roller mill usually is lower than the power required to perform the same task in conventional dry tumbling mill circuits, possibly by as much as 50% (Allis-Chalmers). On the other hand, a considerable amount of pneumatic energy is required in a roller mill to furnish an air sweep which conveys fine particles out of the mill. Tumbling mills do not require nearly so great an air sweep or may require none. Thus the total energy per ton attributable to the roller mill may be as low as 60% to as high as nearly 90% (for very hard material) of conventional circuit energy for grinding to about 80% - 200 mesh.

Roller mills have several other advantages. They are very quiet compared to a tumbling mill. More important, if the air sweep consists of hot gases, much drying of the feed material can be accomplished within the mill. Whereas dry tumbling mills ordinarily are limited to feeds with less than 0.5% moisture, the roller mill often can process material with moisture as high as 18% and thus can efficiently use hot waste gases from other plant operations, i.e., a rotary kiln. The roller mill is seen, then, as performing four functions: it crushes (accepts 100 mm feed); it grinds (to -200 mesh); it dries (handles up to 18% moisture); and it classifies (internally).

A very recent study by Schonert (1979) suggests that cement clinker grinding energy can be reduced to 33% to 50% of present levels. His bench scale experiments consist of very highly stressing a bed of clinker in a press followed by tumbling in a conventional mill. Both the press and the mill are operated in a closed circuit with a screen. Although the pressures attained in a 20 mm bed of clinker ranged from 50 to 300 MPa (7250 to 43,500 psi) and required about 3 to 7 kWh/tonne for this step, the total energy requirement was fairly constant at about 10 kWh/tonne for a clinker with a Zeisel grindability of 32 kWh/metric ton.

To make this process practical, Schonert is investigating the use of rolls as a means of stressing the clinker. Unlike compaction processes, it is not necessary that the clinker be formed into a strong flake by the rolls. Thus very little dwell time is required, and the rolls can be operated at fairly high speed with correspondingly high throughput. The discharged specimens, having undergone high stresses and been flattened into agglomerates, would then be fed to a much smaller-than-normal conventional ball mill for the rapid disintegration of the stressed particles into the size distributions of normal finished cement. A recycle ratio of 3.5 to 4.0 is envisioned for this process.

Since crushers generally require a low amount of energy per ton crushed, their design has not changed rapidly. A suggested evolution of a jaw crusher for restricted head room is being developed by the U.S. Department of the Interior (1978). It is likely that crushers will be built to handle still larger feed sizes and also to become more widely used in crushing to finer sizes.

The acceleration of gravity governs the force that a falling ball can exert within a ball mill. To obtain greater force, higher acceleration must be attained. By designing a system such that the mill shell is in a circular orbit, one can achieve this increased acceleration.

A variation of this design, which has been reported to have moderate success grinding -6 mesh gold ore to -200 mesh in South Africa, has received some notice (Bradley et al., 1975). Here, the grinding media is in a cylinder driven by gears so that it is also in an orbit as it rotates. Two or three such mill bodies are coupled together for balance.

Attrition mills come in several variations. The stirred ball mill consists of a vertical cylinder with an axial rotating stirrer (Herbst and Sepulveda, 1978; Davis et al., 1980). The stirrer has bars affixed that serve as paddles to move small balls or shot around the mill. Feed particles are comminuted by the media as stirring takes place at fairly high speeds. Thus, the throughput does not have to be insignificant, and there is potential for scale-up. Coal can be wet ground to 50% -5 microns at 200 kWh/ton, which is lower specific energy than in conventional ball mills. Other attrition mills, such as the sand mills, have disc shaped impellers and use sand or glass beads as the grinding media (Morehouse-Cowles, 1965).

The Szego mill (Trass, 1980) consists of a stationary vertical cylinder with several rollers made to roll on its inside wall. Particles are crushed as the rollers press on them. The roller can be a helical grooved rotor mounted on a flexible wire rope. Feed rates for prototype mills range from 500 to 3000 kg/hr. No claim is made for significant reduction in energy consumption, but less floor space is required than for a conventional mill.

Vibrating ball mills are being built by several manufacturers. They all impart motion to the grinding media by first causing the nonrotating mill shell to vibrate. This vibrating action may be circular, oval, or straight line. A circle of vibration up to 1 in. in diameter can be induced in some of this equipment. Media can be standard steel balls or rods or ceramic balls, appropriately sized, or other shapes. The bearings of these mills do have scale-up limits so that maximum throughput usually is considerably less than 10 tons per hour. The energy requirement per ton in a vibrating mill is of the same order of magnitude as in conventional tumbling mills. In some cases, especially on harder material, the vibrating mill may be more efficient. Although these mills are noisy, their compact size is often an advantage over conventional ball mills. An application not seen in this country is the reported use of vibrating mills in the USSR for wet grinding cement clinker (sometimes coground with sand) at the site of use of the cement mix (Ur'ev, 1979).

Jet or fluid energy mills are built by several manufacturers. Some jet mills use air or other gas to accelerate particles into paths where much mutual abrasion and collision occur thereby hastening comminution. Many of these jet mills are fairly small, with throughputs measured in pounds, but some handle hundreds or thousands of pounds per hour. The grinding energy per ton in these mills is also fairly high, although research to reduce it is underway. The mills are effective for special purposes, usually at finer sizes than in conventional tumbling mills.

Operation of a mill in a vacuum offers the potential of eliminating windage. Planiol (1962) has constructed several impact type mills designed for such operations. However, other problems arise such as heat dissipation, the evaporation of moisture or release of other gas and the cost of its subsequent removal, and the efficient operation of air locks.

Investigations are being made by Cohen and coworkers at Imperial College, London, of grinding at elevated temperatures and pressures. A batch rod mill has been operated near 400-450°C. Smaller vibrating ball mills also have been used in these studies. The evaporation of water in the mills raises the pressure to high levels. Some mechanical-chemical effects have been noted, and the breakage pattern appears to be more intergranular than for ordinary grinding.

4.7.2 NOVEL METHODS OF CEMENT GRINDING IN THE USSR

A new wet process method for cement has proven quite effective in the USSR. The wet grinding of cement raw materials permits a substantial (up to 10%) reduction in energy and at the same time increases the mix activity. However, when this aqueous slurry enters the kiln, energy must be expended to evaporate the water. It is therefore of interest to reduce the water content to a minimum for the transport of the slurry rather than convert to a dry system.

Following the technique of vibrating wet raw meal to eliminate air pockets, the Soviets apply similar vibration to the slurry in the transport troughs. The vibrational frequency is in the range of 50-200 Hz. With modest amounts of vibrational energy they claim a large reduction in viscosity, thereby permitting much less water to be used (Ur'ev and Taleisnik, 1976). Also in addition to the vibratory energy, with the use of additive oxyethyl-akylphenol, the drop in viscosity is more rapid, and less energy is required. This method is now in use in cement plants in the USSR and Eastern Bloc countries. It has been described, with tables and graphs, by Ur'ev and Taleisnik (1976).

4.7.3 ULTRASONIC COMMINUTION

Preliminary laboratory evaluations of ultrasonic coal grinding under simulated roller mill conditions have been made by the Energy and Minerals Research Company (EMRC).

Static stress was applied at a level sufficient only to achieve contact between the nip and the coal mass. There was no size reduction of the 1/16 in. feed until ultrasonic vibration was applied; 2 to 7 micron ultrafines were then produced. The increasing needs for plant-grind (75 micron) and clean ultrafine coal (< 10 micron) have led to a current EMRC project on ultrasonic fine grinding for the Department of Energy (Tarpley et al., 1980).

In analogous work, these investigators had achieved size reduction of a number of solid particles in liquid systems (Tarpley et al., 1955, 1958, 1959). In precipitating thorium oxalate from an aqueous solution, equiaxed particles were fractured along fissures and inclusions. They have also produced, incident to dry boring of dense artificial graphite, swarf of substantially reduced particle size with increased uniformity (Maropis, 1964).

The material tested was a batch of Peerless seam (West Virginia) coal of -10 +14 mesh (or approximately 1/16-in, 1500 microns). As determined by scanning electron microscopy, the 1500-micron particles were reduced to the ultrafine 2 to 7 micron range. At the very low static stress applied, the control (nonultrasonic runs) showed no size reduction, indicating that comminution was entirely a result of ultrasonic mechanisms. The ultrasonic comminution increases the prodution of -200 mesh particles, with signficiant increases in the production of -325 mesh (-44 micron) particles and substantial fractions in the -20 micron and -10 micron ranges.

Ultrasonics provide particle fracture mechanisms at sites of weakness within the particle. These mechanisms, demonstrated or inferred in similar applications, include at least six size effects: (1) preferential energy absorption at stress concentrations, (2) fatigue promotion of crack growth, (3) reduction in shielding of larger particles by smaller ones, (4) micropumping of fluid into small cracks, (5) high local stresses produced by cavitation, and (6) preferential shear comminution of large particles.

