AN EXPERIMENTAL, MODELING AND SIMULATION STUDY OF AUTOGENOUS GRINDING WITH EXTERNAL CRUSHING

by

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ABSTRACT

A mathematical model of an autogenous mill in circuit with a crusher has been developed, programmed and used to determine how the circuit should perform under varying operating conditions. The model of the circuit utilizes models of individual unit operations previously reported in the literature. Theses models were adapted directly where possible, and where major modification was required, a detailed discussion is presented.

The autogenous-mill-with-crusher configuration, when compared to an autogenous mill configuration alone, is expected to increase production rates for a given size of mill, allow smaller grinding units to process a given amount of ore, improve control of the mill product size and reduce power consumption per unit of ore processed.

Test work leading to the formulation of the circuit configuration was done in a small mill, batch grinding both autogenously and with a small amount of balls. Results of the test work are presented and discussed.
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INTRODUCTION

Milling prior to concentrating is an expensive step in the processing of mineral ores.

A typical conventional plant consists of a crushing section followed by a grinding section. The crushing section reduces the run-of-mine ore to less than 1/2 to 1 inch in size with jaw-, gyratory-, or cone-type crushers in closed circuit with vibrating screens. This feed then goes to the grinding section, where rod mills and/or ball mills in closed circuit with hydrocyclone, rake or screw classifiers reduce the ore to the size required for mineral liberation.

An alternative approach is to do away with the crushing section and its associated costs and treat run-of-mine ore (with only primary crushing) directly in large mills described as autogenous grinding (AG) mills, in which the ore is all that is used as grinding media, or semi-autogenous grinding (SAG) mills if some balls are required to supplement the ore as media.

Ore as media has been used in pebble mills for years. Advances in mill drive mechanisms have allowed the use of larger mills making primary AG feasible.

A serious drawback is that the higher impact forces at
work in these larger mills cause the big pieces of ore to break prematurely. This makes the media distribution too small to generate the forces required to break the middle-sized particles at a sufficiently high rate, and there is consequently a buildup in the mill of middle-sized or "critical-sized" particles.

There is no perfect solution to this buildup problem. The simplest way is to add a relatively small number of large (up to 5 inch) diameter steel balls to the mill to generate the required high-impact forces. This is called semi-autogenous grinding (SAG) and is in use in several plants at present. The major drawbacks are (a) an increased power draw by the mill which, while being more than offset by an increase in production rate, can be a serious problem if the mill drive is inadequate to power the increased load, (b) increased material costs for liners, the wear rate being higher due to the large impact forces of balls against the liners, and, (c) of course, the cost of the balls themselves. The most severe problem is that of liner wear, because in addition to the cost of the liners there is the labor cost and production lost during the liner-change period. Another problem is that the balls tend to break up the "media-sized ore" even more.

Another method has been to install pebble ports in the
grate discharge to remove these critical-sized particles for use as pebbles in regrind mills, or to treat them in a crusher for return to the mill.

It seems clear that these problems are caused by the inappropriate design of the mill in relation to the ore it is to treat. The diameter of the mill and its speed of rotation are limited by the competence of the ore[1], and the length of the mill is basically determined by the desired size distribution of product, which is a function of the residence time. This implies that, for autogenous grinding, the ore itself determines the capacity of a mill unit and hence the number of mills required for a given tonnage. While the designer is largely constrained by the ore characteristics, the operators are almost entirely at the mercy of the ore because these characteristics are constantly changing. This is why there have been so many disastrous experiences with autogenous grinding. In addition, very little is known about the sizing of mills of this type, and most designs have been based on pilot-sized mills, where media competency was hard to assess. At the present time, the state of the minerals industry as a whole is such that there are few new plants being built. As a consequence, there is very little actual performance data and most research is confined to pilot scale or smaller
units. Much effort has been focused on the determination of scaleup factors, but a clear statement of what size of machine is required for any given ore is not, at present, available.

The sizing of autogenous and semi-autogenous mills is beyond the scope of the current work. This project will focus on how mills can operate as efficiently as possible so as to allow the use of autogenous mills on ore which has previously been considered unsuitable for autogenous grinding.

In order to suggest possible ways in which grinding circuit performance can be improved, it is necessary to understand how these circuits work. Once the mechanisms at work in the mill are better understood it will be possible to suggest probable configurations which can be modeled, tested and eventually used in a plant situation.

It is the hypothesis of this thesis that impact and abrasion are both important to the size reduction of ores. Further, it is proposed that abrasion proceeds at a higher rate after impact breakage has been applied. Hence, it is postulated that the best configuration for size reduction is a mill of the correct size to generate impact forces just large enough to prevent a buildup of large particles, a grate discharge with large holes in it to allow for a
large circulating load, and a crusher to provide fast, efficient impact breakage to the middle-sized ore. It is this impact/abrasion cycle which makes the process so much more efficient. A flow sheet of this proposed configuration is shown in Figure 1.
Figure 1 - Flow Sheet of Proposed Configuration
LITERATURE SURVEY

A comprehensive review of the literature was conducted in order to find what information is available pertaining to the subject of autogenous and semi-autogenous grinding.

History, Economics and Reviews

The history of autogenous grinding up to 1955 was presented by Hardinge[2]. In 1908, the first recorded autogenous mill, in the shape of a cone, produced a fine product at the rate of two tons per hour. Hardinge suggested that the test was not sustained as there was no mention of a critical-size problem. In 1916, Nilsen developed an 8 foot diameter mill with a large diameter to length ratio. It treated 1.5 to 2 tons per hour, operated at 85-90% critical speed and produced a product that was 95% minus 44 microns. In 1932 the Hadsel mill was developed. It was 24 feet in diameter by 3 feet long and operated by using buckets which lifted the ore and dropped it on stationary steel plates. The product from this mill was minus 195 microns. It was not popular mainly due to its inefficient classification system, which was improved in 1934 by Hall. Hardinge said that by 1955 there had
already been many costly errors made.[2]

Bond[3] reviewed the early autogenous grinding processes and stated that in general less fines are produced than by conventional grinding equipment. He also proposed the idea that the required pebble size could be calculated from a known ball size by multiplying by the cube root of the ratio of specific gravities of steel and ore.

Much work has been done to determine the economics of autogenous and semi-autogenous grinding. Thunaes and Colborne[4] said that single stage autogenous grinding is the most economical but "... few ores lend themselves to finished grinding [in one-stage autogenous mills]...without a sacrifice in capacity and power costs." They went on to state that it is important to select ores for testing which are representative of what will be produced, and to enlist the aid of competent designers.

Others[5-7] reported that power consumption in autogenous mills is from 16-100% higher than in conventional mills. Kenyen[7] calculated in 1984 that at power costs below 4.5 cents per kilowatt-hour, AG and SAG mills are more economic than conventional mills. Barratt[8] compared operating and cost data for several plants using conventional and semi-autogenous grinding. He
stated: "The estimated and direct capital and operating cost savings in favor of the application of semi-autogenous grinding to porphyry copper ores are evident at any throughput", but especially at levels above 30000 tons per day. Themelis and Last[9] said that most copper ores cannot be ground autogenously due to their low media competency.

Ramsey[10] listed the AG and SAG advantages as lower capital costs, ease of handling wet sticky ore, lower steel consumption, and lower maintenance and labor costs. He also cited the drawback of higher power consumption, but gave no figure. He said that pilot plant testing is essential for the design of plants, but acquisition of enough ore can be a serious problem. Pizarro and Schlitt[11] listed similar advantages for AG and SAG and added that "If [SAG mills are] properly sized, the secondary crushing and screening plants, together with a secondary ball mill, can be eliminated."

McPherson et al[12], in a set of articles, discussed the state of AG and SAG mills, citing advantages and disadvantages and listing plants that are using the practice.

Bassarear[13] listed and described the seven basic configurations of rock grinding and lists plants where each.
is used. They are: (1) Autogenous - single stage, (2) Autogenous + crusher, (3) Autogenous + ball mill + crusher, (4) Autogenous + pebble mill, (5) Autogenous + ball mill, (6) Semi-autogenous - single stage, and (7) Semi-autogenous + ball mill. He also discussed steel and power consumption and computer control.

In the mid-1970's, Dannenbrink[14] said that larger mills will be the trend of the future, particularly for autogenous grinding, as more competition drives the industry towards lower grade ores. He also said that while in the past low capital cost was the most important criterion, lowest total mill cost will predominate in the future. At about the same time, McManus[15] said that the trend is toward three-stage crushing with ball mills or primary crushing with two-stage semi-autogenous mills, while Coburn and Mortenson[16] said that the next generation of systems will be the autogenous/ball mill/crusher configuration.

Operating Plant Reports

The literature contains numerous examples[17-35] of plants that are now using autogenous or semi-autogenous mills with varying degrees of success.
Most successful in using AG are the Swedish, notably at Boliden.[17-21] They have commissioned several fully autogenous plants (most using more than one stage) with good results.[17] Fahlstrom[18] said that 6 inch balls were used but discontinued due to high liner wear and crushing of the coarse charge. Plant personnel decreased the size of a mill by putting a 2 foot thick concrete layer onto the shell of a full-size operating mill. The result was that it was as efficient as the larger one. Decreasing speed and diameter gave lower liner wear and higher capacity. He also said that rubber liners provide cushioning which helps prolong the life of the large chunks of ore. Klomstadlien[19] reported a single-stage autogenous mill with an L/D ratio of about 1 which ground in closed circuit to 100% minus 8000 microns. He said there was no sign of segregation or overgrinding. Li[20] and Hoppe[21] reported that at the Stekenjokk Copper/Zinc project two-stage autogenous grinding is done in mills with L/D ratios greater than one. The feed ore is blended from coarse and fine ore bins in order to control the media size. Hoppe claimed long mills offer better control than short mills.

In the United States and Canada[22-32] the trend has been more towards semi-autogenous grinding than autogenous
grinding. McManus[22] said that the SAG mills at Lornex are controlled by the ratio of 4 inch to 5 inch balls that are charged, and that the more larger balls the coarser the discharge becomes. He said that the mining plan should provide for a consistent feed to the mill. By grinding drill core samples in a small test mill and comparing with actual operating experience on the ore extracted from the respective regions, it is possible to predict and plan the feed to be delivered to the mill. Talbott and Hartzog[23] reported that at the Delamar Mine, primary SAG mill ball addition is based on feed rate; balls are added if the throughput drops below a specified rate. The mill thus can operate almost fully autogenously. Hydrocyclone underflow is split between the primary SAG mills and secondary ball mills to provide for liner protection in the primaries. Liner life at both Lornex and Delamar is 5-8 months. Pettibone and Cody[24] reported that at the L-Bar Uranium mill 12 foot diameter mills operate with an 8% ball charge composed of 3-4 inch balls. This mill mainly scrubs sand grains from clay and carbonaceous cementing agents. Niemi[25] reported that at Cyprus Bagdad in Arizona an Autogenous/Ballmill/Crusher circuit is used. The plus 1/2 inch material in the AG mill discharge (which happens to be about 50% of discharge) is
crushed in a 7 foot shorthead cone crushe... inch and returned to the AG mill. The minus 1/2 inch material is treated in ball mills in closed circuit with hydrocyclones. Sisselman[26] reported that at Hibbing Taconite in Minnesota 36 foot autogenous mills grind 400 tonnes per hour to 78% minus 44 microns. Feed rate is controlled to maximize power draw. The recirculating load is only 100%. The hydrocyclones are controlled by the density of minus 1/8 inch material delivered to them (that is, the amount of water added to the feed sump). Plus 1/8 inch material is scooped back into the mill. He also reported[27] that at the New McDermott Mercury mine no crushers are used because the ore is primarily clay. The ore is milled in an 18 foot mill using opalite as media, the addition of which is controlled by power draw. Grind is controlled by pneumatic apexes on the hydrocyclones.

In Canada, several autogenous or semi-autogenous mills are operated. Nice[28] reported on the start up at Similkameen which was designed to use an autogenous mill, controlled, in large part, by a crusher. Testwork showed great promise as far as capacity and grind size control were concerned[29] (to be discussed later). In practice, however, the technique was a failure. Throughput was far below design, and the pulp level in the mill was too high.
All efforts to improve the situation, including closing the crusher gap and cutting larger holes in the grate, were of little use. Inspection of the interior of the mill showed that there was little coarse material, most of which was segregated by the grate at the discharge end. There was also an excess of fines produced. After everything else had failed, they added balls to the mills (7-8%) and it started working well. Liner wear was "bearable", but higher than autogenous milling. Counsell and Webber[30] reported on Denison Mines' Elliot Lake operation, which was expanded with two-stage semi-autogenous/pebble mills. The primary SAG mills are 28 feet by 10 feet and are charged with 4 inch balls. The pebble mills are 15.5 feet by 22 feet and are charged with pebbles screened from the ore prior to feeding the primary mills. Recirculating load is controlled by pneumatic apexes on the cyclones. Feed to the SAG mills is controlled by power draw and hydrostatic bearing pressure. Pebble addition is controlled by power draw. Liner change is done weekly (a 12 hour shutdown) and grate changes are done every 3-4 months. They reported that capital costs for the plant were 29% lower than for a conventional plant and that operating costs are lower, although no figures were given. Schabas[31] reported that Teck's Highmont Mine operates using the process that
failed at Similkameen, except that two-stage grinding is used. Two 34 foot diameter mills with 2.5 inch grate openings discharge 1-2.5 inch pebbles for the secondary mills, minus 1/2 inch hydrocyclone feed, and a middle-sized material which is crushed in a 7 foot cone crusher (which operates about a third of the time) to a nominal 1/2 inch. The plant was originally designed for SAG operation, but the plant metallurgist believed "if we could do semi-autogenous, we could do fully autogenous. He really stuck his neck out." McKim and Ambler[32] reported that at the Scully mine iron ore is ground autogenously in 24 foot mills. The liners are maintained on a 6 week schedule. There were no serious problems reported in the operation of this plant.

In Australia, Kleeman[33] reported that at Kambalda Nickel Operations two-stage autogenous milling is used. The ore is initially crushed to minus 8 inches and then screened into fractions for rock grinding media, pebble grinding media, cone crusher feed, and mill feed. The media additions are controlled by power draw to the mills. The 7 foot Symons crusher is operated in closed circuit. The AG mills are only 11 feet in diameter and have rubber grates with 1/2 inch by 1 1/4 inch openings. The pebble mills are 12.5 feet in diameter. Argall[34] reported
that Dizon Copper Mine in the Philippines operates a SAG/ball mill circuit with 28 foot diameter primary SAG mills using 4 inch balls at 6% of the mill volume. The grate openings are 1/2 inch and the mill is operated in open circuit. The grind of the SAG mill is 1-2% minus 210 microns. Kennedy[35] reported that the Guelbs Iron Ore project in Mauritania uses 32 foot diameter SAG mills with 1 inch balls. This operation is dry.

The Proceedings of the Autogenous Grinding Seminar in Trondheim, Norway[36] contain 18 papers reporting on the aforementioned and other plants. The majority of the papers deal with Scandinavian operations, but North American, Australian and South African operations are also chronicled.

The largest difference in the design of the mills is that the mills modeled after the Scandinavians tend to have a length to diameter ratio greater than one (or often two) whereas those modeled after the Americans tend to have a ratio less than one. This difference in design length to diameter ratio is a controversy which is still going on today. The proponents of the "short" mills say it is necessary to prevent overgrinding and ensure adequate mixing, while the proponents of the "long" mills say overgrinding is not as big a problem as media competency.
The latter view seems to be supported by the need of many of the American mills to use balls to supplement the media.

Research

Research has tended to focus on the modeling of mill performance, using data acquired from batch tests, and trying to predict the performance in a pilot or full-sized plant. Austin et al[37] and Weymont[38] attempted to model the autogenous mill using first order kinetics and treating the various forms of breakage separately. Test work was done in batch mills to determine specific rates of breakage for the three forms of breakage, namely from balls, from self-breakage, and from pebbles. Then previously-derived scaleup factors were used to adapt the rates to larger mills. Austin suggested that because of insufficient data from testwork, the models did not predict the performance very well. However, in Weymont’s thesis[38] some three years later, the model still did not work very well because, in this author’s opinion, the rates are not, in fact, first order, the scale-up factors are wrong, or the method of determining breakage rates is inadequate.

In later work[39]-[41], Austin et al modified the
models to account for chipping breakage of unrounded new feed to the mill.

Mular[42] discussed the mechanisms of autogenous grinding and proposed approximate breakage distribution functions for them. Kerl[43] did tests on several different types of ore. Since he was mainly interested in pebble grinding, he initially tumbled the pebbles to round them off. Data clearly showed that the production of fines, initially at a high rate, stabilized to a nearly constant and much lower rate as the pebbles are rounded off. He also presented data which showed that at middle sizes the rate of breakage is a minimum. He showed that while the presence of balls initially produces more fines (during the period when the feed particles are rounding off), the rate of fines production from the rounded ore is the same as without balls. This would imply that balls are of most use when there are rough edges to be chipped, but when the ore becomes rounded the balls produce the same breakage as ore. Johnson[44] identified six basic comminution mechanisms. As he defined them, they are:

1. **Impact** - breakage caused by high energy collisions with mill liners. Damage not enough to cause breakage is probably cumulative and extends crack length.

2. **Fracturing** - like impact but occurs along planes of
weakness either inherent from geology or induced from mining or crushing.

3. **Abras ion** - frictional breakage from contact with other media or mill liners producing particles which go directly to mill pulp or product.

4. **Chipping** - somewhere between impact and abrasion and generally applied to rough edges.

5. **Crushing** - defined as impact breakage of particles below critical size (which he did not define).

6. **Grinding** - the further reduction of crushed particles. Johnson said 60-65% of the weight loss of the feed can be attributed to chipping and abrasion, most of which goes directly into the product. He attempted to correlate breakage with static and dynamic material strength tests in order to produce a workable model of autogenous grinding, but was not able to do so.

As the terminology used in the literature to describe breakage is not consistent, that used in this thesis will be explained. Using the convention of Kelly and Spottiswood[45], breakage can be considered as consisting of three main types, which in order of increasing force or energy input are abrasion, resulting from localized stress, cleavage, resulting from compressive forces, and shatter, resulting from impact. These are shown in Figure 2[45].
Figure 2 - Representations of the Mechanisms of Particle Fracture and the Resulting Particle Size Distributions
Rowland and Kjos[46] developed a method for predicting the grinding circuit configuration likely to be best for a given ore based on various bench tests. Media competency is an important factor here. However, this work is mainly used to streamline the pilot plant operation, eliminating much of the testing time and ore requirements. MacPherson[47] correlated milling data on an 18 inch mill with actual results obtained in plants. He said ore from a new deposit can be evaluated using the small mill to predict the size of mill required, based on those in current operation. He said very few ores are consistent enough to be ground autogenously. He proposed an Autogenous Work Index which can be used to compare the capital and operating costs with conventional plants designed using the Bond Work Index. He suggested that the mining plan be such that ore varying in hardness be delivered to the mill in a blended manner. Also, after mine development has progressed, more samples should be tested so as to make alterations to the mill design, if necessary.

McDermott et al[48] studied autogenous grinding, including the effects of:

1. Feed size - Throughput is proportional to the fraction of the mill charge greater than six inches.
2. Mill charge volume - Power draw is proportional to charge volume.

3. Mill length - For a given reduction ratio, throughput is proportional to mill length for a 20% change in length.

4. Type of discharge - Peripheral discharge gives 5-10% lower power draw mainly due to less overgrinding.

5. Percent solids - No change was noted in trunnion discharge mills. In peripheral discharge mills, more throughput and a coarser grind were noted with lower percent solids.

6. Effect of 5 inch balls - A 2% ball charge produces a coarser grind in open circuit and is more expensive due to liner wear and ball costs.

These authors also noted that by removing 5% of the crude ore for use elsewhere as pebbles (1-3 inch material), throughput increased by 15-20% in the primary mill. They suggested that grinding efficiency is increased without this portion "getting in the way." Also, "by crushing approximately 5-10% of the [1-3 inch] material based on the crude feed rate, a 20% increase in throughput was attained."

Adam and Hirte[49] compared the performance of a 5.5 foot diameter mill at various lengths. They reported:
1. Grinding rate is higher per foot for short mills.
2. Power/ton of minus 74 micron material produced was the same for any length.
3. For longer mills, the circuit product particle size distribution was finer, circulating load was smaller and mill charge distribution was finer.

They observed, contrary to what was reported by McDermott et al.[48], that throughput is not proportional to mill length. This is mainly because they considered a larger range of length change.

Fahlstrom[50], in an article written in 1961, said that a large diameter to length ratio ensures little segregation, short retention time and lowest installation cost per unit of power. Contrary to what later researchers have found, he said energy consumption in AG and SAG mills is generally not more than conventional grinding despite the large reduction ratio involved. He added: "...each contact between two fragments in motion, irrespective of whether they are regarded as grinding bodies or material to be ground, involves a partial comminution." He also stated that in pebble mills the mass of media should be the same whether it is steel or rock, but since the density is lower in autogenous mills, the volume and the speed of the mill must be greater to compensate. He suggested that running
mills at supercritical speeds will build up a layer of rock on the liners to protect them. He noted that on some mills an electric ear is used to control coarse material additions. Finally, he asserted that mill speed can control whether a fine grind (slow speed) or a coarse grind (high speed) will be more efficient. He did not mention the effect of speed increases on coarse media; at high speeds the media will deteriorate so the grind will eventually become finer.

Bergstedt and Fagremo[1] did extensive work on a 10 foot by 3 foot autogenous mill. They varied the feed rate composition between "large" and "small" fractions. They reported that capacity is maximized when impact breakage is dominant in a mill, but the discharge then tends to be coarse. Also, the recirculating load should be minimized to maintain a high feed rate. They suggested that there should be coarse and fine ore bins, and that the fine ore should be further crushed prior to being charged into the mill. The most interesting conclusion of their work is that the size and speed of a mill is limited by the strength of the ore. The mill selected should be the biggest mill "which permits the coarse fraction in the feed to create a good working mill charge." They stated that tests in a pilot mill cannot tell if an ore is suitable for
autogenous grinding. This is the only reference to this particular problem that was found. The point which is implied is that for a given ore there is a limit to the size of mill which will allow the ore to establish a good working media, and if this size of mill has sufficient capacity, autogenous milling is feasible.

Hellyer and Campbell[29] did the testwork which was the basis of the design of the plant at Similkameen (see [27]). They tested this hard copper ore in a 6 foot by 2 foot pilot mill, comparing the performance of autogenous, semi-autogenous and autogenous-with-crusher circuits. The third had the lowest power per ton and more than twice the throughput of the autogenous mill. The semi-autogenous milling had a higher throughput than the autogenous with crusher, but consumed more power per ton. The product of the autogenous mill was finer than the other two. "By controlling the amount of crusher feed, it appears possible to control the size distribution of the media and, consequently, the size distribution of the mill discharge." Unfortunately, this configuration didn't work in practice due, in this author's opinion, to the fact that the ore was not competent enough to withstand the much larger forces at work in the full-sized mill.

Other researchers[51-53] have reported on the
benefits of crushing the middle-sized portion of the feed or recirculating load.

Another benefit of autogenous grinding is increased flotation recovery attributed to the absence of iron interaction.[54] This is partially offset by the higher energy costs. Khristov[55] stated that classification after autogenous grinding of ores or slags is more efficient than after ball milling mainly due to a lowering of the yield of the intermediate fractions (caused, one would assume, by the rounding due to abrasion).

Bachman et al.[56] tested samples of copper ores and found variations in feed rate at steady state could be tenfold. They stated that it is "... necessary not only to identify the rock types and their grinding characteristics, but also to develop mining plans that will assure that serious production throughput fluctuations are avoided." They also pointed out the need for more than one feed bin for control.

Wakeman and Mai[57] produced results which indicate "that, at least on a pilot scale, top ball diameter... [has] ...a very significant impact on power consumption and mill throughput."

Kavetsky et al.[58]–[59] did research on ball mills and noted: "... a reduced throughput related to the reduced
breakage rates of coarse particles in larger mills. It is proposed that at least part of this decrease in performance of large diameter mills is due to the relative decrease in effectiveness for breakage of high energy ball impacts" and that "... reducing the top ball size in the mill may improve the performance of large mills."

There has not been a good model of media size distribution and mass which can satisfactorily predict the effect of changing feed rates and, more importantly, ore characteristics, on the breakage rates and product size distributions of the mill. Since ore characteristics are random in nature and are very hard to estimate or predict, it seems that unless a probe is invented that can measure the size distribution inside the mill, accurate modeling of real mills will be impossible.
PRELIMINARY TESTWORK

In order to learn more about the grinding process in autogenous and semi-autogenous mills a series of tests were carried out. The tests were done in a small cylindrical batch mill of 18 inch diameter by 13 inch length constructed of mild steel with eight 3/4 inch square lifter bars welded to the shell. The mill was attached to a frame equipped with a digital tachometer and an analog torquemeter. The drive was a 1/4 horsepower variable speed motor linked through a series of pulleys, a gearbox and the torque- and speed-measuring apparatus to the mill.

The ore which was studied was from the Amax Corporation’s Henderson Mine. The ore was dried and sized in a Gibson vibrating screen deck. A sample of the fines was sized using Tyler-mesh screens and the Cyclosizer. The results of the sized sample are shown in Table 1. The specific gravity was measured and found to average about 2.8.

Red dolomite was selected as a tracer rock in the mill because of its distinctive color. The results of several work index tests showed that the dolomite was similar to the ore in grindability (See Table 2 and Figures 3 and 4). It was found that the dolomite, while having a slightly
### Table 1 - Henderson Ore Size Distribution

<table>
<thead>
<tr>
<th>Size (microns)</th>
<th>% Retained</th>
<th>Cumulative % Passing</th>
</tr>
</thead>
<tbody>
<tr>
<td>+63500</td>
<td>10.8</td>
<td>89.3</td>
</tr>
<tr>
<td>-63500 +31500</td>
<td>29.8</td>
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<td>24.9</td>
</tr>
<tr>
<td>- 9500 + 4760</td>
<td>8.2</td>
<td>16.7</td>
</tr>
<tr>
<td>- 4760 + 2380</td>
<td>3.8</td>
<td>12.9</td>
</tr>
<tr>
<td>- 2380 + 1190</td>
<td>2.1</td>
<td>10.8</td>
</tr>
<tr>
<td>- 1190 + 840</td>
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<td>10.0</td>
</tr>
<tr>
<td>- 840 + 590</td>
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<td>9.4</td>
</tr>
<tr>
<td>- 590 + 420</td>
<td>0.7</td>
<td>8.7</td>
</tr>
<tr>
<td>- 420 + 297</td>
<td>1.0</td>
<td>7.7</td>
</tr>
<tr>
<td>- 297 + 210</td>
<td>0.8</td>
<td>6.9</td>
</tr>
<tr>
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<td>0.7</td>
<td>6.2</td>
</tr>
<tr>
<td>- 149 + 105</td>
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<td>5.6</td>
</tr>
<tr>
<td>- 105 + 74</td>
<td>0.6</td>
<td>5.0</td>
</tr>
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<td>- 48 + 36</td>
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<td>0.4</td>
<td>2.7</td>
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<td>2.4</td>
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<tr>
<td>- 12</td>
<td>2.4</td>
<td>-</td>
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</table>
Table 2 - Work Index Data: Dolomite Versus Test Ore

<table>
<thead>
<tr>
<th>Size (microns)</th>
<th>Test 1: Both Before % Retained</th>
<th>Dolomite After</th>
<th>Ore After</th>
</tr>
</thead>
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<tr>
<td>-6680 +4760</td>
<td>22.1</td>
<td>10.3</td>
<td>12.4</td>
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<td>-4760 +3360</td>
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<td>14.1</td>
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<td>6.1</td>
</tr>
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<td>9.9</td>
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<td>- 590 + 420</td>
<td>4.0</td>
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<td>2.5</td>
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</tr>
<tr>
<td>- 74</td>
<td>0.0</td>
<td>14.8</td>
<td>10.0</td>
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</tbody>
</table>

Bond Work Index Ratio: 0.868

<table>
<thead>
<tr>
<th>Size (microns)</th>
<th>Test 2: Both Before % Retained</th>
<th>Dolomite After</th>
<th>Ore After</th>
</tr>
</thead>
<tbody>
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<td>-6680 +4760</td>
<td>24.1</td>
<td>12.0</td>
<td>13.3</td>
</tr>
<tr>
<td>-4760 +3360</td>
<td>15.4</td>
<td>8.7</td>
<td>10.3</td>
</tr>
<tr>
<td>-3360 +2380</td>
<td>9.4</td>
<td>4.9</td>
<td>5.8</td>
</tr>
<tr>
<td>-2380 +1680</td>
<td>7.1</td>
<td>3.5</td>
<td>4.1</td>
</tr>
<tr>
<td>-1680 +1190</td>
<td>7.0</td>
<td>4.1</td>
<td>3.8</td>
</tr>
<tr>
<td>-1190 + 840</td>
<td>4.6</td>
<td>3.5</td>
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</tr>
<tr>
<td>- 840 + 590</td>
<td>4.3</td>
<td>4.4</td>
<td>4.5</td>
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<tr>
<td>- 590 + 420</td>
<td>4.7</td>
<td>5.5</td>
<td>5.6</td>
</tr>
<tr>
<td>- 420 + 297</td>
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<tr>
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<tr>
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<td>7.2</td>
<td>7.0</td>
</tr>
<tr>
<td>- 105 + 74</td>
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</tr>
<tr>
<td>- 74</td>
<td>0.0</td>
<td>19.0</td>
<td>12.4</td>
</tr>
</tbody>
</table>

Bond Work Index Ratio: 0.712
Figure 3 - Work Index Test #1: Data in Graphical Form
Figure 4 - Work Index Test #2: Data in Graphical Form
lower work index, was much more breakable when in an environment of harder ore. This will not adversely affect the conclusions.

Initially, it was thought necessary to analyze the portion of the mill product which was too fine to visually determine the proportion of dolomite (by using acid dissolution of dolomite in individual size fractions). Several tests were done using measured blends of dolomite and ore. The method was found to be fairly inaccurate and was not used further because in the actual runs, the proportion of dolomite in the very fine fraction was less than the accuracy of the test.

Next, the optimal mill loading was determined by measuring the torque at various levels of loading and mill speed. The results of these tests are shown in Figures 5, 6 and 7. Figure 5 is a graph of torque versus mill loading for water only. Figure 6 is a similar graph for a 62.5% solids mixture of ore and water. Figure 7 is the same graph for a mixture of ore, balls and water (38.5% ore and 38.5% balls).

The amount of ore and water used in the tests was the autogenous mill load that gave maximum torque. Ore was recombined to approximate the original sample.

The ore was traced in three ways: the large (greater
Figure 5 - Mill Loading Test - Water Only
Figure 6 - Mill Loading Test - Ore and Water Slurry

- ■ = 50% Critical Speed
- ● = 60% Critical Speed
- ♦ = 70% Critical Speed

Torque (in-lbs)

Mill Loading (volume percent)
Figure 7 - Mill Loading Test - Ore, Balls and Water
than 1.5 inch) pieces were each marked individually, two
intermediate size fractions were each painted a different
color, and one size fraction was completely replaced by red
dolomite. In this way it was possible to determine rates
of breakage and at least a partial size distribution of
breakage products for four different sizes. In all, four
runs were performed: two with balls and two without. The
tests were replicated to estimate variance, but the
opportunity was taken to trace different intermediate size
fractions.

The semi-autogenous tests were done using a ball charge
of 6% of the mill volume (including interstitial spaces
between the balls). Ore and water added to the mill were
reduced by the volume of balls added (not including the
interstitial spaces) so that the level of slurry in the
mill remained the same.

The results of these tests are shown in Appendix A in

The data is presented first in table form and then a
graphical representation of all or part of the data
immediately follows.
INTERPRETATION OF EXPERIMENTAL RESULTS

The results of these tests led to the following observations. The dolomite tracer had to be crushed in order to generate enough of the desired size fraction. It was noticed that the top size of dolomite disappeared much faster than the work index tests indicated that it should. It is suggested that the relatively rapid rate of disappearance was at least partly due to the freshly crushed nature of the dolomite, whereas the ore had been handled prior to the test by conveyors in the plant and by shovels into the transport barrel and onto the drying pad.

The large pieces showed a visually-obvious tendency to round off with a measurable decrease in rate of disappearance with time. (See Figures A5 and A6.)

The presence of balls in both cases had a tendency to increase the overall production of fines, but there was a noticeable decrease in the amounts in size fractions just larger than 74 microns. It was clear that the balls were breaking these sizes faster than they were being produced by other breakage going on in the mill. It seems reasonable that the selection of ball size could be based on the size of particles accumulating in the mill.

Also, in all tests there was a tendency for the sizes
just larger than 74 microns to initially increase and then disappear as time went on. This is shown graphically in Figures 8 and 9. The increase and then decrease in the abundance of these particles is due to the rounding of the larger particles; as these larger particles become smoother they produce a finer abrasion product.

These test results suggest that by controlling the amount of impact breakage, the rate of breakage and size distribution of product can be controlled.

Another point of interest did not come from this research, but from the reported results of other researchers. In a large mill the breakage rate of large particles is rapid due to the large mass of the particle (called self-breakage) and the breakage rate of small particles is fast because they can be smashed between large particles. It is the middle-sized particles which take a long time to wear away causing the critical-size problem. Crushing middle-sized particles would quickly move them through this slow-breaking stage.

The above points led to the proposed configuration (which is shown in Figure 1) of an autogenous mill with large grate openings in series with a coarse screen, a splitter to control crusher feed and a crusher to reduce the size and "roughen-up" the edges of the ore before
Figure 8 - Tracer Data for Selected Sizes -
Autogenous Grinding Test 3
Figure 9 - Tracer Data for Selected Sizes -
Semi-autogenous Grinding Test 4
returning it to the mill.

As the results of this early test work were being interpreted, a shift of emphasis from the experimental study of semi-autogenous grinding to the study of the effectiveness of the autogenous-mill-with-crusher configuration was proposed. It should be noted, however, that the test work provided the insight into the workings of tumbling mills which led to the subsequent work.

As time and limited facilities did not allow laboratory research of the autogenous-mill-with-crusher circuit, it was decided to assess the usefulness of the circuit by mathematical modeling. This explains why the test work was discontinued when it was, and why the interpretation of the semi-autogenous part of the test work was not pursued in more detail.
SELECTION OF CIRCUIT CONFIGURATION FOR FURTHER STUDY

The following points detail the thought process which led to the proposed size reduction circuit in which a crusher is used in closed circuit with an autogenous mill.

1. Milling (especially autogenous milling) is a low-energy size reduction process. The particle-to-particle and particle-to-mill liner contacts tend to be low-energy collisions which will only break surface grains, particles damaged by previous handling (explosives, crushers, ore passes, etc.) or small particles which are trapped between the surfaces. This would explain the fact that individual particle breakage rates decrease with time in an autogenous mill environment, until the particles reach a small enough size to be broken by the large particles.

2. Balls, due to their relatively high specific gravity, have a higher kinetic energy in a mill environment and hence produce more violent collisions than ore particles alone. They inflict impact (or shatter) damage which would explain the increase in the breakage rate in semi-autogenous mills. In ball mills, the high
specific gravity and large size of the balls in relation to the ore ensures that most ore particles are broken between ball-to-liner and ball-to-ball impacts.

3. The optimum operating condition in a mill is to have the maximum number of particles in the freshly-damaged state, so that the breakage rates are higher and power consumption per ton broken is minimal. Balls in a ball mill or a semi-autogenous mill provide "in-house" impacting.

4. Ideally, cleavage or shatter should be applied using a process that is designed for that purpose. Balls inside a semi-autogenous mill are energy inefficient and the liners tend to be damaged by ball-to-liner contact. This ball-to-liner interaction is generally less damaging in ball mills because diameters of ball mills tend to be smaller than in semi-autogenous mills.

5. In order to provide for efficient external breakage of particles, the discharge grate should contain large openings. This will produce a large recirculating load in the system, which is normally undesirable in mineral processing due to the overgrinding which generally
accompanies it. However, since this recirculating load is mainly composed of coarse particles, overgrinding will not occur. In fact, paradoxically, the crusher will return ore to the mill which will produce a coarse abrasion product, thus increasing the circuit product size. Also, the large openings in the discharge grate will allow better flushing of the mill, thus reducing overgrinding and preventing a reduction in grinding efficiency associated with the buildup of fines. One possible drawback is that there will be less slurry in the mill to protect the liners.

6. The crusher will move the particles quickly through the critical-size (slow breaking) region.

7. Control of the media size distribution can also be achieved. By passing less material through the crusher, the size distribution of the grinding media gets relatively finer and a finer grind (albeit at a lower rate) is the result. Also, there is relatively more rounded material in the mill which produces a finer abrasion product.

All of these points come together in the proposed
circuit configuration. It was decided to use computer simulation to evaluate the viability of the circuit. The following section describes the various models used in the circuit simulation.
THE CIRCUIT SIMULATION

The circuit simulation utilizes a combination of models of the various unit operations developed by earlier researchers. A brief description of each model used in the circuit simulation is given here. For more detailed explanation the original works are referenced.

Appendix B contains a listing of the suite of computer programs which make up the circuit simulation, which were written in MS-Basic and later compiled for faster execution. The suite of programs consists of subroutines to update section parameters which are saved in files. The main program then executes the sections using an iterative technique. Execution time for the compiled main program is about one minute.

The Autogenous Mill Model

The discussion of this section of the circuit simulation will be in considerably more detail than the other parts. This is because it is an important part of the simulation, there were many changes to the model required, and the description of the model in the original article is inadequate.

The autogenous mill model selected was from
Stanley[60]. The model was designed for autogenous milling where only the interaction of particles or the interaction of particles with the mill liners produce size reduction; no other grinding medium is used.

Stanley points out that autogenous milling differs from conventional milling in two respects:

1) Size reduction occurs by two main methods, namely, (a) detachment of material from the surface of large particles (referred to as abrasion), and (b) disintegration of smaller particles due to the propagation of cracks (called shatter). Furthermore, abrasion and shatter breakage overlap on the size scale. This differs from ball or rod mills, where only shatter breakage is considered.

2) The grinding parameters of an autogenous mill are not independent of mill feed; the mill load is continually generated from the feed, and its parameters therefore depend directly on the feed.

In addition, the breakage history of the feed (mainly in recycled feed) has an effect on the breakage parameters. This will be discussed later, following the description of Stanley's work.

Because of the importance of mill load in autogenous milling, Stanley utilized a discretized mass balance model of the size fractions of the load, also known as the
"perfect mixing model" because that assumption is made.

The model assumes that the mill can be divided into sections or segments where perfect mixing can be assumed. In matrix notation, the perfect mixing model for a segment is written

\[
\frac{ds}{dt} = (BR - R)s + f - p
\]

where \( s \) is the vector of the mass content of the segment in successive size fractions,

\( f \) is a vector of mass flow rates of the successive fractions of feed to the segment,

\( p \) is a vector of mass flow rates of the successive fractions of the discharge from the segment,

\( R \) is the diagonal matrix giving the breakage rate of each component of \( s \), and

\( B \) is a lower triangular matrix (the breakage matrix) containing in each column the breakage distribution function for the corresponding size fraction.

The relationship between content and product is

\[
p = D \cdot s
\]

where \( D \) (the discharge matrix) is a diagonal matrix giving the discharge rates of each component of \( s \).

Equation (1) is really a mass balance about the mill.
Entering a particular size range of the mill contents are the feed \( f \) and the products of breakage from larger sizes \((BRs)\), and leaving that size range are the mill discharge \((Ds)\) and the material selected for breakage \((Rs)\).

Thus,

\[
\frac{ds}{dt} = (BR - R - D)s + f
\]  

and for the steady state condition,

\[
(BR - R - D)s + f = 0
\]  

If \( B \) is known or assumed, and \( f \), \( s \), and \( p \) are known, both \( R \) and \( D \) can be computed. If \( s \), the mill contents, is not known, then \( R \) and \( D \) cannot be separated; however, a combined parameter, such as \( DR^{-1} \), can be calculated if \( f \) and \( p \) are known. The method Stanley used was to determine experimentally the relationships between mill and ore conditions and the parameters \( B \), \( R \), and \( D \). These parameters are then calculated based on assumed mill contents, and mill contents are recalculated. By iteration, the mill contents which establish steady state are found.

Before calculating breakage rates, a reasonable breakage distribution had to be formulated. It is in this matrix that the simultaneous abrasion and shatter breakage
features of the autogenous mill are included.

Breakage Distribution Function

The matrix is lower triangular and contains, in successive columns, the breakage distribution function for the corresponding size fraction. This matrix will hence consist of functions describing abrasion in the coarse sizes, shatter in the fine sizes and a transition zone in the middle.

The abrasion breakage distribution function that was used by Stanley was largely intuitive and based on the nature of abrasion breakage. Abrasion has been defined as size reduction by the superficial detachment of relatively fine material from the surface of larger particles: it results in the slow "whittling away" of a central core with a production of more and more fine detritus. Thus, when particles evenly distributed over a single size fraction are subjected to an abrasion "event", all the particles will be slightly reduced in size and some near the lower limit of the size fraction will pass into the smaller fraction. The product of this abrasion will report to a number of size fractions at considerably finer sizes.

It has been postulated by Wickham[61] that if a mass m has been lost from particles in a single size interval,
then \(0.354\text{m}\) (i.e. \(1/[2^3/2]\text{m}\)) will report to the next finer size interval, and the remaining \(0.646\text{m}\) will form the detritus which is spread over a number of considerably finer size intervals. This postulate, with \(m=0.01\), was used by Stanley. The distribution used is shown in Table 3.

The detritus was arbitrarily given a modified Rosin-Rammler distribution,

\[
B(x,y) = \frac{1 - e^{-(x/y)^p}}{1 - e^{-1}}
\]

where \(B(x,y)\) is the proportion of particles originally of size \(y\) that are smaller than size \(x\) after breakage. The addition of the index \(p\) is a modification proposed by Broadbent and Callcott[62]. Where \(p>1\), the effect is to concentrate the products of breakage nearer the parent size than when \(p=1\). The values used by Stanley were \(p=1\) for the abrasion detritus and \(p=2\) for the shatter breakage function shown in Table 4. One point worth clarifying concerns the initial values in the Rosin-Rammler sections of Tables 3 and 4 (opposite size fraction numbers 8 and 1, respectively). The value of \(y\) used is apparently the geometric mean of the size range, while the value of \(x\) is the upper limit of each size range. This explains why the
Table 3 - Abrasion Breakage Distribution Function, $m=0.01$

<table>
<thead>
<tr>
<th>Number of Size Fraction ($\sqrt{2}$ series)</th>
<th>Proportion Appearing in Fraction After Breakage</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>0.9900</td>
</tr>
<tr>
<td>2</td>
<td>0.0035</td>
</tr>
<tr>
<td>3</td>
<td>0.0000</td>
</tr>
<tr>
<td>4</td>
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<td>5</td>
<td>0.0000</td>
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<tr>
<td>6</td>
<td>0.0000</td>
</tr>
<tr>
<td>7</td>
<td>0.0000</td>
</tr>
<tr>
<td>8</td>
<td>0.0007</td>
</tr>
<tr>
<td>9</td>
<td>0.0012</td>
</tr>
<tr>
<td>10</td>
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<td>11</td>
<td>0.0009</td>
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<tr>
<td>12</td>
<td>0.0007</td>
</tr>
<tr>
<td>13</td>
<td>0.0005</td>
</tr>
<tr>
<td>14</td>
<td>0.0004</td>
</tr>
<tr>
<td>15</td>
<td>0.0003</td>
</tr>
<tr>
<td>16</td>
<td>0.0002</td>
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<tr>
<td>17</td>
<td>0.0002</td>
</tr>
<tr>
<td>18</td>
<td>0.0001</td>
</tr>
<tr>
<td>19 or greater</td>
<td>0.0000</td>
</tr>
</tbody>
</table>
Table 4 - Shatter Breakage Distribution Function

<table>
<thead>
<tr>
<th>Number of Size Fraction (\sqrt{2} series)</th>
<th>Proportion Appearing in Fraction After Breakage</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>0.1979</td>
</tr>
<tr>
<td>2</td>
<td>0.3310</td>
</tr>
<tr>
<td>3</td>
<td>0.2147</td>
</tr>
<tr>
<td>4</td>
<td>0.1226</td>
</tr>
<tr>
<td>5</td>
<td>0.0654</td>
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<tr>
<td>6</td>
<td>0.0338</td>
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<tr>
<td>7</td>
<td>0.0172</td>
</tr>
<tr>
<td>8</td>
<td>0.0087</td>
</tr>
<tr>
<td>9</td>
<td>0.0043</td>
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<td>10</td>
<td>0.0022</td>
</tr>
<tr>
<td>11</td>
<td>0.0011</td>
</tr>
<tr>
<td>12</td>
<td>0.0005</td>
</tr>
<tr>
<td>13</td>
<td>0.0003</td>
</tr>
<tr>
<td>14</td>
<td>0.0001</td>
</tr>
<tr>
<td>15</td>
<td>0.0001</td>
</tr>
<tr>
<td>16 or greater</td>
<td>0.0000</td>
</tr>
</tbody>
</table>
second value in each distribution is larger than the first.

Stanley assumed that the same abrasion and shatter breakage distribution functions hold regardless of the initial particle size. Even though it has been suggested that this assumption is not correct[63][64], there is support for the premise that the general form of the distribution function is more important than its precise details.

Two final parameters need to be known before a complete breakage distribution function can be set up. These are the upper and lower size limits of the transition zone between abrasion and shatter. The upper limit is the size above which no shatter breakage occurs and is called the shatter limit, and the lower limit is the size below which no abrasion breakage occurs and is called the abrasion limit. In Stanley’s testwork, these limits were determined by an inspection of the mill contents. It was possible to fix the size interval in which the sharp edges and conchoidal faces characteristic of shatter breakage first appeared to any significant extent among the smooth fragments generated by abrasion. The upper size of this interval was taken as the shatter limit. The smaller the size the larger was the proportion of sharp edges, until there were no longer any smooth particles. The upper size
of the first interval to contain no rounded particles was taken as the abrasion limit. Generally, the transition from all-abrasion to all-shatter occupied six √2 size intervals.

It was found that the shatter limit could be related to any of the four coarsest √2 size fractions in the mill feed, provided that such fraction constituted at least 2% of the mill feed. Mill feed is composed of circuit feed (new material fed to the circuit) plus classifier underflow. The relationships obtained for the shatter limit are given in Table 5.

[As an aside, it seems strange that for this simulation the shatter limit was not calculated from mill contents, because, as will be seen later, mill contents are calculated by iteration and will, at convergence, be known. For example, if the grate size were to be increased and no external crushing were done so that the classifier underflow were simply returned to the mill, the concentrations of the coarse fractions of the mill feed and hence the calculated shatter limit would change and yet the conditions in the mill should not be any different. This paradox came up several times in Stanley’s work: results which were derived from an inspection of mill contents and should reasonably be a function of mill contents were}
### Table 5 - Equations for Calculating the Shatter Limit

Calculation Size (first $\sqrt{2}$ size retaining at least 2% of mill feed)  | Equation for Shatter Limit, $\mu$m ($x =$ percentage of mill feed retained on given size)
---|---
151712 | $(2.45 + 0.32x) \times \left(\frac{1}{1.3 + x}\right)$
107296 | $(2.45 + 0.12x) \times \left(\frac{1}{1.3 + x}\right)$
75856 | $(2.45 + 0.09x) \times \left(\frac{1}{1.3 + x}\right)$
53648 | $(2.45 + 0.06x) \times \left(\frac{1}{1.3 + x}\right)$
related to mill feed or (worse yet) circuit feed even though the simulation must calculate the mill contents for other parameters to be calculated.]

Because of the erratic nature of the equations in Table 5 at high percentages of the respective size fractions, an upper limit of 26824 μm was set on the computed shatter limit. The abrasion limit was taken as being five √2 size intervals smaller than the shatter limit (i.e. it was one-eighth of the shatter limit) with a maximum of 3353 μm (26824 divided by 8).

Within the abrasion-to-shatter transition zone, the breakage distribution function was calculated as a simple linear combination

\[ B = aB_1 + (1-a)B_2 \]  \hspace{1cm} (6)

where \( B_1 \) is the abrasion breakage distribution function,

\( B_2 \) is the shatter breakage distribution function,

and

\( a \) is the proportion of the distance (in size interval terms) across the transition zone.