A working group of the British Institution of Chemical Engineers, concerned with theory and practice of size reduction, has recently recommended extension of process development in acoustic, fatigue failure, and related novel fracturing methods (Marshall, 1975). Others (Lippman et al., 1977) also have reported success in the ultrasonic comminution of coal.

Ultrasonic comminution may prove to be very useful by making ultrafine coal available at reasonable cost for coal/oil mixtures, diesel and turbine fuels, and other applications. Because of the unique mechanisms at work in ultrasonic fracture (Cain et al., 1975), it may also prove to be useful in comminution for liberation where the particles to be liberated have definite grain boundaries with the other material in the ore.

4.7.4 PRESSURE VARIATION METHODS

The sudden release of pressure causes solids as well as gases to expand, but to a much lesser extent. If rock is confined in a cavity under some initial pressure, and the pressure suddenly is relieved, there may be some expansion and possibly some fracture. This approach was studied in the early 1930's by USBM personnel (Taggart, 1945). The process was inherently a batch operation that could be cycled at slow rates only. More recently this approach was investigated

(Snyder, 1966) under the name Snyder Process. The initial flurry of publicity on the process has since completely died out (Cavanaugh, 1972).

The electrohydraulic effect (originally investigated by Yutkin, 1955) is also a pressure phenomenon. It results from the transformation into a hydraulic pressure rise of the electrical energy discharged by a capacitor. If a rock specimen is in the liquid undergoing such a transient pressure pulse, the pulse may cause stress concentration at cracks or crevices. If the pulse is repeated often enough the rock may fail (Andres, 1977). The efficiency of the electrohydraulic process is also low (Bergstrom, 1961). It has never emerged from the laboratory as a commercial tool, although it is still being studied at BRGM in France and Imperial College, London.

4.7.5 CRYOPULVERIZING

Many materials such as some elastomers, resins, oils, and some pharmaceuticals, cannot be ground at room temperature because they are elastic, soft, or gummy. In addition, there are materials which, while they may be ground at room temperature, can be ground more efficiently when they are embrittled at low temperature. Examples are plastics, elastomers, foods, some minerals, and spices. There is currently high interest in cryogrinding because of the obvious advantages for such materials. A particular interest is the pulverization of plastics for solventless and powder-coating applications.

Liquid nitrogen is an effective and nontoxic coolant. Its specific gravity is 0.8, and its temperature (at the boiling point) is -320°F. When brought into contact with a warm object it vaporizes by absorbing 85 Btu/lb. As the cold vapor warms to room temperature it absorbs an additional 80 Btu/lb. This absorbed heat can be used for precooling, so it is not wasted. The optimum temperature for pulverizing many materials often is higher than -320°F, so it is inefficient to cool to that temperature. By means of insulating jackets through which the cold nitrogen vapor can be circulated, the batch material may be kept at any desired temperature.

Impact hammer mills seem to be best suited, at present, for cryopulverization, for they utlize the embrittled nature of the cold material by fracturing it rather than by shearing or erosion as in other types of mills. However, investigations are yet to be made of all of the possible applicatons.

Cryopulverization of hard-to-grind elastic and plastic materials requires relatively low grinding energy, and in some cases the cost of energy saved exceeds the cost of the liquid nitrogen required. Examples of liquid nitrogen consumption with the Linde-designed apparatus are given by Wary and Davis (1976).

Research on improvement of mineral separation at low temperature is being conducted at the Institute of Mining Chemical Ores in Moscow. They are also using a hammer mill on material cooled to liquid nitrogen temperature. The various minerals in composite ores undergo different degrees of embrittlement at low temperature, and differential fracturing may occur. It was reported that iron ores and phosphorides in particular become quite brittle. They pulverize more readily, therefore, and can be separated by a suitable classifier. Another reported example is sulfur in calcium carbonate. The sulfur becomes brittle and can be separated by a suitable classifier. This research, while promising, has not yet been examined for cost effectiveness.

Cryocomminution has still another advantage, in the standardization of preparation of laboratory specimens. In the particular case of coal, if it is ground at room temperature in the presence of air, the localized high temperature produced causes both surface oxidation and production of gases (Solomon and Mains, 1977).

4.7.6 THERMAL TECHNIQUES

It is well documented that low-density thermal sources can be used to weaken or fracture rocks (Fogelson, 1974; Lauriello and Chen, 1973). An example of a tensile stress-strain curve of charcoal granite at ambient and 500°C is shown in Figure 4-34.

The fragmentation potential of a rock resulting from a heat inclusion is defined by Thirumalai (1979) as the ratio of the thermal stress induced at a given temperature to that at a high temperature. The thermal stress is given as $E\,\alpha\,T/(1-\gamma)$ where E is the elastic modulus, α the thermal expansion coefficient, T the temperature, and γ is Poisson's ratio. Since rock is thermoplastic, with decreasing E and γ with increasing temperature, the ratio, or fracture potential, goes through a maximum. The maximum is rather sharp at 550°C for Dresser basalt, but has a broad plateau in the range of 300-500°C for charcoal granite. This is a reasonable explanation for the experimental data of Figure 4-34. Rock is full of small faults and has many grain boundaries. The stresses from heating begin to propagate the cracks, which makes the rock more friable. The difficulty in exploiting this well-known phenomenon has been both energy efficiency and speed of application.

Not all rock is amenable to rapid heating because of low thermal diffusivity. Flame jets are sometimes used for hole drilling, but in some materials spalling occurs. Heat, therefore, does not penetrate the rock but remains in the spalled chips. In principle, however, spalling may achieve acceptable efficiency where hole drilling or rock cutting is involved. A flame jet with an abrasive material in the jet has been found to work very well in rock cutting (Browning et al., 1965). Heating from the surface inward has been tried with unfocused laser radiation (Lauriello and Chen, 1973; McGarry and Moavenzadeh, 1971; Williamson et al., 1968) and with ion beams (Schumacher and Taylor, 1967). Each of these techniques has drawbacks.

Instead of trying to heat from the surface inward, attempts have been made to heat internally. Direct current and low frequency alternating current have been used (Clark and Lehnhoff, 1974).

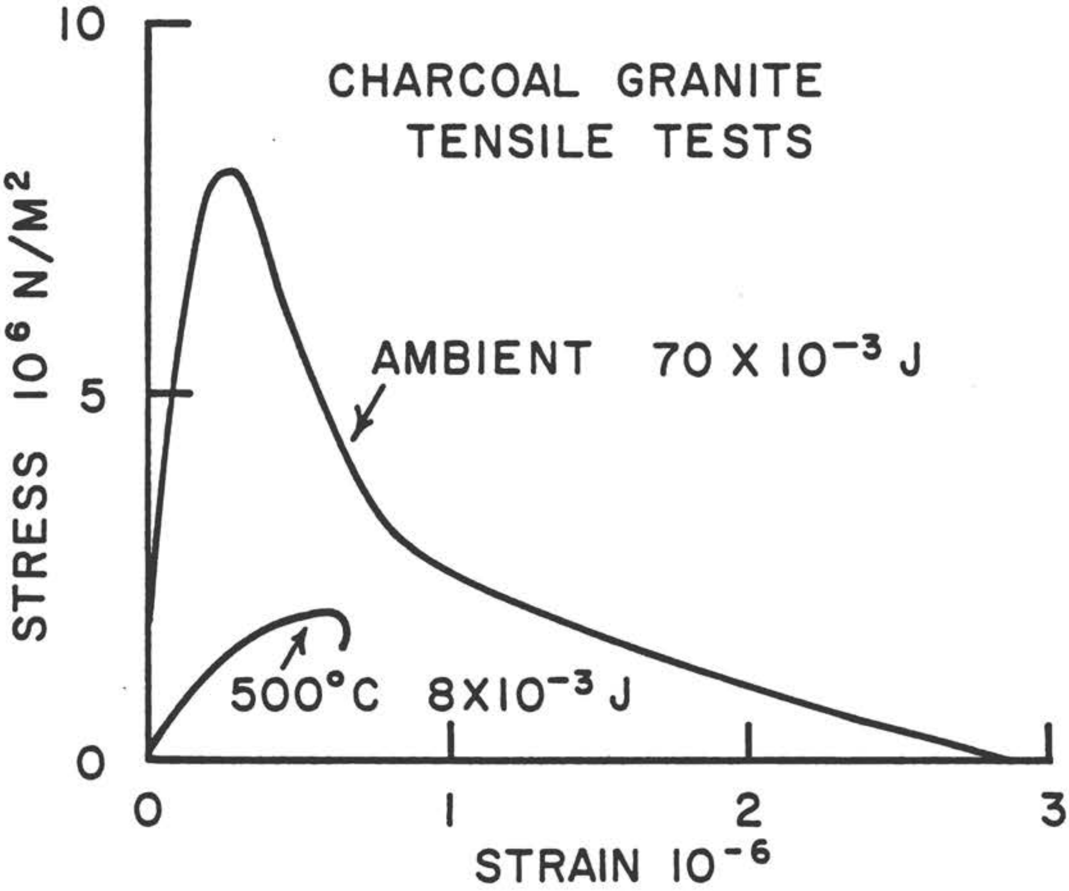


FIGURE 4-34 Stress-strain curves for heated and unheated charcoal granite. (Thirumalai and Cheung, 1974)

However, the conductivity of rock is affected by many variables, such as mineral constituents, structure, moisture content, etc. Therefore, equipment must be versatile. Sarapuu (1969) showed that blocks of magnetic iron ores could be fragmented with an energy consumption of 1 to 2 kWh/ton. Thirumalai and Cheung (1974) showed that high frequency, 100 MHz, could create thermal inclusions and fragment blocks of granite and basalt with an electrical energy consumption of 1.2 to 2.2 kWh/ton. At present the drawback to these methods is that holes should be drilled for the electrodes for optimum placement.