The abrasion, transition and shatter breakage vectors are then combined to give the total breakage distribution matrix. The general form of this matrix is given in Figure 10. It should be noted that a, b, c, d ... correspond to the values in Table 3 for the abrasion breakage
<table>
<thead>
<tr>
<th>Abrasion breakage</th>
<th>Transition zone</th>
<th>Shatter breakage</th>
</tr>
</thead>
<tbody>
<tr>
<td>a 0 0 0 ..</td>
<td>0 0 0 0 0 0 0 0</td>
<td>.. 0 0 0</td>
</tr>
<tr>
<td>b a 0 0 ..</td>
<td>0 0 0 0 0 0 0 0</td>
<td>.. 0 0 0</td>
</tr>
<tr>
<td>c b a 0 ..</td>
<td>0 0 0 0 0 0 0 0</td>
<td>.. 0 0 0</td>
</tr>
<tr>
<td>d c b a ..</td>
<td>0 0 0 0 0 0 0 0</td>
<td>.. 0 0 0</td>
</tr>
<tr>
<td>e d c b ..</td>
<td>0 0 0 0 0 0 0 0</td>
<td>.. 0 0 0</td>
</tr>
<tr>
<td>f e d c ..</td>
<td>6/7a+1/7p 0 0 0 0 0 0</td>
<td>.. 0 0 0</td>
</tr>
<tr>
<td>g f e d ..</td>
<td>6/7b+1/7q 5/7a+2/7p 0 0 0 0 0 0</td>
<td>.. 0 0 0</td>
</tr>
<tr>
<td>h g f e ..</td>
<td>6/7c+1/7r 5/7b+2/7q 4/7a+3/7p 0 0 0 0 0 0</td>
<td>.. 0 0 0</td>
</tr>
<tr>
<td>. h g f ..</td>
<td>6/7d+1/7s 5/7c+2/7r 4/7b+3/7q 3/7a+4/7p 0 0 0 0 0 0</td>
<td>.. 0 0 0</td>
</tr>
<tr>
<td>. . h ..</td>
<td>6/7e+1/7t 5/7d+2/7s 4/7c+3/7r 3/7b+4/7q 2/7a+5/7p 0 0 0 0 0 0</td>
<td>.. 0 0 0</td>
</tr>
<tr>
<td>. . . ..</td>
<td>6/7f+1/7u 5/7e+2/7t 4/7d+3/7s 3/7c+4/7r 2/7b+5/7q 1/7a+6/7p 0 0 0 0 0 0</td>
<td>.. 0 0 0</td>
</tr>
<tr>
<td>. . . . ..</td>
<td>6/7g+1/7v 5/7f+2/7u 4/7e+3/7t 3/7d+4/7s 2/7c+5/7r 1/7b+6/7q 0 0 0 0 0 0</td>
<td>.. 0 0 0</td>
</tr>
<tr>
<td>. . . . . ..</td>
<td>6/7h+1/7w 5/7g+2/7v 4/7f+3/7u 3/7e+4/7t 2/7d+5/7s 1/7c+6/7r 0 0 0 0 0 0</td>
<td>.. 0 0 0</td>
</tr>
<tr>
<td>. . . . . . ..</td>
<td>. 5/7h+2/7w 4/7g+3/7v 3/7f+4/7u 2/7e+5/7t 1/7d+6/7s 0 0 0 0 0 0</td>
<td>.. 0 0 0</td>
</tr>
<tr>
<td>. . . . . . . ..</td>
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<td>.. 0 0 0</td>
</tr>
<tr>
<td>. . . . . . . . ..</td>
<td>. . 3/7h+4/7w 2/7g+5/7v 1/7f+6/7u 0 0 0 0 0 0</td>
<td>.. 0 0 0</td>
</tr>
<tr>
<td>. . . . . . . . .</td>
<td>. . 2/7h+5/7w 1/7g+6/7v 0 0 0 0 0 0</td>
<td>.. 0 0 0</td>
</tr>
<tr>
<td>. . . . . . . . .</td>
<td>. . 1/7h+6/7w 0 0 0 0 0 0</td>
<td>.. 0 0 0</td>
</tr>
<tr>
<td>. . . . . . . . .</td>
<td>. . 0 0 0 0 0 0</td>
<td>.. 0 0 0</td>
</tr>
</tbody>
</table>

Figure 10 - General Scheme of the Breakage Distribution Function
Three-part Matrix
distribution function for sizes 1, 2, 3, 4 ..., respectively. Also, p, q, r, s ... correspond to the values in Table 4 for the shatter breakage distribution function for sizes 1, 2, 3, 4 ..., respectively.

The next step is to calculate breakage rate and discharge rate functions.

Breakage and discharge rates

Figure 11 shows typical breakage (R) and discharge (D) rate functions Stanley computed from experimental work. Starting from the medium values in the coarser sizes, the breakage rate increases rapidly to a maximum at the shatter limit. As shatter begins to appear, the breakage rate drops to very low values for several size fractions. These are the critical sizes in autogenous milling. As the sizes get smaller, the breakage rate rises to another maximum as the combined abrasion and shatter zone is crossed. This second maximum is usually one of several pseudo-maximums occurring at the crest of the main shatter breakage rate maximum. From there the breakage rate drops rapidly as the finest sizes are approached.

The discharge rates (D) are by definition zero for all sizes from the coarsest down to the grate size. They then increase rather rapidly over several sizes towards a
Figure 11 - Typical Breakage Rate and Discharge Rate Functions
constant plateau value, which is similar to the discharge rate of water from the mill. At steady state, the water discharge rate will be equal to the water feed rate to the mill.

Figure 11 clearly illustrates one of the advantages of having a crusher in the circuit. The crusher would take particles as they are to enter the low breakage rate region, or trough, and reduce them to a finer size that has traversed this region.

The perfect mixing model shows that if B, R, D, and the mill feed are known, the mill content and product can be calculated. Modeling of autogenous mills is thus basically the modeling of the three parameters.

The approach to modeling breakage rates in the abrasion breakage zone was to study pebble wear (or pebble mass loss) rates. The main equations of interest reported by Stanley are shown below.

\[
\text{Mass of Pebble (grams)} = \frac{d^3}{496.53}
\]  

(7)

where \(d\)=geometric mean of the passing and retaining screen sizes in millimeters. This equation, which has particle density and a shape factor included in it, was derived from the measurement of several particles. The definition of
wear rate was given as:

Wear rate (Mass/time)) =

\[
\frac{\text{Average mass of pebble entering size fraction} - \text{average mass of pebble leaving size fraction}}{\text{Average residence time in size fraction}}
\]  (8)

Wear rates were then converted to breakage rates $R$ for use in the model.

The general formula for residence time in a size fraction is

\[
\text{Residence time} = \frac{\text{Quantity contained}}{\text{Entry rate}}
\]  (9)

The "quantity contained" is the mass in each $\sqrt{2}$ size interval of the mill load.

The "entry rate" has two components: new feed, and material from coarser sizes within the mill. The fact that the feed material is randomly distributed across the size interval was handled in what Stanley called "the usual method" of assuming all the feed material enters at the top size of the interval but at a rate that is one-half of the actual feed rate to the interval.

To arrive at a feed rate to the size interval from coarser sizes, a "conservation of numbers" hypothesis was used. This is based on the nature of abrasion breakage, that is, size reduction by detachment of fine material from
the surface of a central core. In steady-state abrasion, the rate at which cores enter a size interval from above is simply the sum of the rates that cores are fed to those larger sizes. The mass flow rate into the receiving size interval is the number of particles entering per unit time multiplied by the mass per particle at the top of the interval.

Thus the residence time in a size interval is

\[
\text{Residence time in nth interval = } \frac{\text{Mass in nth interval}}{1/2 \text{ mass/unit time fed to interval} + E} \quad (10)
\]

where \( E \) = mass flow rate from the coarser intervals and can be calculated as the rate of particles entering all coarser sizes, times the mass at the top of this interval. The denominator of Equation (10) includes both the material which will be abraded away and the cores. In order to adjust the residence time for the cores alone, it is multiplied by the ratio of mass of a particle entering the size fraction to the mass of a particle of average size (the geometric mean of the upper and lower limit), that is, \( 2^{3/4} \) raised to the 3/4 power.

Stanley presented data for the pebble wear rate versus size calculated for a typical test. This data showed that
the wear rates were a function of mass for the larger sizes down to a certain limit, where they became a function of surface area. The size at which this transition occurs was constant for the data presented. This size must be a function of material strength and density, and mill size and speed, and could be expected to be constant for the tests reported. Stanley then suggests that if the wear rate at which this volume to surface area transition occurs can be calculated, the wear rates at all other sizes down to the abrasion limit can be calculated.

The equation presented was derived from the test data and has only been shown to apply for this particular ore and mill. It was a regression analysis of the factors that determine the wear rate at the transition size and is:

Wear rate at 53648 μm (kg/hr) =

\[ 0.0071 \times \left( \frac{500}{\text{Mass of dry load (kg)}} \right)^{0.71} \]

(11)

It is worth noting that wear rate is inversely proportional (to the power of 0.71) to the mass of the load, that is, as the mass of the load decreases, the wear rate increases. Stanley explains: "This is a phenomenon well known to pebble mill operators and can be explained as follows. The mill load can be imagined to be an epicyclic
gear running on the inner surface of the mill; the smaller
the mill load, the faster it will rotate and the higher
will be the rate of pebble wear." [Another point worthy of
note is the use of circuit feed to determine the coarseness
of the load. As discussed earlier, the size distribution
of the load will be known at convergence and could have
been used instead.]

The conversion of wear rates to breakage rates is done
as follows. At steady state,

\[
\text{Mass/time entering size interval} = \text{Mass/time leaving} \text{ (12)}
\]

Assuming there is no discharge from the segment,

\[
\text{Mass/time entering} = \text{number of pieces in interval } \times q \text{ (13)}
\]

where \( q = \) the total mass loss from the interval per
particle per unit time, or \( q = \) the rate at which material
is abraded from an average particle plus the rate at which
mass is removed from the interval by the passing of
particles to the next lower size interval.

In the matrix model,

\[
\text{Mass/time leaving} = (1-b_1) \times R \times \text{mass in interval} \text{ (14)}
\]
where \( b_1 \) = the first element of the breakage distribution function applicable to the size interval concerned, that is, the proportion that stays in the parent size interval after breakage; \( R \) = breakage rate for this size interval.

Therefore,

Number of particles in size interval \( \times q = \)

\[(1-b_1) \times R \times \text{mass in size interval} \quad (15)\]

Since each entering particle totally disappears from the interval at the end of its residence time, the factor \( q \) is simply the mass of the particle at the top of the size interval divided by the residence time in the interval.

Therefore,

\[
\frac{\# \text{ of particles} \times \text{particle mass at top}}{\text{Residence time}} = (1 - b_1) \times R \times \text{mass in size interval} \quad (16)
\]

And since,

\[
\frac{\# \text{ of particles in size interval}}{\text{Mass in interval}} = \frac{\text{Mass in interval}}{\text{Average mass/particle in interval}} \quad (17)
\]

then,
Particle mass at top
\[
\frac{\text{Particle mass at top}}{\text{Average mass/particle } \times \text{ residence time}} = (1 - b_1) \times R \quad (18)
\]

Since for \(\sqrt{2}\) intervals the ratio of the particle mass at the top of the fraction to the particle mass of an average particle is 1.706 (2 to the 3/4 power),
\[
\frac{1.706}{\text{Residence time}} = (1 - b_1) \times R \quad (19)
\]

Stanley states that it can be shown that for mass-dependent particle wear, the residence times in successive size intervals, the ranges of which are of constant ratio to each other, are equal, while for surface-dependent particle wear, the residence times in successive size intervals of constant ratio to each other are in the same ratio as the intervals.

It might be helpful to explain how this is shown. Residence time is defined as mass lost in the interval divided by wear rate. Mass lost by abraded particles is the mass of a particle entering the size from above minus the mass of the particle as it leaves. If wear rate is also a function of mass, then residence time is constant because the effect of size cancels out (because mass is a function of size cubed). If wear rate is a function of surface (or size squared), then residence time has a factor
of size left in it and therefore varies with size.

Hence, from equation (19), it follows that the breakage rate function will be constant for all sizes in which mass-dependent wear occurs, and will increase exponentially with decreasing size in the surface-dependent wear region until the crushing limit is reached. Thus, if the breakage rate of any size can be determined and the transition point between the two types of wear is known, the breakage rates for all sizes above the shatter limit can be calculated.

Equations (7), (8), (11) and (19) can be combined to give the breakage rate for Stanley's tests in the 53648 μm size interval, which is the change-over point from mass- to surface-dependent wear. The equation is shown below.

\[
R(53648) = \frac{0.0213 \times \text{circuit feed} \% + 53648 \times \left(\frac{500}{\text{Load mass}}\right)^{0.71}}{(1 - b_1)}
\]

(20)

The R values for all sizes coarser than 53648 μm are then the same as that for 53648 μm, while for smaller sizes down to the shatter limit, the values are calculated from the relationship \( R_n = \sqrt{2} \times R_{n-1} \), where \( n \) = number of size interval, increasing with decreasing size.
Stanley also states that, because the studies of particle wear rates showed that there were two types of abrasion breakage, the breakage distribution function shown in Figure 10 was inadequate. He believed that it was necessary to develop separate breakage distribution functions to describe mass- and surface-dependent abrasion.

Stanley suggests that mass-dependent abrasion is chipping breakage or breakage that results from the breaking off of edges or corners from a crack network confined to a relatively thin layer on the surface of the parent particle, and that the abrasion breakage distribution function shown in Table 3 adequately describes this type of breakage. This author agrees with this conclusion, but would suggest that the cracks and chipping breakage are caused by the higher impact forces generated by the large particles. The edges and corners Stanley refers to are not evident above the shatter limit, except in freshly crushed ore. This will be further discussed in the next section.

The surface-dependent abrasion appeared to be a whittling away of surface grains by friction, which would produce a product size distribution that is constant regardless of the size of the particle. Stanley states that "the breakage [distribution] function would again show
the bulk of the parent [particles] remaining in the original size interval and a mathematically related proportion appearing in the next smaller fraction, but the detritus would appear in the same, considerably smaller, size intervals regardless of the size of the original [particles]." Thus, the basic breakage distribution function describing surface-dependent abrasion was the same as that for mass-dependent abrasion, except that the model provided for the retention of the lower terms (the detritus products) of the function in the same size intervals, regardless of the intervals in which the top terms appeared.

The final form of the breakage distribution function, which is shown in Figure 12, included four parts: mass-dependent abrasion and surface-dependent abrasion as discussed above, the transition zone and the shatter zone.

As indicated by the dotted lines in Figure 11, the shatter breakage portion of the breakage rate curve can be modeled by two intersecting straight lines of slope +1 and -1. This was true for all of Stanley’s tests, and indicates that the probability of shatter breakage increases with decreasing size to a maximum, and thereafter decreases with decreasing size. The only confirmation for this model of the breakage rate was from data which Stanley
<table>
<thead>
<tr>
<th>Weight-dependent abrasion breakage</th>
<th>Surface-dependent abrasion breakage</th>
<th>Transition zone</th>
<th>Shatter breakage</th>
</tr>
</thead>
<tbody>
<tr>
<td>a 0 0 0 0 0 0</td>
<td>0 0 0 0 0 0</td>
<td>0 0 0 0 0 0</td>
<td>0 0 0 0 0 0</td>
</tr>
<tr>
<td>b a 0 0 0 0</td>
<td>0 0 0 0 0 0</td>
<td>0 0 0 0 0 0</td>
<td>0 0 0 0 0 0</td>
</tr>
<tr>
<td>c b a 0 0 0</td>
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</tr>
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<tr>
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</table>

**Figure 12** - General Scheme of the Breakage Distribution Function
Four-part Matrix
collected from a larger (5.1 meter diameter x 5.2 meter) mill, which also showed this trend but with slopes of +1.38 and -1.38. In order to support his hypothesis, Stanley discussed the relationship between the shatter breakage rate and load size distribution. He suggested that if the rate of removal of material (breakage rate [R] times material in size [s]) over a number of size fractions were constant and, of course, the entry rates were constant, then the breakage rates would be inversely proportional to the contents. He then presented data which showed that the mill contents demonstrated a minimum corresponding to the breakage-rate maximum at the relevant sizes.

Stanley then suggested that the problem of modeling the shatter breakage-rate function in autogenous mills comprises three elements:
1) determining the size at which the breakage-rate maximum occurs,
2) determining the magnitude of the breakage rate at the maximum, and
3) determining the slopes of the curves adjacent to the maximum.

Testwork indicated that the size at which the breakage-rate maximum occurred was a function of circuit feed size, and was given by
\[
\log_{10}\text{(size (\mu m) at breakage rate maximum)} = \\
2.63 + 0.014 \times \text{(circuit feed \% +107296 \mu m)} \quad (21)
\]

[This is yet another example of Stanley using circuit feed sizing to calculate a parameter which is a function of mill content sizing.]

The magnitude of the breakage-rate maximum was found to be controlled by three factors:

1) the net mill energy input per unit mass of ore fed to the mill (new feed plus recirculating load),

2) the mass of dry load (presumably as a measure of the "epicyclic gear" effect suggested in the discussion of pebble wear), and

3) the concentration, in the pebble portion of the circuit feed, of a "most effective" pebble size.

[It should be noted that the use of total mill feed as the denominator for the net mill energy input calculation disregards the form of the recirculating load. The case for using total mill feed is that it will, when considered with mill contents, provide a measure of residence time. And once again, Stanley has used the concentration (in this case of a "most effective" pebble size) in circuit feed to calculate a parameter instead of the same concentration in
mill contents which will be known to the model at convergence.

The relationship presented by Stanley and used in the model was:

Maximum shatter breakage rate =

\[
35.3 \times \text{kw-hr/tonne total feed} \times \left( \frac{500}{\text{Load mass (kg)}} \right) \\
\times \left( \frac{\text{Circuit feed \%} - 151712 + \text{53648 \text{\mu m}}}{\text{Circuit feed total \%} + \text{53648 \text{\mu m}}} \right)
\]  

(22)

Finally, the slopes of the breakage-rate curve adjacent to the peak were assumed to be +1 and -1, and simulation of the shatter breakage-rate curve was complete. This was combined with the abrasion breakage rates to give a prediction of the entire breakage rate function. In the transition zone, the rates used were those calculated assuming shatter breakage only. The reason for ignoring the abrasion breakage rate in the transition zone was not explained by Stanley, but the rate function as calculated does approximate experimental results satisfactorily.

Discharge function

The problem of simulating the discharge function of a non-overflow (grate-discharge) mill can be reduced to two elements:
1) determining the maximum rate of slurry flow through the grate. This flow rate is called $D_{\text{max}}$. And
2) determining the maximum particle size for which this value extends. This maximum particle size is called size $D_{\text{max}}$.

The discharge function for particles between size $D_{\text{max}}$ and the aperture size of the grate is given by a curve (suggested by Stanley) of the form

$$D(i) = D_{\text{max}} \frac{(x-b)^2(2x-3a+b)}{(b-a)^3}$$

(23)

where $x = \log \text{ size } i$,

$a = \log \text{ size } D_{\text{max}}$, and

$b = \log \text{ mill discharge grate aperture size}$.

Any particle larger than the grate aperture has a discharge rate of 0.

Analysis of Stanley's test results showed that $D_{\text{max}}$ was controlled by two factors:

1) The pulp density of the mill discharge (decreasing linearly with increasing density), and
2) the size distribution of the load.

Size $D_{\text{max}}$ depended only on the pulp density of the discharge, increasing with increasing density.

[In this instance, Stanley has used the size
distribution of the load as a parameter-determining factor.

It appears that Stanley considered only one size of discharge grate, because it is reasonable to suggest that both \( D_{\text{max}} \) and size \( D_{\text{max}} \) are a function of grate size. An attempt was made to alter the equations to include grate size variation in the model. This will be discussed later.

The prediction equations proposed by Stanley for the discharge functions were:

\[
D_{\text{max}} = 130 \left( \frac{\text{Mass of load} + 37929 \ \mu m}{\text{Mass of load} + 4741 \ \mu m} \right) - 56.9 \\
- 0.897 \ (\text{discharge \% solids} - 70) 
\]  

(24)

and

\[
\log_{10}(\text{size } D_{\text{max}}) = \\
(0.044 \times \text{discharge \% solids}) - 0.294 
\]

(25)

Stanley comments that the dependence of maximum discharge rate on pulp density and load size distribution indicates (at least for low-discharge grate mills) that the pulp has to find its way though the mass of grinding media to reach the discharge grate, and is not discharged from some sort of classifier pool\(^{[45]}\) extending along the "toe" of the cascading load. This implies that the bulk of the pulp is located in the interstices of the load. The
The significance of the ratio of the load in size fractions in the $D_{\text{max}}$ calculation is that it is in effect a distribution modulus and, as such, is an index of the permeability of the pulp in the load, or of the ease with which the pulp can flow through the load. Also, the pulp density or apparent viscosity determines the flow of pulp through the media to the grate. The presence of pulp density in the size $D_{\text{max}}$ equation is expected because denser pulps can suspend coarser particles. Thus, increasing the pulp density will have the effect of increasing the residence times of some of the particles (mainly finer ones) and decreasing those of others (mainly coarser particles).

The next section of Stanley's work discusses further relationships required by the model. The calculation of breakage rates involves the mill energy input per unit mass of mill feed, and mill energy input depends on the mass of the load. This means that there needs to be a relationship between mass of load and mill power. The net mill power for each test (that is, the indicated power from kilowatt-hour meter less empty mill power) was plotted against mass of dry load and the following relationship was derived.
Net mill power (kw) =

\[ 12.1 \log_{10} \text{mass of dry load (kg)} - 26.4 \]  \hspace{1cm} (26)

This relationship is only valid for loads in the range of 152 to 1000 kilograms. This relationship was also found to be inadequate and was modified. This will be discussed later.

Another problem in the use of the model was that the predicted load mass was too high at low pebble wear rates, and vice versa. In order to correct this problem the following change was made. For -75856 +53648 \mu m ore, Equation (19) can be rewritten as

\[ R = 3 \times \text{Wear rate} / (1-b_1) \]  \hspace{1cm} (27)

where \( R \) is wear rate in kg/hr and \( b_1 \) is the first term in this size's column of the breakage function.

In order to correct for the inaccuracies mentioned, the equation was modified to

\[ R = \frac{3 \times \text{Wear rate} + 1.2 \times (\text{Wear rate} - 0.082)}{(1 - b_1)} \]  \hspace{1cm} (28)

The basic model description is now complete.
Modification of the Model to Account for Crushed Ore
Content of Circulating Load

As has already been discussed, the application of an impact to an ore particle should produce a surface containing micro-cracks or flaws which will allow it to abrade or chip away at a higher rate than an "aged", smoother surface. The method of modeling the breakage of material which has undergone impact is to assume that "fast" abrasion is a result of surface properties. New material presented to a mill will be completely covered with this "damaged" surface. Material that has been in a mill environment for some period of time will have been rounded and the damaged surface layer will have been removed. When rounded material is passed through a crusher, new surface areas containing flaws are created.

The first major assumption is that all material fed to the crusher has been rounded, and hence contains no particles containing fast-breaking surfaces. In reality, and as calculated using the perfect-mixing model, some material will leave the mill still in the fast-breaking, unrounded state, but for ease of computation this material will be considered to be in the rounded state on leaving the mill.

Secondly, any material below the abrasion limit (that
is, the size below which abrasion in the mill will have no significant effect) will not receive any benefit from the crushing process. This is probably not true, as the same micro-cracks which make a large particle abrade quicker would certainly make a smaller particle shatter at a higher rate. However, the effect of crushing on these particles should be small, and this assumption should not affect the results significantly.

Thirdly, all the rounded surface area, in crusher feed material above the abrasion limit, will appear in crusher discharge material above the abrasion limit. This means that new surface area generated by the crusher is calculated by subtracting the surface area of crusher feed material larger than the abrasion limit from the surface area of the crusher discharge material larger than the abrasion limit. This assumption will tend to underestimate the amount of new surface area produced, because the surface of the ore in contact with crusher liners should be reduced to a size much smaller than the abrasion limit (typically about 3000 μm).

In order to arrive at a mass of material which is fast-breaking, it is necessary to estimate the thickness of the breakage layer. This is a variable in the model and the effects of variation in this parameter will be
discussed later.

The calculation of surface area of a particle was derived from Stanley's estimation of particle mass shown in equation (7) and repeated here for convenience:

\[ \text{Pebble mass (grams)} = \frac{d^3}{496.53} \]  

(29)

where \( d \) is the geometric mean of the passing and retaining screen aperture sizes in millimeters. It can also be written that

\[ \frac{1}{6} \pi d^3 \rho \psi_v = \frac{d^3(1000)}{496.53} \]  

(30)

where \( d \) is defined as before except in centimeters, \( \rho \) is the specific gravity of the ore (3.1 for Stanley's ore) and \( \psi_v \) is the shape factor. For the ore that Stanley measured, (specific gravity of 3.1 g/cm\(^3\)) \( \psi_v = 1.24 \).

Assuming the shape factor for surface area should be the volume shape factor raised to the 2/3 power, the surface area shape factor (\( \psi_s \)) becomes 1.15. Hence,

\[ \text{Particle surface area (cm}^2) = \pi d^2 \psi_s = 3.63 \, d^2 \]  

(31)

The average size of a particle in an interval is generally regarded as the geometric mean of the upper and
lower limits of that interval, or

\[ S_m(i) = [S(i) \times S(i+1)]^{0.5} \quad (32) \]

where \( S(i) \) and \( S(i+1) \) are the upper and lower limits of the size interval, respectively. Now, the total surface area (in square centimeters) of several particles in size \( i \) which total \( W(i) \) grams is

\[
\text{Total Surface} = \frac{W(i) \times 496.53 \times (S_m(i)/10000)^2 \times 3.63}{(S_m(i)/1000)} \quad (33)
\]

where \( S_m(i) \) is in microns. This is simply dividing the gross mass by the average mass per particle and multiplying by the average surface area per particle. This then reduces to

\[
\text{Total surface area} = \frac{W(i) \times 18024}{S_m(i)} \quad (34)
\]

It is now possible to calculate the total surface area of material by size fraction going into and coming out of the crusher. Once this is known, the new surface area for
each size fraction can be calculated by making two
assumptions. First, as previously stated, it is assumed
that all of the old surface area of the crusher feed is
contained in the crusher discharge in sizes above the
abrasion limit. Second, it is assumed that, in the crusher
discharge, the new surface area of each size larger than
the abrasion limit is distributed in the same proportion as
the total surface area of that size. In other words, if
TSA_f is the total surface area in crusher feed larger
than the shatter limit, and TSA_d is the total surface
area in the crusher discharge larger than the abrasion
limit, then the ratio of new surface area to total surface
area in the crusher discharge in size i can be defined as

\[
\text{Ratio of new to total surface area in crusher discharge for size } i = \frac{(TSA_d - TSA_f)}{TSA_d} \quad (35)
\]

for size i greater than the abrasion limit.

If the thickness of the surface layer that is fast
breaking is T_f and the total surface area is fast
breaking, then the mass of fast-breaking material in a
particle of median size having a completely "new" surface
is the mass at S_m(i) minus the mass at (S_m(i) - 2 \times T_f). This is mathematically stated as,
The mass of fast-breaking material in size i is given by:
\[
\text{Mass of fast-breaking material in size } i = \frac{S_m(i)^3 - [S_m(i) - 2 \times T_f]^3}{496.53}
\]  
(36)

and the average proportion of fast-breaking to total material in any size fraction is the ratio of the average masses:
\[
\text{Ratio of fast-breaking material to total material in size } i = \frac{S_m(i)^3 - [S_m(i) - 2 \times T_f]^3}{S_m(i)^3}
\]  
(37)

If only a fraction ω of the surface area is fast-breaking, then the mass of fast-breaking material is ω times the total calculated in equation (36). Hence, in the case of the crusher discharge, the mass of fast-breaking material is the mass of fast-breaking material calculated assuming all surface is fast-breaking, times the ratio of new to total surface area calculated in equation (35).

Once the mass of fast-breaking material from the crusher plus the new feed is known, the proportion of fast-breaking material in the total mill feed can be calculated.
A mass balance at steady-state using the perfect mixing model yields

"Fast-breaking" balance:

\[ \phi_i f_i = \sum_{j=i+1}^{n} b_{ij} R_{fj} \phi'_j s_j + d_i \phi'_i s_i \]  

(38)

"Slow-breaking" balance:

\[ (1 - \phi'_i) f_i + \sum_{j=1}^{i-1} b_{ji} R_{sj} s_j = \sum_{j=i+1}^{n} b_{ij} R_{si} (1 - \phi'_i) s_i + (1 - \phi'_i) d_i s_i \]  

(39)

where \( \phi'_i \) = proportion of feed that is fast breaking,

\( \phi'_i \) = proportion of material in size \( i \) at steady-state that is fast breaking,

\( f, b, s, d \) and \( R \) are the feed vector, the breakage distribution function, the mill contents vector, the mill discharge function and the breakage rate vector, respectively, as defined in the previous section,

\( R_{fj} \) is the fast-breakage rate for size \( i \),

\( R_{sj} \) is the slow-breakage rate for size \( i \),

\( i \) is the number of the size range of interest,

\( n \) and \( l \) are the numbers of the smallest and largest size ranges of interest, respectively.

The reason that the second term in the left-hand side
of the "slow-breaking" balance has no "s" or "f" subscript is that the calculation will establish a single average steady-state breakage rate.

Solving the "fast-breaking" balance for \( \phi'_i \) gives

\[
\phi'_i = \frac{\phi_i f_i}{(1 - b_{ij}) R_{fj} s_i + s_i d_i} \tag{40}
\]

since

\[
\sum_{j=i+1}^{n} b_{ij} R_{fj} s_i = (1 - b_{ii}) R_{fi} s_i \tag{41}
\]

Once \( \phi'_i \) is known, the effective breakage rate can be calculated by applying a weighted average such as

\[
R_i = \phi'_i R_{fi} + (1 - \phi'_i) R_{si} \tag{42}
\]

One problem is that \( R_{fi} \) (and \( R_{si} \)) are unknown. However, if the ratio of \( R_{fi} \) and \( R_{si} \) can be estimated and, at a standard condition \( R_i \) can be calculated, then it is possible to write \( R_{fi} \) and \( R_{si} \) in terms of \( R_i \). The model would then calculate a value of \( R_i \) for the conditions in the mill (based on circuit-feed distribution and the dry mass of mill load, as per Stanley) and then calculate \( R_{fi} \) and \( R_{si} \) in terms of \( R_i \). Then the calculation of \( \phi'_i \) and the recalculation of \( R_i \) are possible (now taking into account the amount of
fast-breaking material present in the mill). With no crusher present, the model will not recalculate a value of $R_j$.

As very little data concerning fast and slow breakage rates for a 1.6 meter mill was available, data by Austin[37][41] was used which indicated that the following relationship would be valid:

$$R_{f1} = 16 \times R_{s1}$$  \hspace{1cm} (43)

The model was run to establish standard conditions which will be used in the following calculations. It was assumed that the thickness of the "fast-breaking" layer was one millimeter. In the example calculation which follows, the value of $R_j$ is calculated at 53648 \(\mu\text{m}\) ($S_m = 63798 \mu\text{m}$) so the proportion of fast-breaking material in the new feed from equation (37) is:

$$\phi = \frac{(S_m/1000)^3 - (S_m/1000 - 2)^3}{(S_m/1000)^3} = 0.09113$$ \hspace{1cm} (44)

The percentage of new circuit feed greater than 53648 \(\mu\text{m}\), and dry mill load for this standard run, were 28.7% and 502.3 kg, respectively. Hence, from equation (11), the
wear rate (in kg/hr) at 53648 μm is

\[
\text{Wear rate} = 0.0071 \times (28.7) \times \left( \frac{500}{502.3} \right)^{0.71} = 0.2031 \tag{45}
\]

and the value of \( R_i \) (hr\(^{-1}\)) was calculated as

\[
R_i = 300 \times \text{wear rate} + 120 \times (\text{wear rate} - 0.082) = 75.46 \tag{46}
\]

Since the discharge grate was set at 3353 μm, the value of \( d_j \) for +53648 -75856 μm material is zero and the value of \( s_j \) is equal to the mill contents (502.3 kg) times the fraction of +53648 -75856 μm material in the mill (.1646). The new mill feed was 515 kg/hr, and 8.6% of the feed material was +53648 -75856 μm, so the fast-breaking material fed to the mill is \( \phi \times 515 \times .086 \), or 4.04 kg/hr. The value of \( b_{ij} \) in this case is 0.01.

From equation (40):

\[
\phi_i' = \frac{4.04}{(0.01) \times R_{f1} \times (.1646 \times 502.3)} = \frac{4.88}{R_{f1}} \tag{47}
\]

and

\[
R_i = \phi_i'R_{f1} + (1 - \phi_i') R_{s1} = \frac{4.88 + \frac{R_{f1}}{16}}{16} - \frac{4.88}{16} = 75.46 \tag{48}
\]
Hence, under standard conditions $R_{fi} = 15.03 \times R_i$, $R_{si} = 0.939 \times R_i$ and $\phi'_i = 0.004$. It is now clear that the assumption to ignore the amount of fast-breaking material being discharged from the mill is reasonable, as less than a half a percent of the material discharged in this size would be fast-breaking. This calculation was repeated for all sizes down to the abrasion limit, and is included in Table C1 presented in Appendix C. Due to the fact that the ratio $R_{fi}/R_i$ changed with particle size, a regression of the values obtained was done and is also shown in Table C1.

It is interesting to note the difference between the two values $\phi_i$ and $\phi'_i$. While $\phi_i$, the proportion of fast-breaking material entering the mill, is high (over 9% in this case), the proportion in the mill at steady state is very low (less than half a percent). This difference is larger than a straight ratio of breakage rates (16 to 1) would indicate.

One final equation is required before the model is complete. The calculation of $\phi_i$ (a value needed for each size greater than the abrasion limit) was done by adding the amount of fast-breaking material in the feed to the amount of fast-breaking material in the crusher discharge and dividing by the total material fed to the
mill. The total material fed to the mill for each size is known to the model. The fast-breaking material in the mill feed is calculated by multiplying the feed rate of material in a size fraction by the ratio of fast-breaking to total material in that size as calculated by equation (37). The amount of fast-breaking material in a size fraction in the crusher discharge is calculated by multiplying the amount of material in that size in the crusher discharge by the ratio of new surface area, calculated by equation (35), times the ratio of fast-breaking to total material as calculated by equation (37).

The calculations required to make the alterations to the model are now complete. The model was amended in this way:

1) Equation (32) was used to calculate the average particle size.
2) Equation (34) was used twice to calculate surface area into and out of the crusher. The values for all sizes greater than the abrasion limit were totalled.
3) Equation (35) was used with the totals in 2) above to calculate the average ratio of new surface area to total surface area in the crusher discharge.
4) Equation (37) was used to calculate the ratio of fast-breaking to total material for each size (assuming
a totally fast-breaking surface).

5) $\phi_i$ was calculated using equation (44).

6) $\phi_i'$ was calculated using equation (40), the value of $R_{f1}/R_i$ for this size, and the value $R_i$ as calculated by the model.

7) The abrasion breakage rate was recalculated using equation (42). For sizes in the abrasion-to-shatter transition zone, only the portion of the rate that is attributable to abrasion was adjusted according to equation (42). This proportion was determined in the same way as in the breakage distribution function calculation. In other words, the largest size in the abrasion-to-shatter transition zone was 6/7 abrasion and 1/7 shatter, while the smallest size in the transition zone was 1/7 abrasion and 6/7 shatter. These modifications are included in the program presented in Appendix B.

The Screen Model

The model selected for the screening operation was taken from Whiten[65], modified slightly according to Lynch[66]. A brief description is given here for completeness.

The screen model was developed from simple
probabilistic considerations and parameters were found by fitting the model to experimental data. The probability that a particle of size $S$ does not pass through a screen with aperture size $h$ and wire diameter $d$ is

$$E(S) = \left\{ 1 - \left( \frac{h - S}{h + d} \right)^2 \right\}^m$$

(49)

where $m$ is the number of attempts at passing through that a particle has on crossing the screen. This expression represents the performance curve for the screen: the quantity $(h-S)^2$ represents the area within an aperture that a particle can pass through without hitting the edge of the aperture. The quantity $(h+d)^2$ represents the unit area of the screen containing the aperture. The number of attempts $m$, is proportional to:

- an efficiency constant, $k_1$,
- the length of the screen, $l$, and
- a load factor, $f$.

Hence,

$$m = k_1 \times l \times f$$

(50)

The load factor will be unity for low feed rates and tend to zero for very high feed rates. Whiten found no evidence in his test work that the load factor varied with feed rate so he assumed that it was unity (until data from heavily
loaded screens could be analyzed). Lynch suggested that the load factor and efficiency constant could be combined, and provided data which showed the combined load and efficiency factors as a function of feed rate for a particular screen. The screens were six feet wide, ten feet long and had an aperture size of 1/2 inch. It was decided that this data would be used in the present work, as the 1/2 inch gap was reasonable for this circuit. The data used is shown in Figure 13, and was modeled by the equation

\[ k_1f = 23.4 - 0.104T + \frac{3.13T^2}{10000} - 1983/T + \frac{8.45 T \ln(T)}{1000} \]  

(51)

where \(T\) is the screen feed rate in tonnes per hour. For feed rates below 300 tonnes per hour, a constant load factor of 2.85 was used. For feed rates above 600 tonnes per hour a load factor of 1.26 was used. But since the mill model calculates the number of screens required, there should be no load effect problem for large variations in load.

To use this model for the calculation of the amount of ore in individual size intervals, an average value of the screen efficiency is required for each size fraction. The unweighted average value is
Figure 13 - Efficiency and Load Factor Curve for the Screen Model
\[ \int_{S_1}^{S_2} E(S) \, dS / (S_2 - S_1) \quad (52) \]

and the approximation suggested by Whiten

\[ E(S) = e^{-m \left( \frac{S - h}{h + d} \right)^2} \quad (53) \]

may be used. Putting this approximation into equation (52) and making the substitution

\[ y = \sqrt{m} \left( \frac{S - h}{h + d} \right) \quad (54) \]

gives

\[ \frac{h + d}{\sqrt{m}} \left[ \int_{y_1}^{y_2} e^{-y^2} \, dy \right] / (S_2 - S_1) \quad (55) \]

This integral is an error function and can be approximated by the following, as proposed by Hart et al[67]:

\[ \int_{-\infty}^{\infty} e^{-y^2} \, dy = \frac{0.124734}{y^3 - 0.437880y^2 + 0.266982y + 0.138375} \quad (56) \]

This model provides an adequate description of the screen behavior, except for the undersize material (minus about 420 microns according to Whiten) of which a fraction
k_3 is expected to go to the oversize. He suggests a value of k_3 of 0.10, but says that for wet ore the value could be much higher. Since this screen is being used ahead of a hydrocyclone, it is assumed that there will be sufficient water available to wash the large particles, thus reducing the misplaced fines to a very low value. Hence, the misplaced fines were neglected and the efficiencies were simply calculated according to the equation. This means that the very small particles have a very good chance of passing through the screen.

The model works as follows. The factor m is calculated by multiplying the efficiency and load factor calculated in equation (51) by the square-root of the screen length (in meters). If the particles are larger than the screen aperture size, the screen efficiency (defined as the proportion of material passing through the screen) is set to zero. If the size range spans the screen aperture size, the program calculates the amount of material that is smaller than the aperture size (r_h) with the equation

\[ r_h = \frac{h - S_{i+1}}{S_i - S_{i+1}} \]  

where h is the aperture size and S_i and S_{i+1} are the upper and lower size limits that span h. Then the screen
efficiency is calculated as shown below for the size interval \( h \) to \( S_{j+1} \), and multiplied by \( r_h \), thus in effect weighting the result.

For size ranges smaller than the aperture size, the factors \( y_1 \) and \( y_2 \) are calculated according to equation (54), and Hart's approximation is used to evaluate the error function. The function is then normalized to give a screen efficiency. The screen section of the program begins on line 1000 in Appendix B.

The Crusher Model

This part of the simulation also comes from Whiten's work[65]. Whiten developed a simple model of the crusher. He suggested that a more complicated model would require more parameters, which could not be accurately estimated with the limited amount of test data he possessed. The basis of this model is that particles may be either broken, or passed through the crusher unbroken. The broken particles then have the same chance as new particles of the same size of being broken further, or of passing through the crusher. Hence, the crusher can be simplified to a single breakage zone and a probability of entering or reentering this zone. Figure 14 shows the symbolic parts of the crusher, and the flows between the
parts.

Figure 14 - Symbolic Representation of Crusher Model

The vectors \( f \), \( x \) and \( p \) give the flow rates in each size fraction. The lower-triangular breakage matrix \( B \) gives the relative distribution in each size fraction after a particle is broken, and the diagonal classification matrix \( C \) gives the proportion of particles entering the breakage region.

The mass balances at the nodes of Figure 14 give

\[
f + BCx = x \quad (58)
\]

and

\[
x = Cx + p \quad (59)
\]
Eliminating $x$ gives

$$p = [1-C] \cdot [1-BC]^{-1} \cdot f \quad (60)$$

which expresses the crusher product in terms of the feed. The matrix $1-BC$ is always non-singular because it is lower triangular, and has no zero elements in the diagonal because a unit element on the diagonal of $BC$ would imply both no breakage and no discharge of that size fraction.

The breakage function that best explains Whiten's data consists of two parts. The first part is a step matrix (that is, a matrix which has columns that are identical except that the columns are transposed downward one row at each successive column; the bottom row contains all the residual material not allocated in the upper portions) $B_1$, which gives the product sizes relative to the size of the original particle. This is calculated from the Rosin-Rammler distribution modified by Broadbent as discussed in the Autogenous Mill section

$$p(S) = (1 - e^{-(S/S')}^u) / (1 - e^{-1}) \quad (61)$$

where $S'$ is the size of the original particle and $p(S)$ is the fraction less than size $S$. The value of $u$ proposed by Whiten was 6 for the cone crusher. This was used in the model.
The second part of the breakage matrix, $B_2$, describes the production of fines. This portion of the product is not dependent on the size of the original particle. That is, the proportion of material that breaks into size $i$ is the same for all sizes larger than size $i$. The program ensures that any material predicted to break into a size greater than the initial size is included in the initial size. Whiten selected the Rosin-Rammler equation to describe this material

$$p(S) = 1 - e^{-(S/S'')}^v$$

(62)

where $S''$ is a fixed size. The values calculated for the parameters on this distribution are $S'' = 3.06$ mm and $v = 1.25$.

Finally, to arrive at an overall breakage matrix, $B$, the two component matrices are added,

$$B = aB_1 + (1-a)B_2$$

(63)

where $a$ is calculated from the gap size in millimeters, $g$, as follows:

$$a = 0.872 + 0.00453 \times g$$

(64)
Whiten points out that the calculated breakage function indicates that the crusher produces a small number of large particles plus a few percent of fines. This corresponds to cleavage breakage as shown in Figure 2.

The diagonal elements of the classification function $C$ are derived from a function of particle size, $c(S)$, which gives the probability that a particle of size $S$ will enter the breakage stage of the crusher model. It is assumed that particles below a certain size $k_1$ will pass through the crusher without being broken. That is,

$$c(S) = 0 \text{ for } S < k_1$$  \hspace{1cm} (65)

Also there is a size $k_2$ above which the particles will always be broken, or,

$$c(S) = 1 \text{ for } S > k_2$$  \hspace{1cm} (66)

Between these two extremes $c(S)$ is assumed to be a parabola with zero gradient at $k_2$, so that

$$c(S) = 1 - \left[\frac{(S - k_1)}{(k_2 - k_1)}\right]^2 \quad k_1 < S < k_2$$  \hspace{1cm} (67)

The elements in the $C$ matrix are calculated as the mean
values of c(S) in the appropriate size range. For the range \( S_1 \) to \( S_2 \) \((S_2 > S_1)\) the matrix element is

\[
\int_{S_1}^{S_2} c(S) dS / (S_2 - S_1)
\]  

(68)

This equation becomes in this case

\[
C_1 = 1 - \frac{(S_2 - k_1)^3 - (S_1 - k_1)^3}{3 (k_2 - k_1)^2 (S_2 - S_1)}
\]  

(69)

The two parameters \( k_1 \) and \( k_2 \) (mm) are predicted by

\[
k_1 = 0.67 \times g
\]  

(70)

and

\[
k_2 = 1.121 \times g + 58.674 \times q + F(T)
\]  

(71)

where \( g \) is the crusher gap in millimeters, \( q \) is the fraction of crusher feed that is +25.4 mm and \( T \) is the feed rate (t/h) to the crusher. Whiten suggested that the function \( F(T) \) should be a natural spline function of degree three[68] through the points

\[(100,-1.234),(250,-2.159),(400,-6.579)\]
The following equation was found to be suitable.

\[ F(T) = -7.767 \times 10^{-5} T^2 + 2.102 \times 10^{-2} T - 2.560 \quad (72) \]

The relation for \( k_1 \), the size below which no crushing occurs, as a function of the crusher gap, is as would be expected. The relation between \( k_2 \), the size above which crushing must occur, and the crusher gap is reasonable. The remaining terms in the \( k_2 \) equation relate to the ease of flow through the crusher. Large particles and low tonnages flow through the crusher more rapidly.

The Hydrocyclone Model

The model used to simulate the hydrocyclone unit operation was from Lynch[66].

The premise of Lynch’s model is that after a reduced performance curve has been fitted to a specific set of data, that curve shape will hold for most other conditions. A performance curve is a graph of the fraction of feed material reporting to underflow (or overflow) versus particle size. The corrected performance curve has the same x- and y-axes, but material which is considered to bypass the separation process has been mathematically removed. The reduced performance curve is the same as the
corrected performance curve, except that the x-axis is a dimensionless size, each size having been normalized by dividing by the corrected $d_{50}$ size (the size on the corrected performance curve at which 50% of the material reports to the underflow).

The variables used in the model are for water split or the proportion of water that reports to overflow and underflow, the corrected $d_{50}$ and the shape, or degree of "flatness", of the reduced performance curve. The water split determines the amount of bypass (the material that does not participate in the separation process) and is used to calculate the actual performance curve under the conditions which exist. It is assumed that fine particles follow the water in some proportion that remains constant over the operating range of the hydrocyclone. The proportion of water in underflow is:

$$ R_f = \frac{W_F - W_{OF}}{W_F} \quad (73) $$

with the water in underflow calculated by the empirical equation:

$$ W_{OF} = 1.07 \times W_F - 3.94 \times Spig + K_1 \quad (74) $$
where WF and WOF are the water in the feed and the overflow in tonnes per hour, respectively, Spig is the spigot diameter in centimeters, and $K_1$ is a constant. Once $K_1$ is determined, the water split can be found for any set of operating conditions. These two equations can be combined to give:

$$R_f = -0.07 + 3.94 \times \frac{\text{Spig}}{WF} - \frac{K_1}{WF} \quad (75)$$

$R_f$ is supposed to represent the proportion of fines that report to the underflow as well as the proportion of water that reports to underflow. In the data used, these proportions were not equal: the proportion of water to underflow was 0.304 and the proportion of fines to underflow 0.419. It is assumed by the model that the fines will follow the water to the underflow in the same ratio of 0.419 to 0.304 or 1.38. So the equation for the ratio of fines reporting to underflow becomes:

$$R_f = -0.0966 + 5.436 \times \frac{\text{Spig}}{WF} + \frac{K_2}{WF} \quad (76)$$

where $K_2$ is a constant calculated from the data.

The $d_{50}(c)$ is the size of particle that undergoes separation (i.e. not included in bypass), that will have an
equal chance of reporting to overflow or underflow. This can be calculated by using the following equation:

\[
\log d_{50}(c) = 0.0173 \times \text{FPS} - 0.0695 \times \text{Spig} \\
+ 0.013 \times \text{VF} + 0.000048 \times Q + K_3
\]  

(77)

where FPS is feed percent solids, Spig is the spigot diameter in centimeters, VF is the vortex finder diameter in centimeters, Q is the volumetric flow rate of the feed in liters per minute, and \( K_3 \) is a constant. Hence, the position of the reduced performance curve can be determined for any operating condition.

The form of the reduced performance curve can be represented in different ways. The one that was selected for this model is:

\[
y_i = 1 - e^{(-0.6931x_i^n)}
\]  

(78)

where \( y_i \) is the fraction of feed of size \( i \) reporting to underflow, \( x_i \) is the geometric mean of the range of the particle size \( i \) divided by \( d_{50}(c) \), and \( n \) is a constant determined to fit a set of sampled data.

The sampled data for this hydrocyclone model was taken from appropriate plant data[69]. Table 6 contains the
Table 6 - Hydrocyclone Sample Data

<table>
<thead>
<tr>
<th>Size (μm)</th>
<th>Feed</th>
<th>Underflow</th>
<th>Overflow</th>
</tr>
</thead>
<tbody>
<tr>
<td>+4760</td>
<td>0.2</td>
<td>0.6</td>
<td>0.0</td>
</tr>
<tr>
<td>-4760 +2380</td>
<td>3.1</td>
<td>4.1</td>
<td>0.2</td>
</tr>
<tr>
<td>-2380 +1190</td>
<td>11.0</td>
<td>14.0</td>
<td>4.0</td>
</tr>
<tr>
<td>-1190 + 590</td>
<td>19.8</td>
<td>23.8</td>
<td>14.4</td>
</tr>
<tr>
<td>-  590 + 297</td>
<td>17.8</td>
<td>19.9</td>
<td>17.0</td>
</tr>
<tr>
<td>-  297 + 149</td>
<td>14.0</td>
<td>13.3</td>
<td>16.3</td>
</tr>
<tr>
<td>-  149 + 74</td>
<td>8.0</td>
<td>6.5</td>
<td>10.7</td>
</tr>
<tr>
<td>-    74</td>
<td>26.1</td>
<td>17.8</td>
<td>37.4</td>
</tr>
</tbody>
</table>

Percent solids - 65.0 35.0

data as provided.