The need for holes can be eliminated by microwave induction heating. Powers of 1 MW can be had in single tube devices with frequencies in the range of 1000-6000 MHz. The feasibility of fracturing hard rocks and concrete with microwaves has been demonstrated by Seldenroth et al. (1965), and Liessmann (1966) reported that granite rocks were fragmented at energy consumption rates of about 2.2 kWh/ton. This concept is being evaluated for selective liberation in ores (Pemsler, 1979).

4.7.7 THERMOMECHANICAL METHODS

Rocks that have been heated can be more readily broken (Figure 4-34). Considerable research has been done on the combination of heating and mechanical excavation. A prototype continuous mining system that uses infrared heaters and an impactor plow has been developed to mine copper which occurs in sandstone, siltstone, and shale (Hanninen and Sipola, 1972). The prototype is currently undergoing field tests.

Lasers have been used for kerfing to enhance the rate of advance of a tunnel drill (Jurewicz et al., 1974). Figure 4-35 shows the reduction in strength of granite vs. input energy from lasers of various powers. There is a striking decrease in strength, starting around 3×10^3 J/cm^2. The power level of the laser is not very important, as the grouping of curves shows. The weakness of the rock leads to a dramatic decrease in the cost of cutting (Figure 4-36). Extrapolation of these data shows the potential improvement in hard rock penetration rate with increasing laser power (Figure 4-37).

The local heating produced by a laser will melt and vaporize rock (Carstens and Brown, 1971; Carstens, 1972). From the experiments described above, this high temperature inclusion will cause differential expansion and propagate radial cracks. Blasting in a mechanically drilled hole requires a substantial portion of the explosive energy just to initiate such a radial crack pattern. An explosive placed in a laser-drilled hole will extend the radial cracks and produce increased fragmentation.

This concept has been verified experimentally by comparing fragmentation in granite holes drilled both mechanically and by lasers (Jurewicz, 1973). The drilling time of the holes by the two techniques was about the same, e.g., about 30 sec. for a 5 cm deep hole. In addition to verifying the concept that greater fragmentation

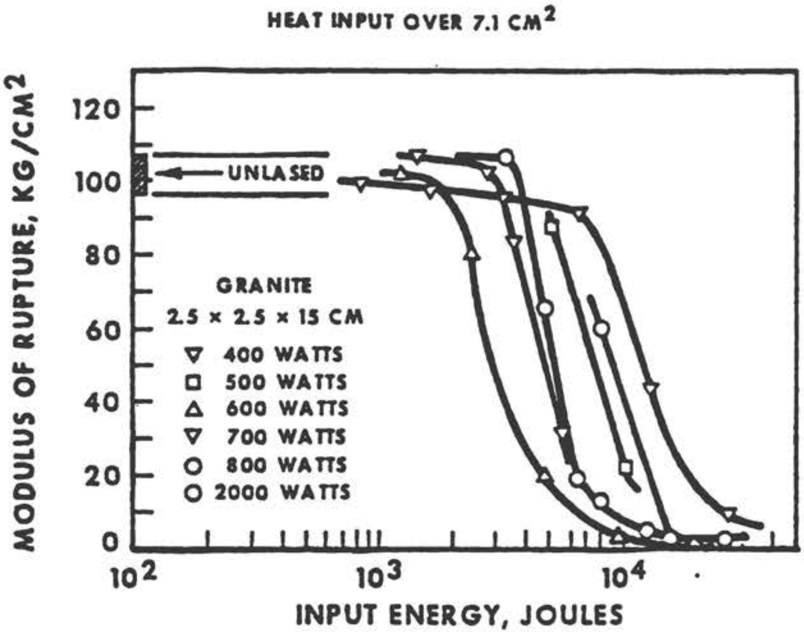


FIGURE 4-35 Reduction of strength of granite vs. input energy of lasers of different power. (Williamson, et al., 1968)

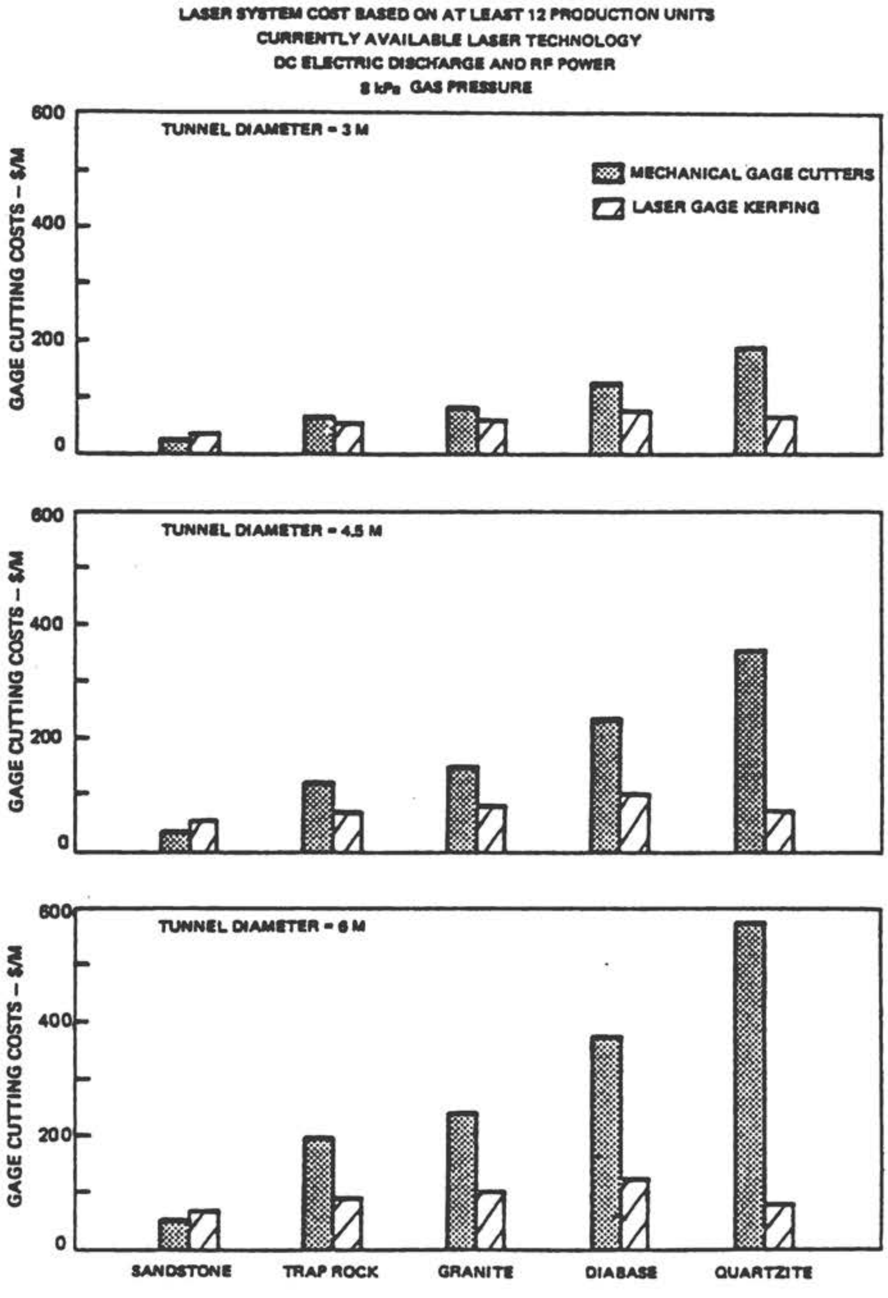


FIGURE 4-36 Comparison of tunnel gage cutting costs of mechanical gage cutters and with laser kerfing. (Jurewicz, et al., 1974)