Data as collected usually will not be totally consistent; there is a need to correct the data in such a way that it is consistent and all the data is given proper consideration. The data in Table 6 is not consistent in that there is no single value of overflow and underflow split with which each of the size range data will mass balance.

The data was corrected using a method proposed by Lynch in a way that minimized the error and the hydrocyclone constants were calculated. The equations used in the model are:

\[ R_f = -0.0966 + 5.436 \times \text{Spig} / \text{WF} + 17.55 / \text{WF} \]  

(79)
\[
\log d_{50}(c) = 0.0173 \times FPS - 0.0695 \times Spig \\
+ 0.013 \times VF + 0.000048 \times Q + 2.40 \tag{80}
\]

and

\[
\left(\times 0.935 \right) \\
y_i = 1 - e^{-x_i} \tag{81}
\]

The hydrocyclone section of the model begins in line 4000 of Appendix B.

**Further Relationships Required**

Once the models had been combined and debugged, it became clear that a few things were not working properly.

The way in which the peripheral parts of the model (the screen, crusher and hydrocyclone) responded to changes in flow rate proved to be a problem. There were instances where the model would cycle as the scaling factors for the unit processes would switch from one integer to another and back again. In addition, the performance of the units varied considerably and tended to mask the differences caused by changes to the configuration and other parameters. It was decided that in order to remove these
variations, the model should be able to provide for constant performance of the unit processes, regardless of feed rate. This is reasonable in that, for a given operation, the unit processes could be designed to operate in the manner dictated. The variation in results from one simulation run to another will then be caused only by the changes in the circuit configuration and the response of the autogenous-mill model to them.

Another problem was in the discharge function described by Stanley. It was clear that his model was for a mill with only one grate aperture size; there was no provision for variation of the grate aperture and hence the discharge function, except for determining the particle size above which there would be no discharge. In order to account for grate aperture variation, the particle size to which the maximum discharge rate extends (size $D_{\text{max}}$) was factored by:

$$1 + \log \left( \frac{\text{grate aperture}}{3353} \right)$$  \hspace{1cm} (82)

where grate aperture is in microns. The value 3353 is the grate aperture used by Stanley. In this way, some of the effect of grate aperture is accounted for. It should be noted that there is no experimental evidence for this
relationship. It is, however, consistent with operating experience.

The maximum discharge rate ($D_{\text{max}}$) from the mill may vary with grate aperture, but Stanley's argument that the permeability of the load limits the discharge rate is considered to be valid. However, the maximum discharge rate should vary with mill loading. As the mill load increases, the area presented to the grate (or the area available for permeation) will also increase. Hence, the value calculated for $D_{\text{max}}$ is factored by the proportion of actual mill load to a full mill load, as the mass of the load (proportional to the volume of the load) is proportional to the area of the grate which is faced by the load.

Another problem was in the calculation Stanley proposed for power draw to the mill. He mentioned that it increases indefinitely with mill load and has a zero value at a load of 152 kg. Data collected from the current testwork and shown in Table 5 was used to estimate mill power draw as a function of mill loading. This was modeled by:

\[
\% \text{ Power draw} = 306.7 - 44.37 \times L - .152 \times L^2 \\
- 754 / L + 12.3 \times L \times \log L \quad (83)
\]
where \( L \) is the percentage of volume of the mill that contains load. It should be noted that this equation only applies for loads above 4%.

Notes on the Model Convergence

The method used to determine a solution is:

1) assume a mill content size distribution and tonnage

2) assume no recirculating loads

3) calculate the model parameters (the B, R and D matrices) using the current values of mill feed and mill contents

4) calculate (new) mill discharge and new mill content tonnages and distributions using the perfect mixing model

5) calculate the tonnages and size distributions for (a) screen feed, overflow and underflow, (b) crusher feed and discharge, and (c) hydrocyclone feed, overflow and underflow, using the respective models

6) combine the recirculating loads with new feed and recalculate the size distribution

7) check for convergence:
   yes - stop
   no - go to step 3)
The model will iterate as described until one of three things may occur.

1) The mill may become totally full of material. During normal start up, the mill contents will increase, with resulting power draw increase, to a point where the power needed to provide enough breakage for that feed rate will be reached. However, there is a level of mill content above which an increase will cause the power draw to decrease. As the power draw decreases, the breakage rate decreases and the mill will fill up with recirculating load which has not been reduced enough in size to become final product. If this occurs, a message will be displayed informing the operator that the feed rate is too high for the conditions under which the mill is operating.

2) The program may take too many iterations attempting to converge. A limit on the number of iterations allowed (set at 50) will prevent the model from reaching a solution that has an abnormally large recirculating load. This occurs at high feed rates (but not high enough for the mill to fill up). The model increases the feed to the hydrocyclones in an effort to increase the amount of overflow (final product). However, this increase in feed to the hydrocyclones is composed
mainly of coarse material which, for the most part, reports to the underflow and is returned to the mill, thus increasing the recirculating load. If allowed to continue, the result would be, in many cases, the mill filling up.

3) Finally, the iterations may converge. The convergence criteria are that, within 50 iterations, and without the mill filling up, the final product flow rate shall be within 0.1-0.2% of the feed rate and the breakage rate of the 53648 µm material does not change by more than 0.3% relative from one iteration to the next.

The convergence criteria were the result of trying many different criteria which did not achieve satisfactory convergence values.
RESULTS OF THE SIMULATION

Having incorporated the changes detailed in the "Further Relationships Required" section into the model, it was ready to be used. The variables that were tested were studied under both configurations (conventional autogenous and autogenous with crusher) for a range of feed rates. The parameters varied were grate aperture size, screen aperture size, crusher gap, surface damage thickness and feed size distribution.

Some of the results are tabulated in Tables 7 to 13, at the end of this section, followed by Figures 15 to 18, which show these results graphically.

At this time, it is necessary to introduce a calculated variable called an operating work index. The use of a work index to classify ore grindability has been used for years since it was first introduced by Bond[70]. The work index is defined as:

$$W_i = \frac{W}{\sqrt{P} - \sqrt{F}}$$  \hspace{1cm} (84)

where \(W\) is the work input in kilowatt hours per short ton, \(P\) is the diameter in microns which 80% of the product passes, and \(F\) is the diameter in microns which 80% of the feed passes. Under identical mill conditions, the
grindability of different ores could be compared using this work index.

If the work input to a mill for a particular configuration, and a measure of the feed and product sizes could be obtained, a work index for that particular case could be calculated. This is called the operating work index. A comparison of different configurations (using the same ore) would then be possible. The lower the operating work index, the "better" the performance of the configuration. In this way a single value can be used to evaluate performance, taking into account power and size reduction. The operating work index was used when comparing the various simulation runs.

The results indicate that:

1. The advantage of using an autogenous-with-crusher (AC) circuit does not appear to be significant at low feed rates, but at higher feed rates the benefit, in terms of the operating work index, is substantial.

2. The AC circuit allows the treatment of a higher tonnage of material before becoming unstable and filling up.

3. A coarser feed (defined here as increasing the coarse
fractions and decreasing the finer fractions of the feed) is not detrimental to autogenous mill operation, either with or without the crusher. This paradox can be explained by the fact that the increase in coarse particles in the feed creates a better working charge.

4. Conversely, a finer feed creates a problem at high throughput due to the deterioration of the charge.

5. An increase in grate aperture size had a detrimental effect on the conventional circuit operation. The final product became coarser and the capacity decreased. However, in the AC circuit, the increase in grate size had a beneficial effect. While the final product became slightly coarser, more of the critical-sized material could be removed and treated by the crusher. This had the effect of reducing the mill content and power draw, allowing a higher treatment rate, and lowering the operating work index.

6. The size of the screen apertures had very little effect on either circuit. What little effect there was could be explained by the hydrocyclone operation; the more coarse material presented to the hydrocyclone, the
coarser the overflow became. In the AC circuit, having screen apertures larger than the crusher gap increased the mill contents and power draw slightly.

7. The crusher gap was shown to be very important to the AC circuit performance, especially at high throughput and large grate aperture size. A smaller crusher gap had the effect of increasing final product fineness, reducing mill content and power draw, and substantially lowering the operating work index.

It should be noted here that the critical particle size for this mill is much smaller than for a larger mill. The crusher gap required to reduce material through the critical size for this mill (about 1/4 in) is smaller than would be considered practical from an operating point of view. The size of the grate aperture and the crusher gap in practice would depend on the size and speed of the operating mill and ore characteristics.

8. The assumption about surface damage thickness was tested by recalculating the values of \( R_f/R \) with the damage thickness halved, as shown in Table C2 in Appendix C. It was found that these values did not
change significantly. The model was modified to account for the changes and rerun with the new reduced thickness of damage. The values at low feed rates were very similar. At higher feed rates, the use of the smaller damage thickness resulted in an operating work index that was about 3% higher than for the larger thickness, indicating a lower efficiency. This would reduce the benefit of using the crusher (in operating work index terms) by only about 10% (that is, from 32% better to 29% better).

This indicates that the benefit of having a crusher in an autogenous milling circuit is not associated to any great extent (at least in this model calculation) with surface damage. Probably 80% of the benefit results from breaking material through the slow-breaking critical-size range.

It should be noted that the operating work index used in this thesis does not have the same units as proposed by Bond. The values used herein are, however, comparable one to another and are used solely to evaluate the difference between one simulation run and another.
### Table 7 - Summary of Simulation Results -
Standard Ore (Feed 80% Passing = 76218 μm)

#### Conventional Autogenous Milling Circuit
Grate = 1/2 in; Screen gap = 1/4 in)

<table>
<thead>
<tr>
<th>Feed Rate (kg/hr)</th>
<th>Product 80% Passing Size</th>
<th>Product % -75 μm</th>
<th>Mill Contents (kg)</th>
<th>Recirculating Load (kg)</th>
<th>Operating Work Index</th>
</tr>
</thead>
<tbody>
<tr>
<td>300</td>
<td>39</td>
<td>90.58</td>
<td>162</td>
<td>63</td>
<td>1.13</td>
</tr>
<tr>
<td>350</td>
<td>60</td>
<td>83.94</td>
<td>212</td>
<td>86</td>
<td>1.26</td>
</tr>
<tr>
<td>400</td>
<td>96</td>
<td>74.84</td>
<td>279</td>
<td>122</td>
<td>1.58</td>
</tr>
<tr>
<td>450</td>
<td>151</td>
<td>63.70</td>
<td>366</td>
<td>176</td>
<td>2.14</td>
</tr>
<tr>
<td>500</td>
<td>246</td>
<td>48.56</td>
<td>503</td>
<td>277</td>
<td>3.11</td>
</tr>
</tbody>
</table>

#### Autogenous Milling Circuit with Crusher
Grate = 1 1/2 in; Screen gap = 1/4 in; Crusher gap = 1/4 in
Surface area damage thickness = 1 mm

<table>
<thead>
<tr>
<th>Feed Rate (kg/hr)</th>
<th>Product 80% Passing Size</th>
<th>Product % -75 μm</th>
<th>Mill Contents (kg)</th>
<th>Recirculating Load (kg)</th>
<th>Operating Work Index</th>
</tr>
</thead>
<tbody>
<tr>
<td>300</td>
<td>40</td>
<td>90.22</td>
<td>154</td>
<td>87</td>
<td>1.15</td>
</tr>
<tr>
<td>350</td>
<td>63</td>
<td>83.26</td>
<td>192</td>
<td>129</td>
<td>1.26</td>
</tr>
<tr>
<td>400</td>
<td>102</td>
<td>73.37</td>
<td>235</td>
<td>195</td>
<td>1.50</td>
</tr>
<tr>
<td>450</td>
<td>166</td>
<td>61.29</td>
<td>283</td>
<td>289</td>
<td>1.88</td>
</tr>
<tr>
<td>500</td>
<td>251</td>
<td>48.24</td>
<td>337</td>
<td>420</td>
<td>2.36</td>
</tr>
<tr>
<td>550</td>
<td>391</td>
<td>35.32</td>
<td>401</td>
<td>616</td>
<td>3.11</td>
</tr>
<tr>
<td>600</td>
<td>734</td>
<td>22.65</td>
<td>481</td>
<td>1003</td>
<td>4.56</td>
</tr>
</tbody>
</table>
Table 8 - Summary of Simulation Results -
10% Coarser Ore (Feed 80% Passing = 83748 µm)

<table>
<thead>
<tr>
<th>Feed Rate (kg/hr)</th>
<th>Product 80% Passing Size</th>
<th>Product Contents % -75 µm (kg)</th>
<th>Mill Work Index</th>
<th>Recirculating Load (kg)</th>
<th>Operating Work Index</th>
</tr>
</thead>
<tbody>
<tr>
<td>300</td>
<td>39</td>
<td>90.63</td>
<td>63</td>
<td>1.13</td>
<td></td>
</tr>
<tr>
<td>350</td>
<td>59</td>
<td>84.07</td>
<td>209</td>
<td>86</td>
<td>1.24</td>
</tr>
<tr>
<td>400</td>
<td>95</td>
<td>75.08</td>
<td>273</td>
<td>120</td>
<td>1.55</td>
</tr>
<tr>
<td>450</td>
<td>149</td>
<td>64.17</td>
<td>357</td>
<td>173</td>
<td>2.08</td>
</tr>
<tr>
<td>500</td>
<td>234</td>
<td>50.04</td>
<td>481</td>
<td>263</td>
<td>2.94</td>
</tr>
</tbody>
</table>

Autogenous Milling Circuit with Crusher

<table>
<thead>
<tr>
<th>Feed Rate (kg/hr)</th>
<th>Product 80% Passing Size</th>
<th>Product Contents % -75 µm (kg)</th>
<th>Recirculating Load (kg)</th>
<th>Operating Work Index</th>
</tr>
</thead>
<tbody>
<tr>
<td>300</td>
<td>40</td>
<td>90.26</td>
<td>153</td>
<td>1.15</td>
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<tr>
<td>350</td>
<td>62</td>
<td>83.37</td>
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<td>128</td>
</tr>
<tr>
<td>400</td>
<td>101</td>
<td>73.66</td>
<td>233</td>
<td>191</td>
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<td>450</td>
<td>163</td>
<td>61.29</td>
<td>280</td>
<td>281</td>
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<td>500</td>
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<td>380</td>
<td>35.93</td>
<td>396</td>
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<td>634</td>
<td>24.11</td>
<td>473</td>
<td>896</td>
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Table 9 - Summary of Simulation Results -
30% Coarser Ore (Feed 80% Passing = 96810 µm)

### Conventional Autogenous Milling Circuit

<table>
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<tr>
<th>Feed Rate (kg/hr)</th>
<th>Product % Passing Size</th>
<th>Product % -75 µm</th>
<th>Mill Contents (kg)</th>
<th>Recirculating Load (kg)</th>
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</tr>
</thead>
<tbody>
<tr>
<td>300</td>
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<td>1.15</td>
</tr>
<tr>
<td>400</td>
<td>87</td>
<td>77.04</td>
<td>241</td>
<td>101</td>
<td>1.39</td>
</tr>
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<td>450</td>
<td>136</td>
<td>66.74</td>
<td>311</td>
<td>140</td>
<td>1.79</td>
</tr>
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<td>3.63</td>
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### Autogenous Milling Circuit with Crusher

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<th>Feed Rate (kg/hr)</th>
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<th>Product % -75 µm</th>
<th>Mill Contents (kg)</th>
<th>Recirculating Load (kg)</th>
<th>Operating Work Index</th>
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<td>519</td>
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<td>430</td>
<td>763</td>
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Table 10 - Summary of Simulation Results -
10% Finer Ore (Feed 80% Passing = 69619 \mu m)

<table>
<thead>
<tr>
<th>Feed Rate (kg/hr)</th>
<th>Product 80% Passing Size</th>
<th>Product -75 \mu m</th>
<th>% Mill Contents (kg)</th>
<th>Recirculating Load (kg)</th>
<th>Operating Work Index</th>
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<td>3.15</td>
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<table>
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<th>Product -75 \mu m</th>
<th>% Mill Contents (kg)</th>
<th>Recirculating Load (kg)</th>
<th>Operating Work Index</th>
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Table II - Summary of Simulation Results - 30% Finer Ore (Feed 80% Passing = 59809 μm)

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<th>Mill Contents (kg)</th>
<th>Recirculating Load (kg)</th>
<th>Operating Work Index</th>
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</thead>
<tbody>
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Conventional Autogenous Milling Circuit

Autogenous Milling Circuit with Crusher

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<th>Feed Rate (kg/hr)</th>
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<th>Product % -75 μm</th>
<th>Mill Contents (kg)</th>
<th>Recirculating Load (kg)</th>
<th>Operating Work Index</th>
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Table 12 - Results of Simulation
% -74 μm in Final Product for Various Operating Settings

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<tr>
<td>Crusher Gap (mm)</td>
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<td>-</td>
</tr>
<tr>
<td>Feed Rate (kg/hr)</td>
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Table 13 - Results of Simulation
Steady-State Mill Contents for Various Operating Settings

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<th>Autogenous with Crusher</th>
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</thead>
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<tr>
<td>12.7</td>
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<td>--</td>
</tr>
<tr>
<td></td>
<td>337</td>
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</table>
Figure 15 - Conventional Milling Circuit Simulation Results - Operating Work Index

- □ = Standard Ore
- ○ = 10% Coarser Ore
- ● = 10% Finer Ore
- ◊ = 30% Coarser Ore
- ♦ = 30% Finer Ore
Figure 16 - Autogenous Mill with Crusher Circuit Simulation Results - Operating Work Index
Figure 17 - Conventional Milling Circuit Simulation Results - Product Size Passing 80%
Figure 18 - Autogenous Mill with Crusher Circuit Simulation Results - Product Size Passing 80%
SUMMARY AND CONCLUSIONS

The use of an external crusher in an autogenous milling circuit appears to be justified both in terms of increased capacity and energy savings. The use of a crusher for increased control of the circuit is also possible, but only at the cost of running sub-optimally in order to provide for a variation in tonnage crushed.

A study of the mechanisms of autogenous grinding has provided insight into the operation of a circuit to best utilize the energies being generated by the mill. The use of balls or an external crusher have the same effect of creating new surfaces by cleavage or impact breakage. These new surfaces break more quickly in an abrasive environment than surfaces which have been smoothed by continued exposure to abrasion. In addition, the product of abrasion of this new surface tends to be coarser than the product of abrasion of smoothed surfaces, thus reducing the amount of overground material being generated. The reason an external crusher should be preferred to the use of balls in the mill is that there are substantial maintenance and material cost savings arising from reduced mill liner wear and ball costs.

The removal and size reduction by crushing of the
slow-breaking, critical-sized, particles has three advantages. First, the generation of new surfaces by crushing increases the breakage rate of these daughter fragments which are returned to the mill. Secondly, the size reduction itself generates particles that have been broken through the slow-breaking critical size to a size that is much more amenable to shatter breakage in the mill. Finally, there is evidence[48] that the mill operates more effectively with this middle size removed; there is a better interaction between the coarse and fine particles when middle-sized particles are removed.

The circuit simulation, after alterations corrected various problems, provided some insight into what factors affect operation of the proposed circuit.

The size distribution of the feed to the mill was not shown to be a very important parameter. For a coarser feed size distribution, the extra energy required to reduce the generally larger material to an acceptable product size was offset by a more efficient mill operation.

The grate aperture size was important to the efficiency both with and without a crusher in the circuit. With no crusher in the circuit, an increase in grate aperture size caused the final product to became coarser, the capacity to decrease and more power was required per ton of ore
treated. This would indicate that the smaller the grate aperture, the more efficient the mill operation would be. This is clearly not true in the extreme of zero grate aperture size and would indicate that the grate discharge part of the model is not totally adequate. With the crusher in the circuit, an increase in grate aperture size enabled more of the slow-breaking material to exit and be treated by the crusher, thus increasing mill capacity while reducing power required per ton. This was not found to be true at Similkameen[28], where an increase in grate aperture size decreased the capacity of the mill. This was due, in this author's opinion, to the fact that the ore was not competent enough to withstand the forces in the mill, and that the larger grate apertures resulted in larger ore being removed for crushing, thus depleting the amount of media even more.

The size of the screen apertures had little effect on the operation of the circuit with or without the crusher.

The crusher gap was very important to the operation of the circuit. A smaller crusher gap increased final product fineness and reduced power per ton of ore treated. There are obviously some operating limits to the size of the crusher gap.

The effect on the model of the thickness of surface
damage generated by crushing (and hence the amount of fast-breaking material) was not great. This would indicate that the majority of the benefit from having a crusher in the circuit results from breaking material through the slow-breaking critical-size range.

In conclusion, the use of a crusher in an autogenous milling circuit seems to be justified both by reduced operating costs and increased capacity. If a crusher can be used to facilitate autogenous operation instead of introducing balls into the mill, substantial material and maintenance cost savings can also be realized.
SUGGESTIONS FOR FUTURE WORK

As with most works of this kind, more questions end up being asked than answered. In addition, gaps in the literature have been revealed which need to be filled by future research. A few of the questions encountered in this research are discussed below.

There is need for much more work to determine exactly what rock properties are necessary for autogenous grinding to be feasible. While tests are currently being developed to determine some measure of media competency, very little is known about what size of mill (and speed of rotation) would be allowed by the ore to develop an adequate working charge. If the mill that the ore could tolerate is large enough to generate breakage at a fast enough rate, then autogenous milling should be considered.

The use of mill speed for grind control has shown some promise[50], and more work could be devoted to this problem. Plants that have operating mills with variable-speed drives could be at the forefront of this research.

Grate discharge modeling is another area where much research is needed. No model was found that could adequately describe the operation of a grate discharge.
This is especially disturbing because most models of autogenous and semi-autogenous mills contain a discharge grate section. Another reason that knowledge of grate discharge operation is important is that it could lead to greater insight into the actual contents of an operating mill, information which is vital to controlling the mill operation.

Another area of interest is the study of surface properties of freshly-broken and rounded ore. Possibly microscopic investigation can provide insight into the thickness of a damage layer, how it is generated, and what crushing process provides the largest amount of benefit. This could also be extended to determine how the use of high- or low-velocity explosives in underground or open-pit mining can control ore characteristics to smooth the operation of the milling circuit.
REFERENCES CITED


52. --------, "Autogenous Grinding Investigation on Amygdaloid Ore, Michigan Technical University, Project 17209, August and September 1971.