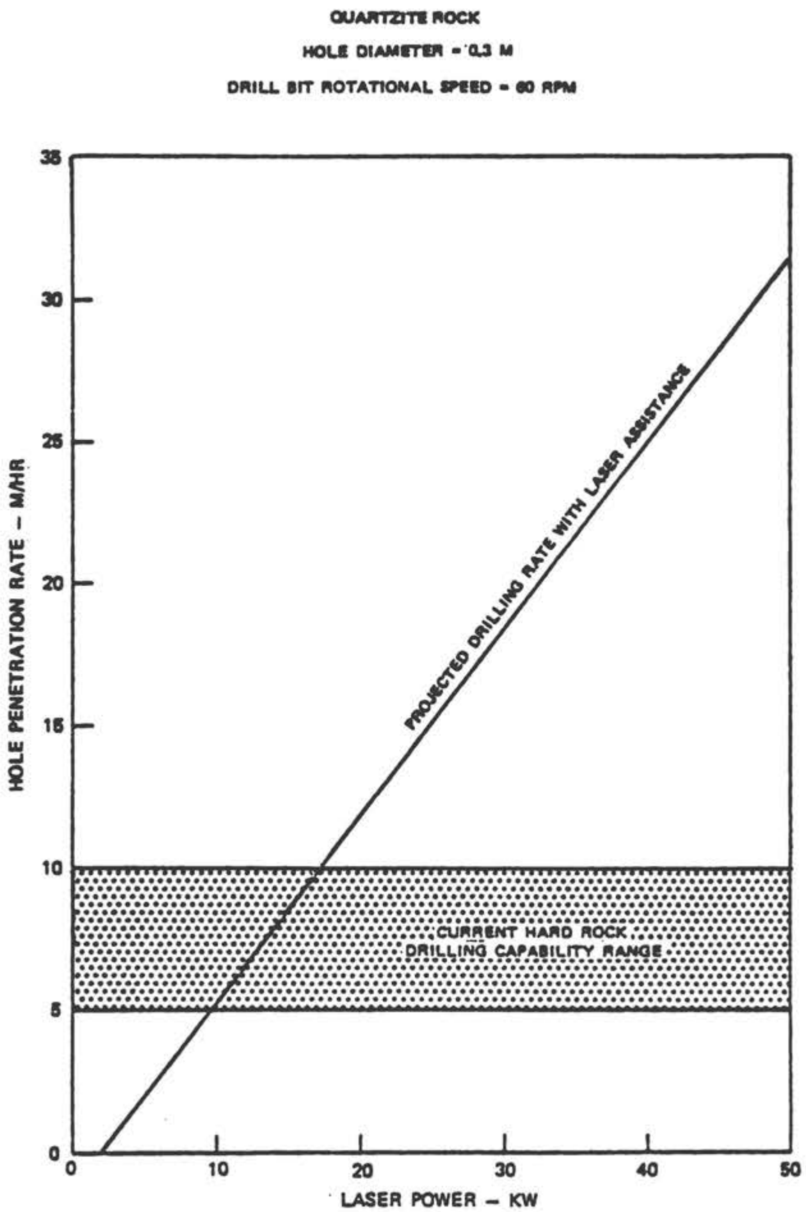


FIGURE 4-37 Potential improvement in hard rock drilling with laser assistance. (Jurewicz, et al., 1974)

results from a laser-drilled hole, the experiment showed that cracks extended inward about five times the hole depth for the laser-drilled hole.

The concept of continuous mining has been developed with continuous laser drilling and small amounts of explosive inserted when the laser moves on. An array of lasers could proceed along a working face parallel to the edge or in a helical array (Peterson, 1973). In areas where continuous blasting poses hazards, laser drilling could be used with a high frequency mechanical impactor (Envirotech, 1974). In this concept, the laser holes would be drilled first to thermally weaken the rock, and then the impactor would be applied to the surrounding surface.

4.7.8 OTHER NOVEL TECHNIQUES

A variety of other novel techniques of comminution are currently being explored. Among these are:

- Tumbling mill with an elastic wall
- Centrifugal rotor mill containing rotating discs and a fixed ring
- Laboratory mill with an abrasive plastic lining
- High temperature plasmas
- Selective crushing with a rubber squeezer
- Fluidized bed target in a jet mill
- Atomization of molten material by applying d.c. or a.c. potential
- High pressure water jets to break rock
- Electrohydraulic grinding
- Powdering of aluminum metal by ultrasound
- A machine to combine the advantages of crushing rolls and the edge-runner mill
- Chemical comminution
- Electrophysical methods of crushing rocks
- Thermal-mechanical size reduction processes

A review of these techniques, with references, is given in the Novel Techniques section of the Journal of Powder Technology's Size Reduction Review from 1971 to 1975.

4.7.9 RECOMMENDATIONS

Improvement is needed in all size ranges of comminution, from more control to yield a more uniform product in the mine with less oversize, to more efficient extreme fine grinding to yield a more uniform product with less undersize material. The pattern of breakage, i.e., the breakage function, appears to be hard to vary significantly under normal comminution conditions. However, it may be possible to alter the environment or to alter the material by thermal, prestressing, or chemical effects, so that when comminution is performed the material breaks to a more uniform size or with preferential liberation or with a significant reduction in energy.

A project should be awarded to critically assess the potential of novel comminution methods, using published experimental data for assessment. This project should be designed to compare the various novel methods on a rational basis in terms of such factors as the specific objectives of the comminution operation, the size distribution of the feed and products, the energy expended to achieve that size distribution, maintenance problems, and metal consumption, among others. Continued support should then be given to research on those novel methods of comminution that have realistic potential. A new technique or device developed in a laboratory may not contribute significantly to energy savings for two decades, but there are promising areas where support may contribute to marked energy savings in the future.

4.8 <u>Blasting</u>

4.8.1 INTRODUCTION

Rock blasting for comminution is an art that has been practiced for many years. Most of this practice has been quarry blasting and mine and tunnel face advancing. In both of these settings, a free face or volume is available into which the rock fragments (muck) are heaved. The objectives of the blasting operations are to (1) produce the desired fragment size distribution, (2) produce a muck pile that can be dug out and moved with relative ease, and (3) accomplish the blast with a minimum of blast noise and flyrock.

Standard blasting practice (Grimshaw and Watt, 1971) is to use a mixture of ammonium nitrate and fuel oil (ANFO) in boreholes of 2 - 15 in. diameter. In both quarry and open pit copper ore blasting the yield is about 4 tons of rock per pound of explosive. The energy content of a pound of explosive is about 15 x 10^3 Btu/lb or

1.4 kWh/lb. It is therefore of considerable interest in terms of both cost and energy to maximize the comminution of the rock in this first step.

Although considerable research has been done over the past 25-30 years, only in recent years have the basic details of commercial blasting events begun to be understood. While the typical production blast has a duration on the order of a second, it consists, nevertheless, of an orderly sequence of events (or perhaps disorderly if it is not designed properly or malfunctions). In the development, utilization, and understanding of military explosives, design and diagnostic techniques have been devised which are finding their way into the commercial sector.

4.8.2 MECHANISMS OF FRACTURE IN BLASTING

There have been basically two schools of thought on the mechanism of rock fracture by an explosion in a borehole. One is that fragmentation arises from a compression wave which, upon being reflected from a free face of the rock, generates a tensile pulse. Since rock is considerably weaker in tension than in compression (under static loading), the tensile pulse fractures it. The other school feels that the primary mechanism of failure is the propagation of radial cracks outward from the borehole by the pressure generated by the gaseous explosion products. It should be noted that these categories are quite simplistic.

When an explosive is detonated in a half plane of an elastic body, two outgoing compression waves are generated. The fast one is called the P, or primary, wave, and the slower one is called the S, or secondary, wave. When these waves strike the exposed surface they are reflected. The P wave reflects a primary, the PP wave, and a secondary, the PS wave; similarly, the S wave generates an SP and an SS wave. The P wave is also called the dilational wave because the deformation associated with this wave causes a volume change. The S wave is a shear wave and involves no dilation.

Careful photoelastic measurements of crack propagation following explosion in a borehole in Homolite 100 have been made by a group at the University of Maryland (Barker et al., 1978; Barker and Fourney, 1978; Fourney and Barker, 1979; Fourney et al., 1979; Holloway et al., 1980). Following the detonation, examination showed a very dense distribution of radial cracks around the borehole. However, from this dense distribution only about 8 to 12 dominant radial cracks were found to propagate any significant distance from the borehole. These dominant cracks sometimes branched and produced more radial cracks. The radial cracks produced large pie-shaped fragments. However, when reflected stress waves were present, circumferential crack networks were produced and the pie-shaped fragments broke into a fairly uniform fragmentation pattern.

Most real materials are filled with flaws, joints, grain boundaries, layers, etc. These investigators, therefore, prepared both internally flawed and layered Homolite 100 and repeated the experiments. In flawed material, both the P and S waves supply energy to propagate both radial cracks and cracks from the flaws whose initial directions are circumferential. Further propagation of existing cracks as well as generation of new cracks is caused by the reflected waves at other flaws. Very large flaws, such as seams of weakly bonded joints, can channel away the pressure of the explosion and reduce the fragmentation. Cracks can be intiated at flaws far from the borehole by the combination of the P wave tail and the shear wave. These flaws generally do not propagate in the radial direction and thereby enhance fragmentation. The maximum distance from the borehole that combined P and S waves can initiate cracks from flaws is a function of both the attenuation of the wave and the severity of the flaw. In the experiments on layered Homolite, these investigators examined both the mechanism of fragmentation from a single explosion and the effect of timing explosions from multiple boreholes. They found optimum conditions for both.