57. Wakeman, John S. and Mai, Ann, "Semi-Autogenous Grinding Performance with Large Diameter Grinding Balls"


69. Spottiswood, D. J., Private communication.

APPENDIX A

Test Data in Numerical and Graphical Form
Table A1 - Gross Weight in Each Size Fraction -
Autogenous Grinding Test Number 1

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</tr>
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</tr>
<tr>
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</tr>
<tr>
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</tr>
<tr>
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Figure A1 - Gross Weight in Each Size Fraction - Autogenous Grinding Test Number 1
Table A2 - Gross Weight in Each Size Fraction - Semi-autogenous Grinding Test Number 2

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Figure A2 - Gross Weight in Each Size Fraction - Semi-autogenous Grinding Test Number 2
Table A3 - Gross Weight in Each Size Fraction -
Semi-autogenous Test Number 3

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Figure A3 - Gross Weight in Each Size Fraction - Autogenous Grinding Test Number 3
### Table A4 - Gross Weight in Each Size Fraction - Semi-autogenous Grinding Test Number 4

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Figure A5 - Tracer Data for Large Pieces (+38100 μm)
Grinding Tests 1 and 2

Average Weight - % of Initial

Time (minutes)

= Autogenous Grind
= Grind with Balls
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### Table A8 - Tracer Data for Large Pieces (+38100 μm)
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Figure A6 - Tracer Data for Large Pieces (+38100 μm)
Grinding Tests 3 and 4

\[ \text{Average Weight} \times 100 \% \text{ of Initial} \]

- □ = Autogenous Grind
- ○ = Grind with Balls

Time (minutes)
Table A9 - Tracer Data for Large-Sized Ore - All Tests

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<th>Time (min):</th>
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Table A10 - Tracer Data for Medium-Sized Ore - All Tests

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Table All - Tracer Data for Dolomite - All Tests

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<td>3.9</td>
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<tr>
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<td>0.0</td>
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<td>0.5</td>
<td>0.0</td>
<td>6.1</td>
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<tr>
<td>-6680 +4760</td>
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<td>0.0</td>
<td>0.8</td>
<td>1.7</td>
<td>1.0</td>
<td>2.0</td>
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<tr>
<td>-4760 +3360</td>
<td>0.0</td>
<td>0.0</td>
<td>0.6</td>
<td>1.2</td>
<td>1.2</td>
<td>1.3</td>
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<tr>
<td>-3360 +2380</td>
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<td>0.0</td>
<td>0.3</td>
<td>0.5</td>
<td>0.7</td>
<td>0.6</td>
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<tr>
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### Test 4 - 6% Balls

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<th>10</th>
<th>30</th>
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<td></td>
<td></td>
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<tr>
<td>-25000 +22400</td>
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<td>853.0</td>
<td>765.8</td>
<td>720.0</td>
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<td>0.6</td>
<td>1.1</td>
<td>2.7</td>
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<tr>
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<td>0.0</td>
<td>0.0</td>
<td>1.3</td>
<td>1.6</td>
</tr>
<tr>
<td>-4760 +3360</td>
<td>0.0</td>
<td>0.0</td>
<td>0.2</td>
<td>0.0</td>
<td>0.2</td>
<td>0.6</td>
</tr>
<tr>
<td>-3360 +2380</td>
<td>0.0</td>
<td>0.0</td>
<td>0.2</td>
<td>0.0</td>
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<tr>
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<td>930.7</td>
<td>918.6</td>
<td>901.9</td>
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Autogenous Grind
Grind with Balls

Figure A7 - Tracer Data for Ore of Size -38100 +31500 μm
Grinding Tests 1 and 2
Figure A8 - Tracer Data for Ore of Size -31500 +25000 μm  
Grinding Tests 3 and 4
Figure A9 - Tracer Data for Ore of Size -25000 +22400 μm
Grinding Tests 3 and 4
Figure A10 - Tracer Data for Ore of Size -19000 +16000 μm
Grinding Tests 1 and 2

% Remaining in Topsize

\[
\begin{align*}
\square &= \text{Autogenous Grind} \\
\bigcirc &= \text{Grind with Balls}
\end{align*}
\]
Figure A11 - Tracer Data for Ore of Size -16000 +12500 µm
Grinding Tests 3 and 4
Figure A12 - Tracer Data for Dolomite of Size -6680 +4760 μm - Grinding Tests 1 and 2
Figure A13 - Tracer Data for All Traced Sizes - Autogenous Grinding Tests 1 and 3

- □ = -38100 +31500 microns
- ○ = -31500 +25000 microns
- ◊ = -25000 +22400 microns
- ■ = -19000 +16000 microns
- ■ = -16000 +12500 microns
- • = -6680 +4760 microns
Figure A14 - Tracer Data for All Traced Sizes -
Semi-autogenous Grinding Tests 2 and 4
APPENDIX B

Computer Program Listings
10 KEY OFF
20 ' MUSHER UNIT MODEL - TO DETERMINE THE EFFECTIVENESS OF A CRUSHER IN LINE
30 ' WITH AN AUTOGENOUS MILL. THE MILL MODEL IS DESCRIBED IN THE SAIMM J.
40 ' NOV. 1974, "MECHANISMS IN THE AUTOGENOUS MILL AND THEIR MATHEMATICAL
50 ' REPRESENTATION" BY G. G. STANLEY WITH MODIFICATIONS BY T. J. COMI
60 ' 
70 ' 
80 KEY OFF:COLOR 7,0:CLS
90 DIM A(26,26),D(26),R(26),MOP(26),MCP(26),MFP(26),MRP(26),MDT(26),MCT(26)
100 DIM TMFP(26),S(26),SCB(26),SC(10),CC(13),CY(16),M1(5),COL(5)
110 DIM CON(5),SFP(26),SE(26),CFP(26),CFP(26),P(26,26),CB(26,26),CB2(26,26)
120 DIM CDP(26),F(26),CI(26),X(26),Y(26),SGM(26),CYUP(26),CYOP(26),C(26,26)
130 DIM MIP(26),PHI(26),PHIP(26),APS(26),SATOCR(26),SAFRCR(26)
140 CON$="N"
150 999 GOTO 9000
160 1000 ' SCREEN SUBROUTINE
170 ' This section of the program calculates the split and size distribution
180 ' of screen products. The parameters SC(I) are input in an
190 ' program in the suite and are held in the file SCREEN.BAS. For a
200 ' complete description of the model see Whiten, 1972 cited in the
210 ' Musher Subroutine of this program. One alteration to Whiten's
220 ' method was from Lynch who in his book "Mineral Crushing and Grinding
230 ' Circuits" presented a curve for the efficiency and load factors which
240 ' has been modelled in this program in the form: EFF = A + B*T + C*T^2
250 ' + D/T + E*T*LN(T) where T is the feed rate in tons per hour. This
260 ' obviously assumes a screen width which is not specified in the text.
270 ' 
280 ' SC(1)=SCREEN HOLE NOMINAL SIZE (IN MILLIMETERS)
290 ' SC(2)=SCREEN WIRE NOMINAL DIAMETER (IN MILLIMETERS)
300 ' SC(3)=LENGTH OF SCREEN (IN METERS)
310 ' SC(4)=EFFICIENCY/LOAD FACTOR EFF FOR LOADS LESS THAN 300 TPH
320 ' SC(5)=A IN EFF(T)=A + B*T + C*T^2 + D/T + E*T*LN(T)
330 ' SC(6)=B IN
340 ' SC(7)=C IN
350 ' SC(8)=D IN
360 ' SC(9)=E IN
370 ' SC(10)=1 IF EFFICIENCY IS CONSTANT; 2 IF IT IS CALCULATED FROM FEED RATE
380 ' ********** CALCULATE M = SQR(# OF PRESENTATIONS TO THE SCREEN) **********
390 ' MD = TPH MILL DISCHARGE (= SCREEN FEED RATE)
400 ' IF SC(10)=1 AND PASS>1 THEN 1470 ' IF EFFICIENCY IS CONSTANT, SKIP CALC.
410 ' IF SC(10)=1 THEN SSCALE=1:SF=SF:SF = 400: GOTO 1310
420 ' IF CON$ = "Y" THEN 1280 ' SKIP SCALE CALC. WHEN MODEL HAS SETTLED DOWN
430 ' SSCALE=INT(SF/600)+1
440 ' SF=SF/SSCALE
450 ' IF SF < 300 THEN EFF=SC(4): GOTO 1320
460 ' IF SF > 600 THEN EFF=SC(4)/2.26: GOTO 1320
470 ' EFF=SC(5)+SC(6)*SF+SC(7)*SF^2+SC(8)/SF+SC(9)*SF*LOG(SF)
480 ' M=EFF*SQR(SC(3))
490 ' ********** DEFINE FUNCTION ERF(ESTIMATED BY HART. 1968 **********
500 ' DEF FNEF(Y)=.124734/(Y^3-.4378805*Y^2+.266892*Y+.138375)
510 ' ********** CALCULATE SCREEN EFFICIENCY FUNCTION SE(I) **********
520 ' WHICH IS THE PROPORTION TO UNDERSIZE
530 ' FOR I=1 TO 25
540 ' IF I=1 THEN SE(I)=1000:GOTO 1450
550 ' IF I=1 THEN SE(I)=1000 THEN K=(SC(1)+SC(2))*/1000/M/(SC(1)+1000-S(I+1))
560 ' Y=0:GOTO 1420
1400 \[ K = (SC(1) + SC(2)) \times 1000 / (S(I) - S(I+1)) \]
1410 \[ Y_1 = \frac{(S(I) - SC(1)) \times 1000}{SC(1)} \]
1420 \[ Y_2 = \frac{(S(I+1) - SC(1)) \times 1000}{SC(1)+SC(2)} \]
1430 \[ SE(I) = 1 - K \times (FNEF(Y1) - FNEF(Y2)) \]
1440 IF \( S(I) > SC(1) \times 1000 \) THEN \( R = (S(I) - SC(1) \times 1000) / (S(I) - S(I+1)) \)
1450 IF \( SE(I) < 0 \) THEN \( SE(I) = 0 \)
1460 NEXT I: SE(26) = SE(25)
1470 "******** CALCULATE THE TPH TO UNDERSIZE, OVERSIZE, TOTAL ***********
1480 SO = 0: SU = 0: SF = SSCALE ' SSCALE IS THE SCALING FACTOR
1490 IF SC(10) = 1 AND PASS = 1 THEN SF = SFE
1500 FOR I = 1 TO 26
1510 SU = SU + SF * MDP(I) / 100 * SE(I)
1520 SO = SO + SF * MDP(I) / 100 * (1 - SE(I))
1530 NEXT I
1540 "******** CALCULATE PRODUCT SIZE DISTRIBUTION AND F80 OF UNDERSIZE & OVERSIZE *
1550 TSUP = 0: TSUP = 0: SOF80 = 0
1560 FOR I = 1 TO 26
1570 CYFP(I) = SF * SFP(I) * SE(I) / SU
1580 TSUP = TSUP + CYFP(I): IF SOF80 = 0 THEN IF TSUP => 20 THEN SOF80 = I - 1: TSUP80 = TSUP
1590 CFP(I) = SF * SFP(I) * (1 - SE(I)) / SO
1600 TSO = TSO + CFP(I): IF SOF80 = 0 THEN IF TSO => 20 THEN SOF80 = I - 1: TSO80 = TSO
1610 NEXT I
1620 CYF = SU
1630 SOF80 = EXP(LOG(S(SOF80)) - (20 + CYFP(SUF80 + 1) - TSUP80) / CYFP(SUF80 + 1)
1640 CYFF80 = SUF80
1650 CF = SO
1660 SOF80 = EXP(LOG(S(SOF80)) - (20 + CFP(SOF80 + 1) - TSO80) / CFP(SOF80 + 1)
1670 CFF80 = SOF80
1680 RETURN
2000 "********** CRUSHER SUBROUTINE **************
2010 "This section calculates the crusher product size distribution *
2020 "based on the matrix equation: p = [I - C][I - BC]^-1 f. *
2030 "the variables CC(I) are the crusher parameters input in a *
2040 "different program in the suite and held in file CRUSHER.BAS. *
2050 "For a complete description of the model see: Whiten, W. J., *
2080 "********
2090 ' CC(1) = B1 DISTRIBUTION: U IN (1 - EXP(-(X/Y)^U)) / ((1 - 1/e)
2100 ' CC(2) = B2 DISTRIBUTION: Y IN 1 - EXP(-(X/Y)^V)) IN MM
2110 ' CC(3) = B2 DISTRIBUTION: V IN 1 - EXP(-(X/Y)^V))
2120 ' CC(4) = D IN A = 0 + E*G FOR CB = A.CB1 +(1-A).CB2
2130 ' CC(5) = E IN A = " " 
2140 ' CC(6) = G IN A = " " 
2150 ' CC(7) = A IN K1 = A*G FOR CLASSIFICATION PARAMETERS K1 AND K2 IN:
2160 ' C(S) = 1 - [(S-K2)/(K1-K2)]^2
2170 ' CC(8) = A IN K2 = [A*G + B*(FRAC. + 1 in. IN FEED)]* F(FEED RATE)
2180 ' CC(9) = B IN " " 
2190 ' CC(10) = A IN F(T) = A*T^2 +B*T + C
2200 ' CC(11) = B IN " " 
2210 ' CC(12) = C IN " " 
2220 ' CC(13) = 1 IF EFFICIENCY IS CONSTANT; 2 IF IT IS CALCULATED FROM FEED RATE
2230 "********* CALCULATE COARSE BREAKAGE FUNCTION CB1 ***********
2240 IF CC(13)=1 AND PASS>1 THEN 2900 ' SKIP CALC. IF EFFICIENCY TO BE CONSTANT
2250 FOR I=1 TO 25
2260 FOR J=1 TO 25
2270 P(J,I)=(1-EXP(-(S(J)/S(I))^CC(1)))/(1-EXP(-1))
2280 IF J>I THEN CB(J-I,1)=P(J-I,1)-P(J,1)
2290 NEXT J
2300 CB(26,1)=P(26,1)
2310 NEXT I
2320 ' * * * * * * * CALCULATE FINE BREAKAGE FUNCTION CB2 * * * * * * * * * * * * * * * * *
2330 FOR I=1 TO 26
2340 P(I,1)=1-EXP(-((S(I)/CC(2))/1000)^CC(3))
2350 IF I>1 THEN CB2(I-I,1)=P(I-I,1)-P(I,1)
2360 NEXT I
2370 CB2(26,1)=P(26,1)
2380 FOR I=2 TO 26
2390 FOR J=I TO 26
2400 IF J=I THEN CB2(J-I,1)=CB2(J-I,1)+CB2(J-I-1):GOTO 2420
2410 CB2(J,1)=CB2(J,1)
2420 NEXT J
2430 NEXT I
2440 CB2(26,26)=1
2450 ' * * * * * * * CALCULATE WIEIGHTING FACTOR, ALPHA * * * * * * * * * * * * * * * * * *
2460 ALPHA=CC(4)+CC(5)*CC(6)
2470 ' * * * * * * * CALCULATE OVERALL BREAKAGE FUNCTION CB * * * * * * * * * * * * * * * * *
2480 FOR I=1 TO 26
2490 FOR J=I TO 26
2500 CB(J,I)=ALPHA*CB(J,I)+ (1-ALPHA)*CB2(J,I)
2510 NEXT J
2520 NEXT I
2530 ' * * * * * * * CALCULATE CLASSIFICATION FUNCTION C * * * * * * * * * * * * * * * * * *
2540 ' * * * * * * * Calculate K1 - lower limit of classification * * * * * * * * * * * * * *
2550 K1=CC(6)*CC(7)
2560 ' * * * * * * * Calculate CQ - fraction of feed + 1 inch * * * * * * * * * * * * * * *
2570 CQ=0
2580 FOR I=1 TO 26
2590 IF S(I)<25400 THEN 2620
2600 CQ=CQ+CFP(I)/100
2610 NEXT I
2620 ' * * * * * * * Calculate K2 - upper limit of classification * * * * * * * * * * * * * *
2630 IF CC(13)=1 THEN CFE=400: CQ=0: GOTO 2660
2640 IF CF > 600 THEN CFE=600 ELSE CFE=CF
2650 IF CF < 300 THEN CFE=300 ELSE CFE=CF
2660 K2=(CC(8)*CC(6)+CC(9)*CQ)
2670 K2=K2+ (CC(10)*CFE^2+CC(11)*CFE+CC(12))
2680 IF K2<K1 THEN K2=K1*.5
2690 ' * * * * * * * CALCULATE CLASSIFICATION FUNCTION C1(I) * * * * * * * * * * * * * *
2700 FOR I=1 TO 26
2710 IF S(I)>K2*1000 THEN F(I)=S(I):GOTO 2740
2720 IF S(I)>K1*1000 THEN F(I)=(K1-K2)/3*K2*1000:GOTO 2740
2730 F(I)=S(I)-( (S(I)-K2*1000)*3)/(((K1-K2)*1000)^2)/3)
2740 NEXT I
2750 FOR J=1 TO 25
2760 C1(J)=(F(J)-F(J+1))/(S(J)-S(J+1))
2770 NEXT J
2780 C1(26)=0
2790 ' * * * * * * * CALCULATE CB*C (STORE IN CB) * * * * * * * * * * * * * * * * * *
2800 FOR I=1 TO 26
2810 FOR J=1 TO 26
2820 CB(J,I)=CB(J,I)*C(I)
2830 NEXT J,
2840 '******** CALCULATE I-CB*C (STORE IN CB) **************
2850 FOR I=1 TO 26
2860 FOR J=1 TO 26
2870 IF J=I THEN CB(J,I)=I-CB(J,I) ELSE CB(J,I)=-CB(J,I)
2880 NEXT J,
2890 '******** CALCULATE X SUCH THAT (I-CB*C)X=CF **************
2900 FOR I=1 TO 26
2910 X(I)=0
2920 FOR J=1 TO I-1
2930 X(I)=X(I)+X(J)*CB(I,J)
2940 NEXT J
2950 IF CB(I,I)=0 THEN X(I)=0: GOTO 2970
2960 X(I)=(C(I)-X(I))/CB(I,I)
2970 NEXT I
2980 '******** CALCULATE PRODUCT VECTOR AND F80 FROM THE CRUSHER **********
2990 CDP(26)=0;CDF80=0;TCDP80=0
3000 FOR I=1 TO 25
3010 CDP(I)=(1-C(I))*X(I)
3020 CDP(26)=CDP(26)+CDP(I)
3030 IF CDF80=0 THEN IF CDP(26)>=20 THEN CDF80=I-1;TCDP80=CDP(26)
3040 NEXT I
3050 CDF80=EXP(LOG(S(CDF80))-(20+CDP(CDF80+I)-TCDP80)/CDP(CDF80+1))
3060 CDP(26)=100-CDP(26)
3070 '**************************************************************************
3080 'CALCULATE THE FOLLOWING QUANTITIES:
3090 'SATOCR = SURFACE AREA OF FEED TO CRUSHER IN SIZE FRACTION I
3100 'SAFRCR = " " CRUSHER PRODUCT " " " 
3110 'TOSATOCR AND TOSAFRCR = TOTAL OF THE ABOVE FOR ALL SIZES >= ABLNO
3120 'SARATIO = RATIO OF NEW S.A TO TOTAL S.A FROM CRUSHER
3130 'FBRATIO = RATIO OF FAST-BREAKING TO TOTAL MATERIAL OF SIZE I
3140 'PHI(I) = THE PROPORTION OF FEED TO THE MILL THAT IS FAST BREAKING
3150 '**************************************************************************
3160 TOSATOCR=0:TOSAFCR=0
3170 FOR I=1 TO ABLNO
3180 APS(I)=SQR(S(I)^2*SQR(2))
3190 SATOCR=1.8024E+10/APS(I)^2*C(I)^2*C(I)/100
3200 SAFCR=1.8024E+10/APS(I)^2*C(I)^2*C(I)/100
3210 TOSATOCR=TOSATOCR+SATOCR
3220 TOSAFCR=TOSAFCR+SAFCR
3230 NEXT I
3240 IF TOSAFCR=0 THEN SARATIO=0:GOTO 3260
3250 SARATIO=(TOSAFCR-TOSATOCR)/TOSAFCR
3260 FOR I=1 TO ABLNO
3270 FBRATIO=((APS(I)/1000)^3-(APS(I)^2/1000-FT^2)*3)/(APS(I)/1000)^3
3280 PHI(I)=(FBRATIO*MF*MFP(I)/100+FBRATIO*SARATIO*C(I)/CDP(I)/100)
3290 IF PHI(I)>1 THEN PHI(I)=1
3300 NEXT I
3310 RETURN
This Subroutine calculates the split and size distribution of a hydrocyclone. The performance curve is based on the Plitt equation.


This Subroutine calculates the split and size distribution of a hydrocyclone. The performance curve is based on the Plitt equation. For a complete description of the model see Lynch, A. J., 1977, Mineral Crushing and Grinding Circuits, Elsevier Scientific Publishing Company, New York, pp 105-114.

CY(1) = CYCLONE FEED PERCENT SOLIDS
CY(2) = SPIGOTT DIAMETER (IN CENTIMETERS)
CY(3) = VORTEX FINDER DIAMETER (IN CENTIMETERS)
CY(4) = AVERAGE SPECIFIC GRAVITY OF ORE
CY(5) = NUMBER OF CYCLONES (WILL CALC. BASED ON FEED RATE IF SET TO 0)
CY(6) = K1 IN RATIO OF FINES TO U/F-RF=K1+K2*SP/H20 IN FEED+K3/H20 IN FEED
CY(7) = K2 IN
CY(8) = K3 IN
CY(9) = K1 IN LOG(D50C)=K1*FPS+K2*SP+K3*VF+K4*VOL. FLOW RATE+K5
CY(10) = K2 IN
CY(11) = K3 IN
CY(12) = K4 IN
CY(13) = K5 IN
CY (14) = N IN PLITT EQUATION: Y=1-EXP(-.693*(X/D50C)*N)
CY(15) = K1 IN WATER TO UNDERFLOW = RF/K1
CY(16) = 1 IF EFFICIENCY IS CONSTANT; 2 IF IT IS CALCULATED FROM FEED RATE
T1=0: T2=0  ' DUMMY VARIABLES
IF CY(16)=1 AND PASS>1 THEN 4540  ' SKIP CALC. IF EFFICIENCY IS TO BE CONST
CYF= TOTAL SOLID FEED TO CYCLONES
W*CYF*100/CY(1)-CYF  ' WATER IN CYCLONE FEED
SGM(1)= GEOMETRIC MEAN OF PARTICLE SIZE RANGES
FOR I=1 TO 26
CYFP(I)= FRACTION OF MATERIAL IN CYCLONE FEED
* IN EACH SIZE FRACTION (FROM MAIN PROGRAM)
NEXT I
FINES REPORT TO UNDERFLOW IN RATIO OF RF
CYFP(I)= FRACTION OF MATERIAL IN CYCLONE FEED
CY(16)=1 IF EFFICIENCY IS CONSTANT; 2 IF IT IS CALCULATED FROM FEED RATE

************ SCALE FOR NUMBER OF CYCLONES ************
IF CY(16)=1 THEN CYF1=300:W1=W*CYF1/CYF:GOTO 4430
IF CON$ = "Y" THEN 4400  ' SKIP SCALE CALC. WHEN MODEL HAS SETTLED DOWN
CY(15)=INT(CYF/400)+1
CYF1=CYF/CY(15)  ' TONS PER HOUR FEED (DRY) TO EACH CYCLONE
W1=W/CY(5)  ' TONS PER HOUR WATER TO EACH CYCLONE

************ CALCULATE CYCLONE FEED PERCENT SOLIDS ************
PI=CYF/(CYF+W)*100

************ CALCULATE R-SUB-F : PROPORTION OF FINES TO UNDERFLOW ************
RF=CY(6)+CY(7)*CY(2)/W1+CY(8)/W1
IF RF > 1 THEN RF=.5