4.8.3 HIGH-SPEED CINEMATOGRAPHY

Winzer and his associates at Martin Marietta Laboratories (Winzer et al., 1979; Winzer and Ritter, 1980) have done field experiments using high-speed cinematography (11,000 to 14,000 images per second). They used blocks of limestone of up to 25 tons because they wished to approximate a production blasting operation while keeping the total yield small enough to screen the entire muck for fragment size distribution.

Since the boreholes were necessarily small, they used Gelamite explosive, water-coupled to the borehole, to simulate the usual ANFO explosive. They followed the usual practice of drilling rows of boreholes parallel to the face of the rock and used a commercial detonation timer. Timing is crucial because the burden is too great for an explosion in an inner row borehole to fragment the rock. Rather, it will blast upward like a cannon with excessive noise and dust. Only when firing in an orderly fashion with holes nearer to face (firing first, usually in a diagonal pattern, Figure 4-38), will the explosive maximize the fragmentation. Each block was carefully mapped, and they recorded the pattern, density, and distribution of the preexisting cracks which were large enough to draw in water by capillary action. They painted the cracks for increased visibility in the films.

By analyzing the results of several experiments they found that although framentation of the block is a continuous process, there are two distinct periods within the event. In the first period, cracks appear on the face in a region opposite the bottom of the borehole, almost coincidental in time to the arrival of the P wave at the face. There is no venting of gas, however. It is not known if these cracks

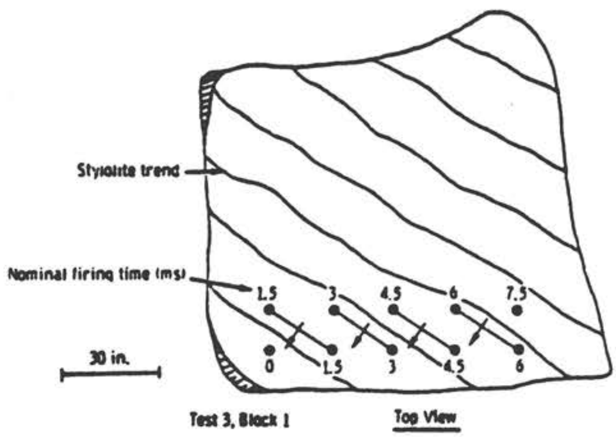


FIGURE 4-38 Firing pattern in multiple borehole test. (Winzer, et al., 1979)

are caused by the compressional P wave or by the tensile reflected PP wave. As blocks fly out from the face, a second event is seen. The blocks fragment within the air with no venting of gas. This secondary fragmentation is believed to arise from stress waves trapped in the blocks after they become detached. The initial, marked cracks propagate, but new fractures develop at the preexisting cracks and propagate both parallel and perpendicular to the natural bedding planes of the rock.

These results are consistent with the findings of the University of Maryland group for flawed and jointed Homolite, discussed above. The fragmentation results have shown that it is of paramount importance to know the rock structure in the design of blasting. Winzer et al. varied the firing times, and these findings show that both the tensile stress and the radial crack models of fragmentation are operative. They also established quantitative relationships between firing times and burden motion, flyrock velocity and production of backbreak and oversize (Winzer et al., 1980) and further showed that currently used timing guidelines are too short. However, of great interest in this study (Winzer et al., 1979) was the finding that the delay initiators from four different reputable companies all had very large statistical variation in timing accuracy. In sequential blasting this can cause inefficient fragmentation as well as excessive noise and dust. Clearly, better standards in manufacturing are required.

4.8.4 COMPUTER SIMULATION OF FRAGMENTATION

Large finite difference and finite element wave propagation computer codes, first developed at the multidisciplinary national laboratories, have been used to study the complex stress states in composite (bedded-like) structures. Their application to this geomechanics problem was easily accomplished. However, there is a much stronger motivation for utilizing these computer codes in the case of oil shale blasting. As oil shale processing technology is developing, it appears that much of the shale will be processed in place by the modified *in situ* (MIS) process.

In the MIS process, about 20-25% of the oil shale to be processed is mined and removed from underground. The remaining 75-80% is blasted and expanded to distribute the initial void uniformly through the rubbled zone to facilitate subsequent pyrolysis (retorting) of the shale to produce oil. This process has many complications compared to bench blasting. First, the blasting operation is not amenable to cinematography techniques. Second, a single blast operation for the preparation of an *in situ* retort involves several orders of magnitude more rock than a simple quarry bench blast, so the economic loss from an improper blast operation is sizeable.

Third, and perhaps most important, the confined nature of the blast severely constrains the design of the blasthole layout and timing delays. To illustrate this constraint, recall that a well rubbled muck pile is about 45-50% void. Since it is desired, for economic reasons, to mine out only 20-25%, the remaining 75-80% must be blasted and expanded in such a way as to distribute this void uniformly thoughout the muck (rubble) within the confines of the retort walls. This is a severe constraint and is the topic of considerable research and development for in situ oil shale processing technology.

Because of the above problems and complications, Sandia Laboratories undertook the development of a rock fragmentation model for use in the large shock wave propagation computer codes. The current state of development is in its application to field cratering events. The model computes both the fractured volume and the fragment size distribution within that volume. Although it is currently usable only in simple geometries, it can be extended to multiple blast hole configurations. The work of Kipp and Grady (1980) will now be summarized.

In a blast event the gas can reach a pressure of 200 kbar, compared to which the strength of the rock is negligible. This event pulverizes the rock immediately surrounding the explosion. The stress wave, moving outward, will continue to crack the rock until the stress falls below that required for compressive crushing. In addition, radial cracks are propagated, as mentioned above. The reflected wave stresses the rock in tension; the tensile strength is one tenth or less of the compressive strength, so further fracture occurs both radially and circumferentially.

Instead of explicitly treating individual crack response to wave impingment, Kipp and Grady modeled the dynamic fracture process as a time-dependent accrual of damage, where the level of damage characterizes the current load-carrying ability of the rock. The model also incorporates the concepts of inherent fracture-producing flaws and observed strain rate effects in rock fracture. The complexity of the waves generated by explosive charges, and the appearance of relief waves from regions that have fractured, necessitated the use of wave-propagation codes to address realistic geometries. The codes numerically integrate the conservation equations of mass, momentum, and energy along with the constitutive equations for the materials. In the present case, oil shale parameters were used.

When a rock mass responds to an applied load by fracturing, the size distribution of the resulting fragments is closely linked to the nature of the loading conditions. For example, in a medium grade of oil shale, the average fragment size is observed to vary with strain rate as illustrated in Figure 4-39. At low strain rates (quasistatic loading), large fragments form, because only a few cracks dominate the fracture process. At the other extreme, developed under plate impact conditions, tensile strain rates of 10^4 - 10^5/s develop for a very short time. As a result, tiny fragments are formed by the interaction of many small cracks growing simultaneously. The overall trend of

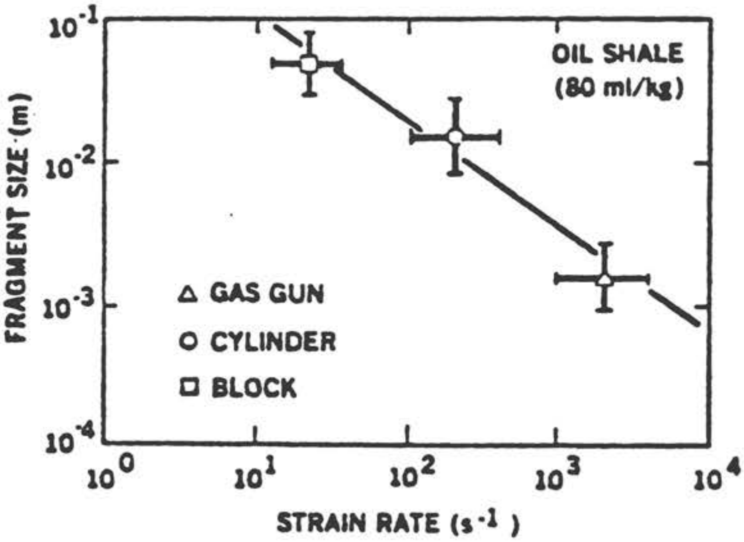


FIGURE 4-39 Experimental fragment size vs. strain rate for oil shale. (Kipp and Grady, 1980)

fragment size as a function of strain rate portrayed in Figure 4-39 is particularly important in blasting situations, since the rock surrounding an explosive charge may experience the entire range of these strain rates. The tensile strain rate from the divergence of the explosive-induced shock wave may be 10^2 - 10^3/s near the source and decay to 10^0 - 10^1/s at large distances. A nearby free surface may lead to strain rates of 10^3 - 10^4/s. Consequently, one expects to observe fine fragments near the explosive charge, with fragment size increasing with distance from the charge. Fine fragments are also expected near a free surface.