************ CALCULATE D50(C) ************
D=CY(9)*PI+CY(10)*CY(2)+CY(11)*CY(3)
D=D+CY(12)*(W1*1000+CYF*1000/CY(4))/60+CY(13)
D=EXP(2.3025851*D)
FOR I=1 TO 26
4560 ******* CALCULATE FOR EACH SIZE FRACTION PROPORTION TO U/F *******
4570 IF SGM(I)=0 THEN 4600
4580 Y(I)=1-EXP(-(.6931472*(SGM(I)/D)^CY(I4)))
4590 GOTO 4610
4600 Y(I)=0
4610 Y(I)=Y(I)+(1-Y(I))*RF
4620 CYUP(I)=CYF*CYFP(I)*Y(I)/100
4630 CYOP(I)=CYF*CYFP(I)/100-CYUP(I)
4640 T1=T1+CYUP(I)
4650 T2=T2+CYOP(I)
4660 NEXT I
4670
4680 ******* CALCULATE % BY SIZE AND F80 OF UNDERFLOW AND OVERFLOW *******
4690 TCYUP=0:TCYOP=0:CYUF80=0:CYOF80=0
4700 FOR I = 1 TO 26
4710 CYUP(I)=CYUP(I)/T1*100
4720 TCYUP=TCYUP+CYUP(I)
4730 IF CYUF80=0 THEN IF CYUF80=20 THEN CYUF80=I-1:TCYUF80=TCYUP
4740 CYOP(I)=CYOP(I)/T2*100
4750 CYTOP=TCYOP+CYOP(I)
4760 IF CYOF80=0 THEN IF CYOF80>=20 THEN CYOF80=I-1:TCYOF80=TCYOP
4770 NEXT I
4780 CYUF80=EXP(LOG(S(CYUF80))-(20+CYUF80+1-TCYUF80)/CYUF80+1)
4790 CYOF80=EXP(LOG(S(CYOF80))-(20+CYOF80+1-TCYOF80)/CYOF80+1)
4800 CYU=T1
4810 CYO=T2
4820
4830 ******* ADJUST R-SUB-F TO PROPORTION OF FEED WATER TO U/F *******
4840 RFF=RF/CY(15)
4850 CYO.FPS=CYO*100/(CYO+W*RFF)
4860 CYO.FPS=CYO/100/(CYO+W*(1-RFF))
4870 RETURN
4880 '***** CALCULATION FOR THE CRUSHING LIMIT AND ABRASION LIMIT - NOTE: ****
4890 '***** CRUSHING LIMIT = 8 * ABRASION LIMIT *******
4900 X=0: I=0
4910 WHILE (X<2 OR I<2)
4920 I=I+1
4930 IF CONFIG=I THEN X=X+(TMF*TMFP(I)-CF*CFP(I))/(TMF-CF)
4940 IF CONFIG=2 THEN X=X+(TMF*TMFP(I)-CF*CDP(I))/(TMF-CF)
4950 IF I=26 THEN 5100
4960 IF X>5 THEN PRINT "INSUFFICIENT COARSE MATERIAL IN MILL FEED":I=2:X=5
4970 IF X>5 THEN X=5
4980 CON(2)=.32: CON(3)=.12: CON(4)=9.000001E-02: CON(5)=.06
4990 IF I > 5 THEN PRINT "INSUFFICIENT COARSE MATERIAL IN MILL FEED":I=2:X=5
5000 IF X>5 THEN X=5
5010 IF CONFIG=I THEN X=X+(TMF*TMFP(I)-CF*CFP(I))/(TMF-CF)
5020 IF CONFIG=2 THEN X=X+(TMF*TMFP(I)-CF*CDP(I))/(TMF-CF)
5030 IF I=26 THEN 5100
5040 WEND
5050 CON(2)=.32: CON(3)=.12: CON(4)=9.000001E-02: CON(5)=.06
5060 IF I > 5 THEN PRINT "INSUFFICIENT COARSE MATERIAL IN MILL FEED":I=2:X=5
5070 IF X>5 THEN X=5
5080 IF CONFIG=I THEN X=X+(TMF*TMFP(I)-CF*CFP(I))/(TMF-CF)
5090 IF CONFIG=2 THEN X=X+(TMF*TMFP(I)-CF*CDP(I))/(TMF-CF)
5100 IF I=26 THEN 5100
5110 IF X>5 THEN PRINT "INSUFFICIENT COARSE MATERIAL IN MILL FEED":I=2:X=5
5120 IF (CRLIM>S(I))/((S(I-1)-CRLIM) > 1 THEN 5180
5130 IF (CRLIM>S(I))/((S(I-1)-CRLIM) > 1 THEN 5180
5140 IF (CRLIM>S(I))/((S(I-1)-CRLIM) > 1 THEN 5180
5150 IF (CRLIM>S(I))/((S(I-1)-CRLIM) > 1 THEN 5180
5160 IF (CRLIM>S(I))/((S(I-1)-CRLIM) > 1 THEN 5180
5170 IF (CRLIM>S(I))/((S(I-1)-CRLIM) > 1 THEN 5180
5180 IF (CRLIM>S(I))/((S(I-1)-CRLIM) > 1 THEN 5180
5190 NEXT I
5500  '*** THIS SECTION CALCULATES THE BREAKAGE DISTRIBUTION MATRIX AS SPECIFIED
5510  '*** BY STANLEY IN THE SAIMM JOURNAL NOVEMBER 1974.
5520  IF CRLNO=CRLNOL THEN 5700
5530  FOR I=1 TO S:T=0
5540  FOR J=I TO 25
5550  A(I,J)=MAB(I,J):T=T+A(I,J)
5560  NEXT J: A(I,26)=1-T:NEXT I
5570  FOR I=6 TO CRLNO:T=0
5580  FOR J=I TO 25
5590  A(I,J)=MAB(I,J):T=T+A(I,J)
5600  NEXT J:A(I,26)=1-T:NEXT I
5610  FOR I=CRLNO+1 TO ABLNO:T=0
5620  FOR J=I TO 25
5630  A(I,J)=MAB(I,J)*(ABLNO-I+1)/7+MCB(J-I+1)*(I-CRLNO)/7:T=T+A(I,J)
5640  NEXT J:A(I,26)=1-T:NEXT I
5650  FOR I=ABLNO+1 TO 26:T=0
5660  FOR J=I TO 25
5670  A(I,J)=MCB(J-I+1):T=T+A(I,J)
5680  NEXT J:A(I,26)=1-T:NEXT I
5690  CRLNOL=CRLNO
5700  RETURN

6000  '******************************************************************************
6010  '*** CALCULATION OF THE D (DISCHARGE) MATRIX SECTION  
6020  '******************************************************************************
6030  '*** THIS SECTION CALCULATES MAX THE MAXIMUM DISCHARGE RATE FROM THE MILL, THE MAXIMUM SIZE FOR WHICH THIS RATE IS APPLICABLE (SIZEDMAX), 
6040  '*** AND APPLIES AN S CURVE FOR VALUES BETWEEN SIZEDMAX AND THE GRATE SIZE  
6050  '******************************************************************************
6060  '******************************************************************************
6070  'CALCULATE DMAX - MAXIMUM DISCHARGE RATE
6100  WTLD6=0:  WTLD12=0
6110  FOR I=1 TO 6
6120  WTLD6=WTLD6+MC*MCP(I)/100
6130  WTLD12=WTLD12+MC*MCP(I)/100
6140  NEXT I
6150  FOR I=7 TO 12
6160  WTLD12=WTLD12+MC*MCP(I)/100
6170  NEXT I
6180  IF WTLD6/WTLD12 < .5 THEN WTLD6=WTLD12*.5
6190  DMAX=(130*(WTLD6/WTLD12)-56.9-.897*(DISCH-70)))*(MC/1000)
6200  IF DMAX < 1 THEN DMAX=1
6210  SIZEDMAX=(10^(.044*DISCH -.294))*(1+LOG(GRATE/3353))
6220  A=LOG(SIZEDMAX)
6230  B=LOG(GRATE)
6240  FOR I=1 TO 26
6250  IF S(I) > GRATE THEN D(I)=0: GOTO 6290
6260  IF S(I) < SIZEDMAX THEN D(I)=DMAX: GOTO 6290
6270  X=LOG(S(I))
6280  D(I)=DMAX*(X-B)^2*(2*X-3*A+B)/(B-A)^3
6290  NEXT I
6300  RETURN

6500  '********* CALCULATION OF R - THE BREAKAGE RATE FUNCTION **********
6510  COARSE1=0:  COARSE2=0:  COARSE3=0
6520  FOR I=1 TO 5
6530  COARSE1=COARSE1 + MFP(I)
6540 NEXT I
6550 WEARRATE = .0071*COARSE1*(500/MC)^.71
6560 R(S)=(3*WEARRATE+1.2*(WEARRATE-.082))/(1-MAB(1,1))
6570 FOR I=1 TO 4
6580 R(I)=R(S)
6590 NEXT I
6600 FOR I=6 TO CRLNO
6610 R(I)=SQR(2)*R(I-1)
6620 NEXT I
6630 FOR I=1 TO 3
6640 COARSE2=COARSE2 + MFP(I)
6650 NEXT I
6660 FOR I=3 TO 5
6670 COARSE3=COARSE3 + MFP(I)
6680 NEXT I
6690 SIZECBP=10*(2.63+.014*COARSE2)
6700 IF MC>1000 THEN PRINT "FEED RATE TOO HIGH - MILL OVERFILLING";GOTO 12000
6710 MCE=MC/10
6720 POWER=.072*(306.7-44.37*MCE-.1517*(MCE^2)-783.98/MCE+12.27*MCE*LOG(MCE))
6730 IF POWER < 0 THEN POWER = 0
6740 POWER=POWER/TMF*1000
6750 RATECBP=35.3*POWER*(500/MC)*(COARSE3/COARSE1)
6760 FOR I=1 TO 26
6770 IF SIZECBP >= S(I) THEN 6790
6780 NEXT I
6790 CBPNO=I-1
6800 FOR I=CRLNO+1 TO 26
6810 R(I)=SQR(.5)*ABS(I-CBPNO)*RATECBP
6820 NEXT I
6830 IF CONFIG = 1 THEN 7100 'SKIP THIS SECTION IF NO CRUSHER IN CIRCUIT
6840 ***********************************************************************************************************
6850 ** CALCULATE PHI(1) = PROPORTION IN MILL CONTENTS AT STEADY STATE **
6860 ** THAT IS FAST BREAKING AND RECALCULATE THE ABRASION BREAKAGE RATES **
6870 ** R(I) BASED ON THE PROPORTION OF FAST BREAKING MATERIAL. **
6880 ***********************************************************************************************************
6890 RRAT=15.75 'R RAT(IO) CALCULATED EXTERNALLY BASED ON NORMAL RUN
6900 FOR I=1 TO 4
6910 RRAT=RRAT-.1 'REGRESSION OF CALCULATED RATIOS
6920 PHI(I)=PHI(I)*TMF*TMFP(I)/100/(((1-A(I,1))*R(I)*RRAT+D(I))*MC*MCP(I)/100)
6930 IF PHI(I) < 0 THEN PHI(I)=0
6940 R(I)=PHI(I)*RRAT*R(I)+(1-PHI(I))*RRAT/16*R(I)
6950 NEXT I
6960 FOR I=CRLNO+1 TO ABLN0
6970 RRAT=RRAT-.375
6980 PHI(I)=PHI(I)*TMF*TMFP(I)/100/(((1-A(I,1))*R(I)*RRAT+D(I))*MC*MCP(I)/100)
6990 IF PHI(I) < 0 THEN PHI(I)=0
7000 R(I)=PHI(I)*RRAT*R(I)+(1-PHI(I))*RRAT/16*R(I)
7010 NEXT I
7020 FOR I=1 TO ABLN0+1 TO ABLN0
7030 RRAT=RRAT-.836
7040 PHI(I)=PHI(I)*TMF*TMFP(I)/100
7050 IF (((1-A(I,1))*R(I)*(7-I+CRLNO)/7*RRAT+D(I))*MC*MCP(I)/100)=0 THEN
7060 PHI(I)=0;GOTO 7070
7070 IF PHI(I) < 0 THEN PHI(I)=0
R(I) = (I-CRLNO)/7*R(I)+PHIP(I)*RRAT*R(I)*(7-I+CRLNO)/7
+(1-PHIP(I))*RRAT/16*R(I)*(7-I+CRLNO)/7
NEXT I
7100 RETURN

REM*****MATRIX MULTIPLICATION SUBROUTINE (MATMULT)**************
710 REM**MATRIX C = MATRIX A X MATRIX B
720 FOR I = 1 TO 26:PRINT I
730 FOR J = 1 TO 26
740 P(I,J) = 0
750 FOR K = 1 TO 26
760 P(I,J) = P(I,J) + CB(I,K) * CB2(K,J)
770 NEXT K
780 NEXT J
790 NEXT I
800 RETURN

REM**SUBROUTINE TO SOLVE FOR MILL CONTENTS ***************
800' THIS SECTION OF THE PROGRAM SOLVES THE EQUATIONS:
801' TMFT = (D+R-AR)*MCT AND MDT = D*MCT WHERE:
802' TMFT = TOTAL MILL FEED FLOWRATE VECTOR BY SIZE
803' D = DISCHARGE FUNCTION
804' R = BREAKAGE RATE FUNCTION
805' A = BREAKAGE DISTRIBUTION FUNCTION
806' MCT = MILL CONTENTS WEIGHT VECTOR BY SIZE
807' MDT = MILL DISCHARGE FLOWRATE VECTOR BY SIZE
808' *************** CALCULATE D+R-AR ***************
810 FOR I=1 TO 26
811 FOR J=I TO 26
812 P(I,J)=A(I,J)*R(I)
813 IF I=J THEN C(I,J)=D(I)+R(I)-P(I,J) ELSE C(I,J)=-P(I,J)
814 NEXT J
815 NEXT I

8160 MCT(I)=TMF*TMFP(I)/100/C(I,I):MDT(I)=D(I)*MCT(I)
8170 FOR I=2 TO 26
8180 SUM=0
8190 NEXT J
8200 MCT(I)=SUM+MCT(J)*C(J,I)
8210 NEXT J
8220 IF C(I,I)=0 THEN C(I,I)=.01
8230 MCT(I)=(TMF*TMFP(I)/100-SUM)/C(I,I)
8240 IF MCT(I) < 0 THEN MCT(I)=0
8250 MDT(I)=D(I)*MCT(I)
8260 NEXT I
8270 T=0:T1=0
8280 FOR I=1 TO 26
8290 T=T+MDT(I)
8300 T1=T1+MCT(I)
8310 NEXT I
8320 TMDFP=0:TMCP=0:MF80=0:MC80=0
8330 FOR I=1 TO 26
8340 MDP(I)=MDT(I)/T*100
8350 TMDP=TMDFP+MDP(I)
8360 MCP(1) = MCT(1) / T * 100
8370 TMCP = TMCP + MCP(1)
8380 IF MDF80 = 0 THEN IF TMCP > 20 THEN MDF80 = I - 1: TMCP80 = TMCP
8390 IF MCF80 = 0 THEN IF TMCP > 20 THEN MCF80 = I - 1: TMCP80 = TMCP
8400 SF(I) = MCP(I)
8410 NEXT I
8420 MDF80 = EXP(LOG(S(MDF80)) - (20 + MDF80 + 1 - TMCP80) / MDF80 + 1)
8430 MCF80 = EXP(LOG(S(MCF80)) - (20 + MCF80 + 1 - TMCP80) / MCF80 + 1)
8440 MD = T: MC = T1
8450 SF = MD
8460 RETURN
9000 ' PROGRAM LEADER SUBROUTINE '}
9010 IF M F <> 0 THEN 9250
9020 COL(1) = 1: COL(2) = 1: COL(3) = 7: COL(4) = 2: COL(5) = 2
9030 CLS: KEY OFF: LOCATE 1, 1, 0: A = 719: B = 288: FOR I = 1 TO 2: A = A - 1: B = B - 1: LINE(I, I) - (A, B), COL(I). B = NEXT I
9040 FOR I = 1 TO 4: PRINT: NEXT I
9050 LOCATE 4, 29: COLOR 0, 7: PRINT " COLORADO SCHOOL OF MINES "
9060 LOCATE 6, 22: COLOR 5, 0: PRINT " Department of Metallurgical Engineering "
9070 LOCATE 7, 0: LOCATE 8, 35: PRINT " Ph.D. Thesis " : PRINT : PRINT : PRINT
9080 PRINT " PROGRAM: THE MUSHER UNIT MODEL "
9090 PRINT " AUTHOR: Tom Com! "
9100 PRINT " VERSION: 1.0 "
9110 PRINT " DATE CREATED: September 22, 1986 "
9120 LOCATE 19, 27
9130 COLOR 0, 4, , 64: PRINT " HIT SPACE BAR TO CONTINUE "
9140 A$ = INKEY$: IF A$ = " " THEN CLS: COLOR 7, 0, 0, 0 ELSE 9140
9150 ' ---------- INPUT DATA SUBROUTINE ----------
9160 COLOR 0, 7
9170 LOCATE 5, 20: PRINT " PLEASE INPUT THE FOLLOWING INFORMATION "
9180 COLOR 7, 0
9190 LOCATE 8, 20: INPUT " OPERATOR NAME: ": H1$
9200 LOCATE 10, 20: INPUT " DATE: ": H2$
9210 LOCATE 12, 20: INPUT " SUBJECT: ": H3$
9220 LOCATE 15, 27: PRINT " HIT 'RETURN' TO CONTINUE";
9230 A$ = INKEY$: IF A$ = " " THEN 9230
9240 IF A$ = CHR$(13) THEN 9250 ELSE 9230
9250 ' INITIALIZE SOLUTION, INPUT FILES, AND SET UP BREAKAGE VECTORS
9260 PASS = 1
9270 CLS: COLOR 0, 2, 0, 64: PRINT " WORKING ": COLOR 7, 0, 0, 0
9280 OPEN " !. I", #1, " SCREEN.BAS "
9290 FOR I = 1 TO 10
9300 INPUT #1. SC(I)
9310 NEXT I
9320 CLOSE #1
9330 OPEN " !. I", #1, " CRUSHER.BAS "
9340 FOR I = 1 TO 13
9350 INPUT #1, CC(I)
9360 NEXT I
9370 CLOSE #1
9380 OPEN " !. I", #1, " SIZE.BAS "
9390 FOR I = 1 TO 26
9400 INPUT #1, S(I)
9410 NEXT I
9420 CLOSE #1.
9430 OPEN "!"#,1,"CYCLONE.BAS"
9440 FOR I=1 TO 16
9450 INPUT #1,CY(I)
9460 NEXT I
9470 CLOSE #1
9480 OPEN "!"#,1,"MILL.BAS"
9490 FOR 1=1 TO 5
9500 INPUT #1,M(1)
9510 NEXT I
9520 TMFP=0: MFF80=0
9530 FOR 1=1 TO 26
9540 INPUT #1,MF(I)
9550 TMFP=TMFP+MF(I)
9560 IF MFF80=0 THEN IF TMFP>20 THEN MFF80=1:TMFP80=TMFP
9570 "**** CALCULATE MILL FEED F80 (SIZE WHICH PASSES 80% OF MATERIAL ****
9580 MFF80=EXP(LOG(S(MFF80))-(20+MF(MFF80+1)-TMFP80)/MF(MFF80+1))
9590 CLOSE #1
9600 MF=M(1)
9610 GRADE=25400*M(2)
9620 CONFIG=M(3)
9630 DISCH=M(4)
9640 FT=M(5)
9650 ' ABRASION BREAKAGE DISTRIBUTION MATRIX
9660 MAB(1,1)=.99: MAB(1,2)=.0035: MAB(1,8)=.0007: MAB(1,9)=.0012
9670 MAB(1,10)=.0011: MAB(1,11)=.0009: MAB(1,12)=.0007
9680 MAB(1,13)=.0005: MAB(1,14)=.0004: MAB(1,15)=.0003
9690 MAB(1,16)=.0002: MAB(1,17)=.0002: MAB(1,18)=.0001
9700 MAB(2,2)=.99: MAB(2,3)=.0035: MAB(2,9)=.0007: MAB(2,10)=.0012
9710 MAB(2,11)=.0011: MAB(2,12)=.0009: MAB(2,13)=.0007
9720 MAB(2,14)=.0005: MAB(2,15)=.0004: MAB(2,16)=.0003
9730 MAB(2,17)=.0002: MAB(2,18)=.0002: MAB(2,19)=.0001
9740 MAB(3,3)=.99: MAB(3,4)=.0035: MAB(3,10)=.0007: MAB(3,11)=.0012
9750 MAB(3,12)=.0011: MAB(3,13)=.0009: MAB(3,14)=.0007
9760 MAB(3,15)=.0005: MAB(3,16)=.0004: MAB(3,17)=.0003
9770 MAB(3,18)=.0002: MAB(3,19)=.0002: MAB(3,20)=.0001
9780 MAB(4,4)=.99: MAB(4,5)=.0035: MAB(4,11)=.0007: MAB(4,12)=.0012
9790 MAB(4,13)=.0011: MAB(4,14)=.0009: MAB(4,15)=.0007
9800 MAB(4,16)=.0005: MAB(4,17)=.0004: MAB(4,18)=.0003
9810 MAB(4,19)=.0002: MAB(4,20)=.0002: MAB(4,21)=.0001
9820 MAB(5,5)=.99: MAB(5,6)=.0035: MAB(5,12)=.0007: MAB(5,13)=.0012
9830 MAB(5,14)=.0011: MAB(5,15)=.0009: MAB(5,16)=.0007
9840 MAB(5,17)=.0005: MAB(5,18)=.0004: MAB(5,19)=.0003
9850 MAB(5,20)=.0002: MAB(5,21)=.0002: MAB(5,22)=.0001
9860 MAB(6,6)=.99: MAB(6,7)=.0035: MAB(6,13)=.0007: MAB(6,14)=.0012
9870 MAB(6,15)=.0011: MAB(6,16)=.0009: MAB(6,17)=.0007
9880 MAB(6,18)=.0005: MAB(6,19)=.0004: MAB(6,20)=.0003
9890 MAB(6,21)=.0002: MAB(6,22)=.0002: MAB(6,23)=.0001
9900 MAB(7,7)=.99: MAB(7,8)=.0035: MAB(7,13)=.0007: MAB(7,14)=.0012
9910 MAB(7,15)=.0011: MAB(7,16)=.0009: MAB(7,17)=.0007
9920 MAB(7,18)=.0005: MAB(7,19)=.0004: MAB(7,20)=.0003
9930 MAB(7,21)=.0002: MAB(7,22)=.0002: MAB(7,23)=.0001
9940 MAB(8,8)=.99: MAB(8,9)=.0035: MAB(8,13)=.0007: MAB(8,14)=.0012
9950 MAB(8,15)=.0011: MAB(8,16)=.0009: MAB(8,17)=.0007
9960 MAB(8,18)=.0005: MAB(8,19)=.0004: MAB(8,20)=.0003
MAB(8,21) = .0002: MAB(8,22) = .0002: MAB(8,23) = .0001
MAB(9,9) = .99: MAB(9,10) = .0035: MAB(9,13) = .0007: MAB(9,14) = .0012
MAB(9,15) = .0011: MAB(9,16) = .0009: MAB(9,17) = .0007
MAB(9,18) = .0005: MAB(9,19) = .0004: MAB(9,20) = .0003
MAB(10,10) = .99: MAB(10,11) = .0035: MAB(10,13) = .0007: MAB(10,14) = .0012
MAB(10,15) = .0011: MAB(10,16) = .0009: MAB(10,17) = .0007
MAB(10,18) = .0005: MAB(10,19) = .0004: MAB(10,20) = .0003
MAB(10,21) = .0002: MAB(10,22) = .0002: MAB(10,23) = .0001
MAB(11,11) = .99: MAB(11,12) = .0035: MAB(11,13) = .0007: MAB(11,14) = .0012
MAB(11,15) = .0011: MAB(11,16) = .0009: MAB(11,17) = .0007
MAB(11,18) = .0005: MAB(11,19) = .0004: MAB(11,20) = .0003
MAB(12,12) = .99: MAB(12,13) = .0042: MAB(12,14) = .0012
MAB(12,15) = .0011: MAB(12,16) = .0009: MAB(12,17) = .0007
MAB(12,18) = .0005: MAB(12,19) = .0004: MAB(12,20) = .0003
MAB(12,21) = .0002: MAB(12,22) = .0002: MAB(12,23) = .0001
MAB(13,13) = .99: MAB(13,14) = .0054
MAB(13,15) = .0011: MAB(13,16) = .0009: MAB(13,17) = .0007
MAB(13,18) = .0005: MAB(13,19) = .0004: MAB(13,20) = .0003
MAB(13,21) = .0002: MAB(13,22) = .0002: MAB(13,23) = .0001
MAB(14,14) = .99
MAB(14,15) = .0065: MAB(14,16) = .0009: MAB(14,17) = .0007
MAB(14,18) = .0005: MAB(14,19) = .0004: MAB(14,20) = .0003
MAB(14,21) = .0042: MAB(14,22) = .0002: MAB(14,23) = .0001
MAB(15,15) = .99: MAB(15,16) = .0074: MAB(15,17) = .0007
MAB(15,18) = .0005: MAB(15,19) = .0004: MAB(15,20) = .0003
MAB(15,21) = .0002: MAB(15,22) = .0002: MAB(15,23) = .0001
MAB(16,16) = .99: MAB(16,17) = .0081
MAB(16,18) = .0005: MAB(16,19) = .0004: MAB(16,20) = .0003
MAB(16,21) = .0002: MAB(16,22) = .0002: MAB(16,23) = .0001
MAB(17,17) = .99
MAB(17,18) = .0086: MAB(17,19) = .0004: MAB(17,20) = .0003
MAB(17,21) = .0002: MAB(17,22) = .0002: MAB(17,23) = .0001
MAB(18,18) = .99: MAB(18,19) = .0003: MAB(18,20) = .0003
MAB(18,21) = .0002: MAB(18,22) = .0002: MAB(18,23) = .0001
MAB(19,19) = .99: MAB(19,20) = .0093
MAB(19,21) = .0002: MAB(19,22) = .0002: MAB(19,23) = .0001
MAB(20,20) = .99
MAB(20,21) = .0095: MAB(20,22) = .0002: MAB(20,23) = .0001
MAB(21,21) = .99: MAB(21,22) = .0097: MAB(21,23) = .0001
MAB(22,22) = .99: MAB(22,23) = .0098
MAB(23,23) = .9998
MAB(24,24) = .9998
MAB(25,25) = .9998
MAB(26,26) = 1
CRUSHING BREAKAGE DISTRIBUTION VECTOR
MCB(1) = .1979: MCB(2) = .331: MCB(3) = .2147: MCB(4) = .1226: MCB(5) = .0654
MCB(6) = .0338: MCB(7) = .0172: MCB(8) = .0087: MCB(9) = .0043: MCB(10) = .0022
MCB(11) = .0011: MCB(12) = .0005: MCB(13) = .0003: MCB(14) = .0001: MCB(15) = .0001
CRUSHING BREAKAGE DISTRIBUTION MATRIX AS
* * * THIS SECTION CALCULATES THE BREAKAGE DISTRIBUTION MATRIX AS
* * * SPECIFIED BY STANLEY IN THE SAIMM JOURNAL NOVEMBER 1974. ***
FOR I = 1 TO 5: T = 0
FOR J = 1 TO 25
A(I,J) = MAB(I,J): T = T + A(I,J)
NEXT J: A(I,26) = 1 - T: NEXT I
FOR I=6 TO CRLNO:T=0
FOR J=I TO 25
A(I,J)=MAB(I,J):T=T+A(I,J)
NEXT J
A(I,26)=1-T
NEXT I
FOR I=CRLNO+1 TO ABLNO
T=0
FOR J=I TO 25
A(I,J)=(ABLO-I+1)/7+MCB(J-I+1)*(I-CRLNO)/7:T=T+A(I,J)
NEXT J
A(I,26)=1-T
NEXT I
FOR I=ABLNO+1 TO 26:T=0
FOR J=I TO 25
A(I,J)=MCB(J-I+1):T=T+A(I,J)
NEXT J
A(I,26)=1-T
NEXT I
FOR I=1 TO 26
TFP(I)=MFP(I)
MCP(I)=MFP(I)
NEXT I
MC=500
TMF=MF