The prediction of the spatial distribution of fragments is the major goal of the numerical simulation, and the competent rock must be modeled so that the change in load-carrying ability is taken into account as the fracture state progresses. To model dynamic fracture and the subsequent time-dependent strength of the rock, it is necessary to consider the tensile fracture stress behavior. The fracture stress as a function of the loading strain rate determined by several experimental methods for oil shale is shown in Figure 4-40. At low strain rates, the fracture stress is small, because only a few large cracks are participating. At large strain rates, the fracture stress is larger than the quasistatic values by an order of magnitude. This variation in strain rate has been shown to directly reflect the presence of cracks in the rock, based on linear elastic dynamic fracture mechanics considerations.

By use of flaw density and flaw activation stress, these investigators derived a relation for a mean fragment size. As the stress increases, the fragmentation causes a decrease in the elastic and shear moduli and therefore the stress is unloaded. This tensile fracture and fragmentation model was incorporated into a two-dimensional Lagrangian wave-propagation code. The compressive behavior of the oil shale was modeled as elastic-perfectly plastic, with initial density of 2.27 Mg/m^3, bulk sound speed of 3000 m/s, Poisson ration of 0.4, and yield strength of 200 MPa. The damage and fracture surface area integral equations were reduced to approximate ordinary differential rate equations to be integrated in time in the wavecode. The explosive energy source is treated as an isentropic release of energy in a gas from the Chapman-Jouguet point of the explosive (Swegle, 1978).

4.8.5 COMPARISION WITH EXPERIMENT

Computer simulations followed by experiments were performed for four types of explosions: (a) simulations of the detonation of a small explosive charge in a meter-sized block; (b) damage variation with explosive burial depth (burden); (c) damage variation with charge shape and detonation scheme; and (d) simulation of an experiment in the floor of a mine. Good agreement was found for all.

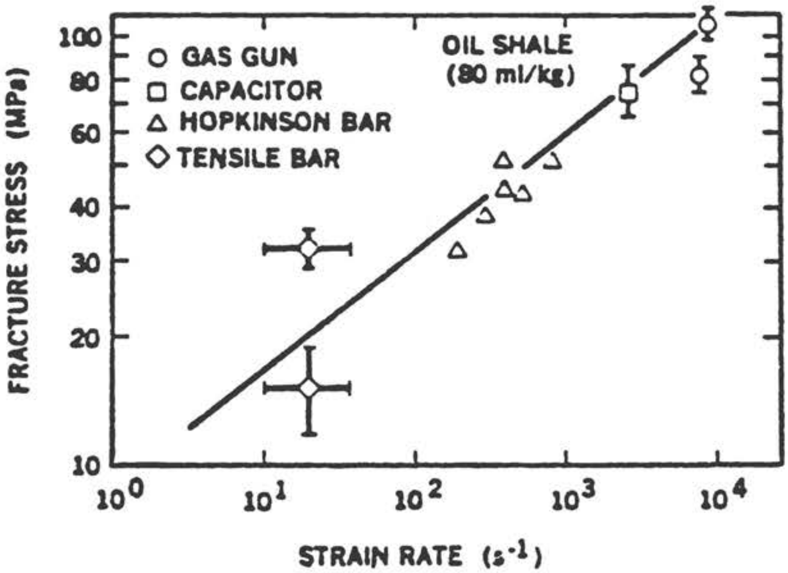


FIGURE 4-40 Experimental fracture stress vs. strain rate for oil shale (Kipp and Grady, 1980)

As an example, we will describe simulation (d). Figure 4-41 shows two orthogonal planes of the excavated crater. Figure 4-42 is a composite drawing in which the dashed lines are the actual crater profiles of Figure 4-41. Regions A, B, and C are the regions of calculated fragment size, which agree well with the experimental determination. A damage parameter, D, is defined as the volume fraction of material that has lost its load-carrying capacity. A value of $D = 0.2$ was calculated for different depths and distances and is shown as the solid line (the outer boundary of region C) in Figure 4-42. Good agreement with the crater profiles is seen.

The overall agreement of the calculations with experiments on different scales indicates that the model correctly accounts for the scale of the event, even though the parameters are based on laboratory-scale experiments. One of the most useful aspects of the model, when coupled into a wave-propagation code, is the ease with which changes in signficant parameters, such as explosive type or detonation position, can be evaluated. Although complex geometries involving multiple boreholes have not yet been calculated, individual borehole studies can reduce the number of uncertainties involved in such designs. This has the potential of greatly enhancing the efficient use of explosives to fragment rock.

The fundamental studies described in this section have significant potential for reducing of energy required in the explosive comminution of rock.

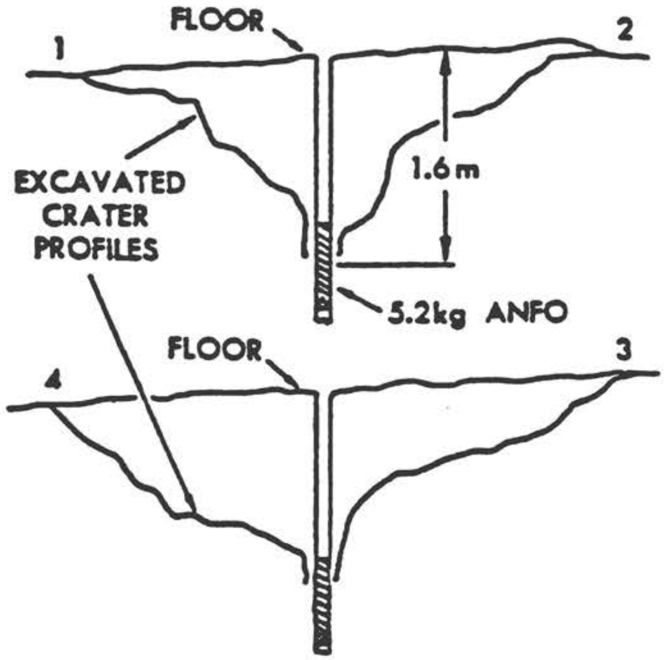


FIGURE 4-41 Two orthogonal profiles of an experimental blast crater (Kipp and Grady, 1980)

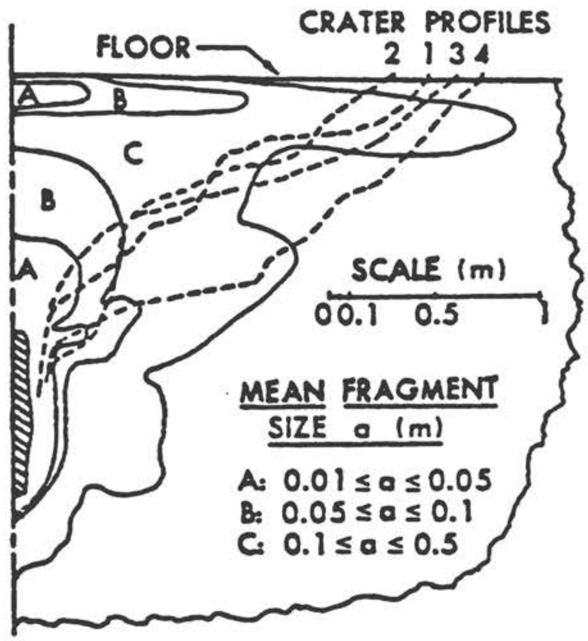


FIGURE 4-42 One section of crater profile of Fig. 4-41; dashed lines labelled 2, 1, 3, and 4 are crater profiles of Fig. 4-41; A, B, and C are regions of calculated mean fragment sizes; solid line outer boundary of region C is the calculated contour of damage D = 0.2 where D is the volume fraction of material that has lost its load-carrying capacity. (Kipp and Grady, 1980)

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Chapter V

CONCLUSIONS AND RECOMMENDATIONS

5.1 Introduction

This study has shown that comminution ranks high in industrial energy consumption in the United States. Within some individual industries--e.g., processing of metalliferous ores--energy consumed in comminution operations constitutes 50% or more of the total energy required for raw materials processing. Further, it has been shown that the vast majority of comminution tasks are carried out at efficiencies of less than 5%. Thus, clearly, size reduction processes should be an important national target for technological improvement.

In this report, study areas have been divided into two groups, Fundamental and Practical. The implication is that the practical group is more closely associated with activities of short-term payoff (three to five years), while the fundamental group is more closely associated with activities of long-term payoff. Tables 5-1 and 5-2 list the topic areas that emerged from each of these groups as being of critical importance in comminution.

Included in Table 5-2 are estimates of the annual energy savings (in billions of kWh) potentially achievable in each area if (a) existing technology were applied and (b) appropriate improved technology were developed. It is the opinion of the committee that these potential savings have not been realized to date because:

TABLE 5-1: Fundamental Topics of Critical Importance in Comminution.

1. Fragmentation Science
2. Particle-Fluid Dynamics
3. Particle-Particle Dynamics
4. Particle Characterization
5. Surface Science
6. Materials Science

TABLE 5-2: Practical Topics of Critical Importance and Estimated Annual Energy Savings*

Topic Area	Energy (10^9 kWh) Potential Savings with Existing Technology	Additional Savings Through Improved Technology
1. Classification Device Design	3	4
2. Comminution Device Design	1	5
3. Control	3	3
4. Grinding Additives	1	2
5. Materials	1	1

*These estimates are based on the 1978 total energy of 32 x 10^9 kWh identified in Section 2.2.

- o The magnitude of the potential savings has not been fully recognized.
- o Comminution is not recognized as a separate field within government funding agencies because of its interdisciplinary nature.
- o Industrial implementation of new or existing but unused technology involves a certain risk which engineering construction firms and users are often unwilling to assume.

This section of the report addresses these problems and suggests possible solutions.

The five major practical topics discussed in the report are:

- o Comminution Device Design (Section 4.2, State of the Art)
- o Classification Device Design (Section 4.3, Classification)
- o Control (Section 4.4, Instrumentation, Control, and Optimization)
- o Materials (Section 4.5, Corrosion and Wear)
- o Grinding Aids (Section 4.6)

The relative importance of the five main topical areas is indicated in Table 5-2, which gives the estimated energy savings based on the 1978 consumption of 32 billion kWh. The first column of Table 5-2 shows the potential savings estimated by the committee to be achievable by applying existing technology. It is expected that the majority of these savings could be realized in the short term, i.e., less than five years. The second column shows the potential savings estimated to be achievable in the long term by the development and application of novel technology.

The short-term estimated values were obtained by combining values and claims for percentage improvements reported in the literature for both practical studies on real plants and simulation studies in cases where direct measurements were either not available or inherently difficult to obtain.

The members of the committee who have been active in comminution research felt that it would be appropriate to give their best estimate of the potential future savings which could result from increased research and development. These are given in the second column of Table 5-2. These numbers are not the opinions of the committee members exclusively, but result from numerous discussions with research specialists both in the U.S. and abroad. Obviously, they are only speculative at present and are not supported by any firm data base.

5.2 Short-Term Problems and Benefits

In Table 5-2, the most important areas for short-term benefits are Classification and Control, while those for longer-term benefits are Classification, Control, and Comminution Devices themselves. The specific research recommendations for the five project areas of Table 5-2 are given in Table 5-3 (1-5). These recommendations are a summary of the recommendations given at the end of each section in the body of the report. If an agressive national effort were mounted to address the short-term areas, a combined potential energy savings of about 9 x 10^9 kWh annually could be realized. These potential savings represent approximately 25% of the total energy consumed in comminution.

Of the short-term areas, the simplest to implement is materials (Table 5-3,5), since no modification of existing comminution equipment is required. However, it should be noted that development and adoption of low-wear-rate media will require a strongly competitive media supply industry, since new products will be adopted by mineral operating plants only if there is a reduction in the total media cost, i.e., the product of media price ($/lb) and media consumption (lb/ton).

Application of known control technology (Table 5-3,3) is more involved, but can be carried out as a stepwise development program with benefits accruing at each stage (see Table 4-7). Implementation will require plant disturbances both for hardware installation and for development of control strategies. However, hardware generally can be installed during scheduled maintenance downtime, and the time required to develop control strategies need not be excessive, particularly if staff and operators have had appropriate training, including experience with real-time simulation models.

Achievement of the benefits expected from improving classification performance (Table 5-3,2) will require the most disturbance to existing plants and, therefore, will meet more management resistance, but it is expected that these benefits will be realized in larger increments. Implementation will require hardware changes to classifiers, pumps, and piping and could involve section shutdowns. A major obstacle to implementation is the uncertainty of achieving the desired results and the magnitude of potential adverse results. For these reasons, activities in off-line evaluation of the performance of classification devices and the use of computers to simulate closed circuit dynamic performance are expected to play important roles.

5.3 Long-Term Problems and Benefits

In the longer term, benefits from improved comminution devices (Table 5-3,1) are expected to approach the benefits anticipated from further improvement of classification techniques. The potential in both of these areas results from the very low efficiency of current comminution methods.

It is expected that an improved basic understanding of comminution will result in the greatest improvements in size reduction technology.

TABLE 5-3: Tabulation of Project Areas for Improvement of Energy Utilization

1. Comminution Device Design

OBJECTIVE: To gain energy benefits through the optimization of comminution device design procedures by more precise matching of strategy and task.

A Improved Equipment Design	B Improved Scale-up Design	C Improved Process Design	D Development of Novel Devices
Optimal equipment size studies	Development of improved mathematical models	Improved subsystem models, eg., crushers, grinding mills, and classifiers	Fragmentation physics, e.g., loading mechanisms for improved fracture along grain boundaries or minimizing small particle production
Liner configuration studies	Correlations between model parameters and design and operating variables	Integrated systems, i.e., circuit and plant, simulation	Transport
Influence of design variables on transport	Consideration of scale-up for liberation	System optimization	
Defining zones of maximum fragmentation and wear	Development of standardized testing procedures	Fragmentation physics	
Fragmentation physics	Fragmentation physics	Transport	
Media mechanics	Transport		

TABLE 5-3 (Continued)

2. Classifier Device Design

OBJECTIVE: To reduce energy consumption in comminution circuits by improved equipment design, including fundamental studies related to classification devices, by improved scale-up procedures, by improved process design and application of classifiers, and by seeking novel classification equipment.

A Improved Equipment Design	B Improved Scale-up Design	C Process Applications	D Development of Novel Devices
Influence of design variables on classifier efficiency	Improved mathematical models	Evaluation of multiple-stage classification	Fundamentals of particle behavior in fluids
Radical modifications such as distributed water injection in a hydrocyclone	Correlations between model parameters and design and operating variables	Evaluation of alternative configurations for size reduction/classification	
Fundamentals of particle behavior in fluids	Development of standardized testing procedures	Development of improved models including the dependence of operating variables	
		Fundamentals of particle behavior in fluids	

TABLE 5-3 (Continued)

3. Control

OBJECTIVE: To obtain reduction in net energy consumption per ton of product by seeking improved and novel instrumentation, by developing more powerful automatic control strategies, and by implementing existing instrumentation and control technology.

A Improved Instrumentation	B Improved Control Strategies	C Implementation of Existing Technology
Extending range of current measurements	Dynamic simulators, e.g., digital or small-scale experimental	Development of easy-to-use simulators
Development of simpler and less expensive devices	Application of modern control theory	
	Improved tuning techniques	
Novel devices, e.g., liberation, surface area, viscosity	Application of microprocessors	

4. Grinding Additives

OBJECTIVE: To obtain improvements in grinding efficiency by application of existing grinding aids or development of new grinding aids.

A Evaluation	B Development of New Additives
Development of standard testing methods	Molecular design for specific applications
Optimal usage studies	Understanding mechanisms including structural effects
Understanding mechanisms	

TABLE 5-3 (Continued)

5. Materials

OBJECTIVE: To reduce energy consumption in comminution by improved or new materials for liners and grinding media, and control of slurry chemistry.

A Liners	B Media
Evaluation of current alternatives	Evaluation of current alternatives
Mechanisms of liner wear	Mechanisms of media wear
Fundamental composition-wear studies	Fundamental composition-wear studies
Slurry chemistry	Slurry chemistry

The six main fundamental topic areas are listed in Table 5-1. The first three topics--i.e., fragmentation science, particle fluid dynamics, and particle-particle dynamics--relate directly to comminution subprocesses.

Surprisingly, the first of these topics, fragmentation science, is practically nonexistent on the U.S. research scene. The conditions under which a particle will break and the nature of the daughter fragments produced is being actively investigated in Europe, particularly in West Germany. As observed in Section 3.3, work of this type at the Institut für Mechanische Verfahrenstechnik at the Universität Karlsruhe has resulted in the development of a new technology for cement grinding for which an energy savings of 50% is claimed. Research in fragmentation science which complements the Karlsruhe effort should definitely be developed in this country.

The second fundamental topic, particle-fluid dynamics, is concerned with the way in which particles are transported and classified in fluids. This is an interdisciplinary activity which requires input from investigators in both fluid mechanics and comminution. At present, very few specialists in fluid mechanics are working with such complex systems, involving a broad size distribution of irregularly shaped particles in high concentration and with varying chemical and physical properties. As pointed out in Sections 3.4, 3.5, and 4.3, there are a large number of interesting and challenging problems, and individuals with the required expertise must be encouraged to work on them.

The third topic, particle-particle dynamics, is important in understanding particle transport in dry systems and the interaction between particles and the solid surfaces of equipment and grinding media. Researchers in powder mechanics should be encouraged to contribute to work on the types of problems described in Sections 3.4 and 3.5.

The last three fundamental topics in Table 5-1 are basically support areas which are critically important in comminution. Particle characterization--the experimental determination of the size, shape, composition, and other properties of particles--is extremely important for quantifying size reduction operations. As observed in Section 3.2, some new instrumentation is required and standardization of methods within and between industries is highly desirable.

The second topic, surface science, plays an important role in comminution in that the behavior of small particles, including microscopic rheological characteristics and microscopic aggregation characteristics, is often determined by the charge and lyophobicity of their surfaces. As pointed out in Section 4.6, a good understanding of the action of grinding aids, for example, can come only with an understanding of the associated surface phenomena. Interdisciplinary activities involving surface chemists and specialists in fluid mechanics and comminution are required for work on complex problems of this type.

The last topic in Table 5.1 is materials science. Contributions from this field are critical to development of new media and liner materials. The challenge associated with developing abrasion- and corrosion-resistant materials cheap enough to be used in comminution devices is very great. Materials scientists and engineers must be made aware of these problems, and appropriate levels of research funding provided.

The committee recognizes that Table 5-3 is lengthy and lacks priorities or guidelines for fundings. Comminution research in the U.S. has long suffered from underfunding, manpower shortage, and a general lack of recognition as a research field. The committee found that in essentially every aspect of the fields which it examined there were numerous obvious areas where research is needed to improve understanding. With such a plethora of needs the committee felt that if budgeting constraints prevent the broad-spectrum research effort that is required, a larger group than the committee should be involved in establishing priorities. For this reason the committee recommends that one or more workshops be held and attended by sufficiently representative numbers of specialists from government, industry, and university laboratories. From such workshops a more realistic cost-benefit ratio of each type of research may be obtained. With this information, Table 5-3 can be transformed into a list organized by priority.

5.4 General Problem

When contemplating innovation, U.S. industries face a common, overriding problem--the high risk and high cost of seeking the long-term benefits of reduced costs and increased productivity. The implementation of new comminution technology is no exception to this general problem. A standard comminution facility usually is very large, is capable of producing thousands of tons of ore per day and involves large capital costs, often amounting to tens of millions of dollars. Given such a huge scale of operation, what would be the most effective way of putting to practical use an improvement in design or control developed in the laboratory? Should only low-cost modifications be incorporated into existing facilities, or is it feasible to introduce major modifications by new designs?

Transfer from the laboratory to an operational unit involves several scale-up stages, each with attendant problems. If difficulties are encountered at any stage, the design change must be modified or abandoned. Obviously, great amounts of time and money must be devoted to these efforts, none of which carries a guarantee of success.

Industry is often content with existing facilities if they are reliable; as long as such facilities remain competitive, companies can pass along the increased energy costs. Manufacturers of comminution equipment have established product lines, and improvements usually are incremental to existing devices. Up to the present, there has been no attractive incentive for manufacturers to bear the very large costs of developing and proving a new comminution product line, even if some new device is more efficient. Furthermore, mining companies have little incentive to introduce new control devices that are largely untested in the field, and they may lack operating personnel adequately trained on the new devices. A mill shutdown caused by failure of an innovative method would not be viewed kindly by corporate directors.

5.5 Recommendations for Improvements

Because of the significant percentage of national energy usage expended by comminution facilities, this committee believes that it would be consistent with the national goal of energy conservation for the U.S. Government to help provide incentives for improvement in the device and control aspects of comminution. Therefore, we strongly recommend that the U.S. Government take a more active role with a long-term commitment. Three desirable modes of implementing this role are the following:

1. Increased aid to research organizations. Recipients of this aid would include university, industry, government, and nonprofit laboratories. Those involved in comminution research generally lack the modern specialized analytical and operating research equipment and the necessary technical staff support. In the case of universities, of which about six in this country have significant comminution programs, the role includes not only the improvement of our basic understanding of mechanisms, but also the training of new personnel. Serious attention should be directed toward the adequacy of present methods of funding and the encouragement of the interdisciplinary aspects of comminution studies in all these research organizations.

2. Establishment of local or regional pilot plant facilities. The first stage beyond the laboratory or bench-model of a new device or process may be called a pilot plant. In the case of comminution this would involve processing in the range of a few hundred pounds to a few tons per day. Such a flexible, or versatile, plant could quickly prove the feasibility of a new model, device, or process for testing in a large scale, or demonstration, facility. Both university and industry personnel should have use of such facilities, and they would serve as locations for both training and increased interaction of personnel from the two sectors. Although the committee found that in Europe such pilot plant facilities are usually located at universities, such location in the U.S. may be difficult. The Environmental Protection Agency now requires an environmental impact statement for the disposal of all products. University administrations may not wish to be involved in such complications. It may, in some cases, be more feasible to locate pilot plants near existing commercial operations which have solved the disposal problem, and to have funds provided for the travel of university personnel to such facilities.

3. Establishment of a major national facility for large-scale testing. Such a facility would perform the final tests before the incorporation of a new device or design into a plant or a manufactured product. Because of the size and operational costs of such a facility, it should be operated by an existing federal agency which can provide adequate technical support, e.g., the Bureau of Mines. Both university and industrial personnel should have access to this facility, with technical support provided by the facility's staff. The facility would also serve as a training ground for industrial personnel in these new

technologies. As with the pilot plant, the environmental impact requirements of such a demonstration facility could be most readily complied with by its location near an existing commercial operation.

By these means the government could assure more rapid testing and transfer of new comminution technology from laboratory to application without industry's being expected to assume a disproportionate risk.

4. Possible source of funding of scale-up facilities. Federal officials are developing the concept of increased R&D in the metals-mining industry through a 50-50 cost sharing between government and industry (Damask, 1980). This would involve research in generic areas of general interest. With such cooperative programs the knowledge and experience derived would flow to all participants, generally in amounts far greater than could be achieved by individual corporate or federally supported project investment in R&D. The program would be of basic and applied research through pilot plant development. These cooperative activities would be selected by those involved in the support, and the site of the research activity would also be chosen; it could be in-house, in a government lab, or at one or more universities. Discussions with representatives of the Department of Justice Antitrust Division have indicated that if such a program were properly organized it would have approval (Smith, 1980). In fact, the Department of Justice has just released a set of guidelines under which cooperative research between companies would be acceptable (Litvack, 1980).

The committee urges the individuals who will be involved in the development of this cooperative concept to include comminution as an important generic area of research in the mining-metals industry.

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Appendix A

METHOD FOR DETERMINING U.S. ENERGY CONSUMPTION FOR 43 COMMODITIES OF SECTION 2.2

Section 2.2 tabulates the energy used by comminution in the processing of 43 commodities. The results for each commodity were calculated from a knowledge of the energy requirements for comminution in the production of 1 ton of primary product, termed specific energy, and the U.S. production of that commodity for the years 1977 and 1978.

The principal source of U.S. production figures in 1977 and 1978 for the 43 commodities was the Bureau of Mines' Commodity Data Summaries 1979. For the commodities barite, boron, chromium, gold, lead, lithium, manganese, molybdenum, sand and gravel, silver, and zinc the information in the Summaries was not sufficient. In those cases the more complete Minerals Yearbook 1976, Bureau of Mines, was the reference. For example, the commodities gold and silver are recovered both as primary products and as by-products. Where this occurs, the energy requirements for the by-product commodity are credited to the parent product. These cases are footnoted.

The only commodity for which U.S. energy requirements for comminution were not obtained from the specific energy and U.S. production figures was grain. The total U.S. energy purchased for processing grains was multiplied by 0.7 to obtain the U.S. energy requirements for the comminution of grains. The remaining 30% of the purchased electricity is used for grain handling, sacking, and loading and for plant lighting, and other electrical usages.

The percentages of the energy used in the processing of a commodity that is accounted for by comminution was found by dividing the specific energy for crushing and grinding by the specific energy for processing of that commodity. The operations that are accounted for in the value for the overall specific energy depend on how much the commodity is processed. For example, the operations for a primary product such as refined copper may include mine excavation and transportation, crushing, grinding and flotation in the concentrator, smelting (by charge preparation and drying, reverberatory furnace, converter, anode refining furnace, gas cleaning, and acid plant), and refining (by electrolytic refining, cathode melting, and anode transportation). For crushed and sized stone the operations include mining and screening, primary crushing, secondary crushing, pulverizing, and shipping. The steel usage that results from the wear of liners and grinding media in comminution operations is given in Section 2.2.

The U.S. energy requirements for the steel consumed by comminution given in Table 2-2 incorporate the energy used for the liners in crushing devices and the media and liners in grinding devices. In the Battelle Columbus Laboratories 1975 work (on Energy Use Patterns in Metallurgical and Nonmetallic Mineral Processing), the liners and grinding media were assumed to be mild steel, which requires 0.0175×10^6 Btu per pound to produce. One British thermal unit is equivalent to 2.9288×10^{-4} kWh_f. The assumption is made that the energy used in the steelmaking process is convertible from British thermal units to kilowatt-hours with 32% efficiency. Thus 1 lb of steel media or liners requires 1.7 kWh_e.

Appendix B

METHOD FOR DETERMINING THE CONSUMPTION OF STEEL IN TABLE 2-6

The wear of liners in crushers and of media and liners in grinding devices, presented in Table 2-6, Section 2.4, were calculated using an empirical abrasion index, A_I (Bond, 1964). The equations for steel consumption in pounds per ton of comminution product are given in Table 2-6. An average value of the abrasion index for many materials using a standard steel paddle test is given by Bond (1961) as 0.228.

The liners of autogenous mills and the media and liners of rod and ball mills were assumed to be mild steel with an energy content of 0.0175×10^6 Btu per pound of steel. The steel in crusher liners is usually Hadfield, with an energy content twice that of ordinary steel.