********** MAIN BODY **********

** THIS SECTION CALLS THE VARIOUS SUBROUTINES AND CHECKS FOR CONVERGENCE. IT ALSO RECALCULATES RECIRCULATING LOADS AND DISPLAYS THE CIRCUIT FLOWSHEET AS THE CALCULATIONS CONTINUE. **

R5L=R(5)
GOSUB 5000
GOSUB 6000
GOSUB 8000
GOSUB 1000
IF CONFIG=1 THEN 11130
IF CONFIG=2 THEN 11140
TMF=MFP(I)

**** DISPLAY CIRCUIT CONDITION AT EACH PASS ****
CLS:P$="&####
IF CONFIG=1 THEN CF1=CF:CF2=0
IF CONFIG=2 THEN CF1=0:CF2=CF
11230 PRINT USING "& ### & ;"PASS NO.",PASS,"
11240 PRINT
11250 PRINT USING P$;
11260 PRINT
11270 PRINT "\1/" CYC- "--"
11280 PRINT "\ CRUSH/ LONE/ " CYF, */
11290 PRINT USING P$; "",CF2," "W FEED AUTO- "MF, "
11300 PRINT USING P$; "",CF1," "W FEED AUTO- "MF, "
11310 PRINT USING P$; "",CF1," "W FEED AUTO- "MF, "
11320 PRINT USING P$; "",CF1," "W FEED AUTO- "MF, "
11330 PRINT USING P$; "",CF1," "W FEED AUTO- "MF, "
11340 PRINT USING P$; "",CF1," "W FEED AUTO- "MF, "
11350 PRINT USING P$; "",CF1," "W FEED AUTO- "MF, "
11360 PRINT USING P$; "",CF1," "W FEED AUTO- "MF, "
11370 PRINT USING P$; "",CF1," "W FEED AUTO- "MF, "

MUSHER UNIT FLOWSHEET 

CYO," / 

LONE/ " CYF, /*
11380 PRINT "MILL: "
11390 PRINT "----------------------- "
11400 PRINT USING P$;" \%/",MD," "
11410 PRINT USING P$;",CF, "
11420 PRINT "allax "
11430 PRINT USING P$;" SCREEN: "
11440 PRINT "----------------------- "
11450 PASS=PASS+1:IF PASS>50 THEN LPRINT "NOT CONVERGING":GOTO 12000
11460 IF ABS((MF-CYO)/MF) <.05 THEN CON$="Y" :PRINT R(5)," "; ELSE CON$="N"
11470 IF ABS(MF-CYO)>0.5 OR ABS(R(5)-R5L)>0.2 OR CON$ <> "Y" THEN 11050
11480 PRINT "CONVERGENCE REACHED - HIT SPACE BAR TO CONTINUE"
11490 I$=INKEY$: IF I$="" THEN 11490 ELSE IF I$ < > " " THEN BEEP
1150 PRINTEXT: "PRINT OUT INPUT DATA "
11510 PRINT "COLORADO SCHOOL OF MINES"
11520 FOR I=1 TO 2000
11530 NEXT I
11540 P1$=STRING$(70,"*")
11550 LPRINT P1$:LPRINT :LPRINT
11560 LPRINT "COLORADO SCHOOL OF MINES"
11570 LPRINT "AUTHOR: Tom Comi"
11580 LPRINT "PROGRAM: Musher Unit Model"
11590 LPRINT "VERSION: 1.0"
11600 LPRINT "DATE: April 20, 1987"
11610 LPRINT "OPERATOR: ",HI$
11620 LPRINT P1$:LPRINT :LPRINT
11630 LPRINT "SUBJECT: ",H2$
11640 LPRINT "DATE: ",H3$
11650 LPRINT "PLANT CONFIGURATION - "
11660 LPRINT :LPRINT
11670 LPRINT :P1$=STRING$(70,"*")
11680 LPRINT :LPRINT
11690 LPRINT "RESULTS:
11700 LPRINT :LPRINT
11710 LPRINT "MATERIAL FLOW RATE (KGPH) F80 (MICRONS) %
11720 IF CONFIG=1 THEN PRINT "CONVENTIONAL AUTOGENOUS MILL" ELSE PRINT "AUTOGENOUS MILL IN SERIES WITH CRUSHER"
11730 FS="&####&####"
11740 LPRINT :LPRINT
11750 LPRINT USING FS;"GRATE SIZE (MM)=",MI(2)*25.4,"SCREEN GAP (IN) =",SC(1);
11760 IF CONFIG = 1 THEN LPRINT " ":GOTO 11780
11770 LPRINT USING ",&####","CRUSHER GAP (MM) =", CC(6)
11780 LPRINT USING ",&####","SHATTER LIMIT SIZE NUMBER =",CRLNO;
11790 LPRINT USING ",&####","DMAX =",DMAX
11800 LPRINT USING ",&####","NEW SURFACE DAMAGE THICK. (MM)",FT,",
11810 LPRINT USING ",&####","MILL CONTENTS % +3 IN",MCP(1)+MCP(2)+MCP(3)
11820 LPRINT :LPRINT
11830 LPRINT "MATERIAL FLOWRATE (KGPH) F80 (MICRONS) % -200 MESH"
11840 FS=" 
11850 LPRINT USING FS;"NEW FEED",MF,MFF80,MFP(25)+MFP(26)
11860 LPRINT USING FS;"MILL CONTENTS",MC,MCF80,MCP(25)+MCP(26)
11870 LPRINT USING FS;"MILL DISCHARGE",MD,MDF80,MDP(25)+MDP(26)
11880 LPRINT USING FS;"SCREEN OVER SIZE",SO,SOF80,CFP(25)+CFP(26)
11890 LPRINT USING FS;"SCREEN UNDER SIZE",SU,SUF80,CYFP(25)+CYFP(26)
11900 IF CONFIG=1 THEN 11920
11910 LPRINT USING FS;"CRUSHER DISCHARGE",CF,CDF80,CDP(25)+CDP(26)
11920 LPRINT USING F$;"CYCLONE UNDERFLOW",CYU,CYUF80,CYUP(25)+CYUP(26)
11930 LPRINT USING F$;"CYCLONE OVERFLOW",CYO,CYOFS0,CYOP(25)+CYOP(26)
11940 LPRINT :LPRINT
11950 LPRINT CHR$(12)
12000 COLOR 7,0,0,0
12010 END
10 'MENU ENTERED FROM AUTOEXEC - MILL MODEL CONTROL
20 CLS: KEY OFF: LOCATE 1,1: O = 719: B = 288: FOR I = 1 TO 2: A = A - 1: B = B - 1: LINE(I,I) - (A,B),5,B:NEXT I
30 FOR I = 1 TO 4: PRINT: NEXT I
40 LOCATE 4, 29: COLOR 0, 3: PRINT "COLORADO SCHOOL OF MINES"
50 LOCATE 6, 22: COLOR 2, 0: PRINT "Department of Metallurgical Engineering"
60 LOCATE 8, 22: COLOR 0, 4, 64: PRINT "INPUT:"; : COLOR 7, 0, 0: PRINT "'1' TO INPUT OR UPDATE CRUSHER PARAMETERS"
68 LOCATE 12, 24: PRINT "'2' TO INPUT OR UPDATE SCREEN PARAMETERS"
69 LOCATE 14, 24: PRINT "'3' TO INPUT OR UPDATE CYCLONE PARAMETERS"
70 LOCATE 16, 24: PRINT "'4' TO INPUT OR UPDATE MILL PARAMETERS"
72 LOCATE 18, 24: PRINT "'5' TO RUN THE MUSHER UNIT MODEL"
74 LOCATE 18, 24: PRINT "'6' TO RETURN TO MAIN MENU"
75 K$= INKEY$: IF K$="" THEN 75
80 COLOR 7, 0, 0: CLS: IF K$="1" THEN LOAD"CRUSHIN",R
95 IF K$="2" THEN LOAD"SCREENIN",R
96 IF K$="3" THEN LOAD"CYCLONIN",R
97 IF K$="4" THEN LOAD"MILLIN",R
99 IF K$="5" THEN SYSTEM ELSE LOAD"MENU",R
10 ' PROGRAM TO INPUT CRUSHER PARAMETERS
100 KEY OFF:COLOR 7,0:CLS
110 DIM B(30),DISC$(30)
120 N$="N";CV$="CURRENT VALUE =";UV$="UPDATED VALUE ="
150 DISC$(1)="BI IN (1-EXP(-(X/Y)^U))/(1-1/e)"
160 DISC$(2)="B2 DISTRIBUTION: Y' IN 1-EXP(-(X/Y')*V))"
165 DISC$(3)="B2 DISTRIBUTION: V IN 1-EXP(-(X/Y')*V))"
170 DISC$(4)="FOR A IN B = A.B1 +(1-A).B2 IN FORM OF A=D + E*G : D="
180 DISC$(5)="E="
190 DISC$(6)="G= THE CLOSED SIDE SETTING IN MM"
200 DISC$(7)="CLASSIFICATION PARAMETERS K1 AND K2 IN:
C(S)=1-[(S-K2)/(K1-K2)]^2: K1=A*G A ="
210 DISC$(8)="FOR K2=[A*G +B*(FRAC. +1 in. IN FEED)]* F(FEED RATE) A=
220 DISC$(9)="B="
230 DISC$(10)="FOR F(T) = A*T^2 + B*T +C A="
240 DISC$(11)="B="
250 DISC$(12)="C="
260 DISC$(13)="INPUT: 1 TO FIX EFFICIENCY: 2 TO CALC. FROM FEED RATE"
6000 ' *************** INPUT DATA SUBROUTINE ***************
6100 CLS:COLOR 0,7;PRINT
6110 PRINT " THIS PROGRAM IS FOR INPUTTING CRUSHER OPERATING PARAMETERS "
6120 PRINT " PLEASE SELECT THE METHOD OF DATA ENTRY: ",COLOR 7,0:PRINT
6130 COLOR 7,0:PRINT
6140 PRINT " 1 - USE ONLY THE DATA STORED IN FILE CRUSHER.BAS"
6150 PRINT " 2 - INPUT ALL NEW DATA"
6160 PRINT " 3 - USE SOME OF THE STORED DATA WITH THE REST"
6170 PRINT " BEING INPUTTED"
6180 PRINT
6190 PRINT
6200 PRINT
6210 I$=INKEY$:IF I$="" THEN 6210
6220 IF I$="1" THEN 8000
6225 IF I$="3" THEN 6240
6230 IF I$="2" THEN 6300 ELSE 6210
6240 OPEN "1","#1,"CRUSHER.BAS"
6250 FOR I=1 TO 13
6260 INPUT #1,B(I)
6270 NEXT I
6280 CLOSE #1
6290 IF I$="3" THEN 6340
6300 CLS:COLOR 0,7;PRINT
6310 PRINT " SINCE ALL DATA ARE TO BE ENTERED "
6320 PRINT " BE SURE TO ANSWER 'Y' TO EACH QUESTION "
6330 FOR I=1 TO 2000:NEXT I:PRINT
6340 CLS:COLOR 0,7;PRINT
6350 LOCATE 1,20;PRINT " INPUT THE FOLLOWING INFORMATION "
6360 COLOR 7,0:LOCATE 3,1;PRINT "DESCRIPTION"
6370 LOCATE 3,30;PRINT "CURRENT VALUE"
6380 LOCATE 3,50;PRINT "UPDATE VALUE ? ('Y' OR 'N')"
6390 FOR J=1 TO 7
6400 LOCATE 2*J+4,1;PRINT DISC$(J)
6410 LOCATE 2*J+5,30;PRINT USING"##.#######":B(J)
6420 LOCATE 2*J+5,50
6430 U$=INKEY$:IF U$="" THEN 6430
6440 IF U$="B" AND J=1 THEN BEEP: GOTO 6340
6450 IF U$="B" THEN LOCATE 2*J+4,1:PRINT STRING$(80," "):PRINT STRING$(80," "):J=J-2:GOTO 6480
6460 IF U$="N" OR U$="n" THEN 6480
6470 LOCATE 2*J+5.5:PRINT UV$;:INPUT " ",B(J)
6480 NEXT J
6482 CLS:COLOR 0,7:PRINT
6483 LOCATE 1,20:PRINT " INPUT THE FOLLOWING INFORMATION 
6484 COLOR 7,0:LOCATE 3,1:PRINT "DESCRIPTION"
6485 LOCATE 3,30:PRINT "CURRENT VALUE"
6486 LOCATE 3,50:PRINT "UPDATE VALUE ? ('Y' OR 'N')"
6488 FOR J=8 TO 13
6490 LOCATE 2*J-11,1:PRINT DISC$(J)
6500 LOCATE 2*J-10.3:PRINT USING"####.#######";B(J)
6510 LOCATE 2*J-10.50
6520 U$=INKEY$:IF U$="" THEN 6520
6530 IF U$="B" AND J=8 THEN BEEP:GOTO 6340
6540 IF U$="B" THEN LOCATE 2*J-11,1:PRINT STRING$(80," "):PRINT STRING$(80," "):J=J-2:GOTO 6546
6542 IF U$="N" OR U$="n" THEN 6546
6544 LOCAL 2*J-10.50:PRINT UV$;:INPUT " ",B(J)
6546 NEXT J
7440 CLS:PRINT "HIT 'C' TO CONTINUE, 'B' TO BACK UP AND 'R' TO RESTART THE DATA SECTION"
7450 U$=INKEY$:IF U$="" THEN 7450 ELSE IF U$="B" THEN 6482
7460 IF U$="R" THEN 6340 ELSE IF U$ <> "C" THEN BEEP:GOTO 7440
7470 ' UPDATE DATA FILE
7480 CLS:COLOR 0,7
7490 LOCATE 12,12:PRINT " NOTE: DATA FILE NOW BEING UPDATED "
7500 COLOR 7,0
7510 OPEN "O",#1,"A:CRUSHER.BAS"
7520 FOR I=1 TO 13
7530 WRITE #1,B(I)
7540 NEXT I
7550 CLOSE #1
8000 LOAD"MUSHMENU",R
10 'PROGRAM TO INPUT SCREEN PARAMETERS
100 KEY OFF:COLOR 7,0:CLS
110 DIM B(30),DISC$(30)
120 N$="N":CV$="CURRENT VALUE =":UV$="UPDATED VALUE ="
130 DISC$(1)="SCREEN HOLE NOMINAL SIZE (IN MILLIMETERS)"
140 DISC$(2)="SCREEN WIRE NOMINAL DIAMETER (IN MILLIMETERS)"
150 DISC$(3)="LENGTH OF SCREEN (IN METERS)"
160 DISC$(4)="EFFICIENCY/LOAD FACTOR K1 FOR LOADS LESS THAN 300 TPH"
170 DISC$(5)="FOR K1(T)=A +B*T +C*T^2 +D/T +E*T*LOG(T) A=
180 DISC$(6)="B="
190 DISC$(7)="C="
200 DISC$(8)="D="
210 DISC$(9)="E="
220 DISC$(10)="INPUT: 1 TO FIX SCREEN EFFICIENCY; 2 TO CALC. FROM RATE"
6000 ' ******************** INPUT DATA SUBROUTINE ********************
6100 CLS:COLOR 0,7:PRINT
6110 PRINT "THIS PROGRAM IS FOR INPUTTING SCREEN OPERATING PARAMETERS"
6120 PRINT "PLEASE SELECT THE METHOD OF DATA ENTRY:"
6130 COLOR 7,0:PRINT
6140 PRINT "1 - USE ONLY THE DATA STORED IN FILE SCREEN.BAS"
6150 PRINT
6160 PRINT "2 - INPUT ALL NEW DATA"
6170 PRINT
6180 PRINT "3 - USE SOME OF THE STORED DATA WITH THE REST"
6190 PRINT "BEING INPUTTED"
6200 PRINT
6210 I$=INKEY$:IF I$="" THEN 6210
6220 IF I$="1" THEN 8000
6225 IF I$="3" THEN 6240
6230 IF I$="2" THEN 6240 ELSE 6210
6240 OPEN "I",#1,"SCREEN.BAS"
6250 FOR I=1 TO 10
6260 INPUT #1, B(I)
6270 NEXT I
6280 CLOSE #1
6290 IF I$="3" THEN 6340
6300 CLS:COLOR 0,7:PRINT
6310 PRINT "SINCE ALL DATA ARE TO BE ENTERED"
6320 PRINT "BE SURE TO ANSWER 'Y' TO EACH QUESTION"
6330 FOR I=1 TO 2000:NEXT I:PRINT
6340 CLS:COLOR 0,7
6350 LOCATE 1,20:PRINT "INPUT THE FOLLOWING INFORMATION"
6360 COLOR 7,0:LOCATE 3,1:PRINT "DESCRIPTION"
6370 LOCATE 3,30:PRINT "CURRENT VALUE"
6380 LOCATE 3,50:PRINT "UPDATE VALUE ? ('Y' OR 'N')"
6390 FOR J=1 TO 10
6400 LOCATE 2*J+2,1:PRINT DISC$(J)
6410 LOCATE 2*J+3,30:PRINT USING"#####.#####";B(J)
6420 LOCATE 2*J+3,50
6430 U$=INKEY$:IF U$="" THEN 6430
6440 IF U$="B" AND J=1 THEN BEEP:GOTO 6340
6450 IF U$="B" THEN LOCATE 2*J+2,1:PRINT STRING$(80," "):PRINT
6460 IF U$="N" OR U$="n" THEN 6480
6470 LOCATE 2*J+3,50:PRINT UV$:::INPUT " ".B(J)
6480 NEXT J
CLS:PRINT "HIT 'C' TO CONTINUE, 'B' TO BACK UP AND 'R' TO RESTART THE DATA SECTION"
    U$=INKEY$: IF U$="" THEN 7450 ELSE IF U$="B" THEN 6340
    IF U$="R" THEN 6340 ELSE IF U$ <> "C" THEN BEEP:GOTO 7440
    UPDATE DATA FILE
    CLS:COLOR 0,7
    LOCATE 12,12:PRINT "NOTE: DATA FILE NOW BEING UPDATED"
    COLOR 7,0
    OPEN "O",#1,"A:SCREEN.BAS"
    FOR I=1 TO 10
        WRITE #1,8(I)
    NEXT I
    CLOSE #1
8000 LOAD"MUSHMENU",R
10 'PROGRAM TO INPUT HYDROCYCLONE PARAMETERS
100 KEY OFF:COLOR 7,0:CLS
110 DIM B(30),DISC$(30)
120 N$="N":CV$="CURRENT VALUE = " :UV$="UPDATED VALUE ="
150 DISC$(1)="CYCLONE FEED PERCENT SOLIDS: FPS = "
160 DISC$(2)="SPIGOTT DIAMETER (IN CENTIMETERS): SP = "
165 DISC$(3)="VORTEX FINDER DIAMETER (IN CENTIMETERS): VF = "
170 DISC$(4)="AVERAGE SPECIFIC GRAVITY OF ORE"
180 DISC$(5)="NUMBER OF CYCLONES (WILL CALC. BASED ON FEED RATE IF SET TO 0)"
190 DISC$(6)="RATIO OF FINES TO U/F-RF=K1+K2*SP/H20 IN FEED+K3/H20 IN FEED: K1 = "
200 DISC$(7)="K2 = "
210 DISC$(8)="K3 = "
220 DISC$(9)="LOG(D50C)=K1*FPS+K2*SP+K3*VF+K4*VOL. FLOW RATE+K5: K1 = "
230 DISC$(10)="K2 = "
240 DISC$(11)="K3 = "
250 DISC$(12)="K4 = "
260 DISC$(13)="K5 = "
270 DISC$(14)="FOR PLITT EQUATION: Y=1-EXP(-.693*(X/D50C)"N): N = "
280 DISC$(15)="WATER TO UNDERFLOW = RF/K1: K1 = "
290 DISC$(16)="INPUT: 1 TO SET EFFICIENCY; 2 TO CALCULATE FROM FEED RATE"

6000 ' * * * * * * * * * * * * * * * * INPUT DATA SUBROUTINE * * * * * * * * * * * * * * * * * * * * * * *
6100 CLS:COLOR 0,7:PRINT
6110 PRINT " THIS PROGRAM IS FOR INPUTTING CYCLONE OPERATING PARAMETERS "
6120 PRINT " PLEASE SELECT THE METHOD OF DATA ENTRY: 
6130 COLOR 7,0:PRINT
6140 PRINT " 1 - USE ONLY THE DATA STORED IN FILE CYCLONE.BAS"
6150 PRINT " 2 - INPUT ALL NEW DATA"
6160 PRINT " 3 - USE SOME OF THE STORED DATA WITH THE REST"
6170 PRINT " "
6180 PRINT " "
6190 PRINT " "
6200 PRINT
6210 I$=INKEY$:IF I$="" THEN 6210
6220 IF I$="1" THEN 8000
6225 IF I$="3" THEN 6240
6230 IF I$="2" THEN 6300 ELSE 6210
6240 OPEN "1",#1,"CYCLONE.BAS"
6250 FOR I=1 TO 16
6260 INPUT #1,B(I)
6270 NEXT I
6280 CLOSE #1
6290 IF I$="3" THEN 6340
6300 COLOR 0,7:PRINT
6310 PRINT " SINCE ALL DATA ARE TO BE ENTERED "
6320 PRINT " BE SURE TO ANSWER 'Y' TO EACH QUESTION "
6330 FOR I=1 TO 2000:NEXT I:PRINT
6340 CLS:COLOR 0,7:PRINT
6350 LOCATE 1,20:PRINT " INPUT THE FOLLOWING INFORMATION "
6360 COLOR 7,0:LOCATE 3,1:PRINT "DESCRIPTION"
6370 LOCATE 3,30:PRINT "CURRENT VALUE"
6380 LOCATE 3,50:PRINT "UPDATE VALUE ? ('Y' OR 'N')"
6390 FOR J=1 TO 8
6400 LOCATE 2*J+4,1:PRINT DISC$(J)
6410  LOCATE 2*J+5,30:PRINT USING"#####.#####";B(J)
6420  LOCATE 2*J+5,50
6430  U$=INKEY$:IF U$="" THEN 6430
6440    IF U$="B" AND J=1 THEN BEEP: GOTO 6340
6450    IF U$="B" THEN LOCATE 2*J+4,1:PRINT STRING$(80," "):PRINT
6460  IF U$="N" OR U$="n" THEN 6480
6470    LOCATE 2*J+5,50:PRINT UV$: INPUT " ",B(J)
6480 NEXT J
6482  CLS:COLOR 0,7:PRINT
6483  LOCATE 1,20:PRINT " INPUT THE FOLLOWING INFORMATION 
6484  COLOR 7,0:LOCATE 3,1:PRINT "DESCRIPTION"
6485  LOCATE 3,30:PRINT "CURRENT VALUE"
6486  LOCATE 3,50:PRINT "UPDATE VALUE ? ('Y' OR 'N')"
6488  FOR J=9 TO 16
6490    LOCATE 2*J-13,1:PRINT DISCS$(J)
6500    LOCATE 2*J-12,30:PRINT USING"#####.#####";B(J)
6510    LOCATE 2*J-12,50
6520    U$=INKEY$:IF U$="" THEN 6520
6530    IF U$="B" AND J=9 THEN BEEP: GOTO 6340
6540    IF U$="B" THEN LOCATE 2*J-13,1:PRINT STRING$(80," "):PRINT
6542    IF U$="N" OR U$="n" THEN 6546
6544    LOCATE 2*J-12,50:PRINT UV$: INPUT ",",B(J)
6546 NEXT J
7440  CLS:PRINT "HIT 'C' TO CONTINUE, 'B' TO BACK UP AND 'R' TO RESTART THE
7450  "DATA SECTION"
7455  U$=INKEY$:IF U$="" THEN 7450 ELSE IF U$="B" THEN 6482
7460    IF U$="R" THEN 6340 ELSE IF U$ <> "C" THEN BEEP:GOTO 7440
7470  'UPDATE DATA FILE
7480  CLS: COLOR 0,7
7490  LOCATE 12,12:PRINT " NOTE: DATA FILE NOW BEING UPDATED 
7500  COLOR 7,0
7510  OPEN "O",#1,"A:CYCLONE.BAS"
7520  FOR I=1 TO 16
7530    WRITE #1,B(I)
7540 NEXT I
7550 CLOSE #1
8000 LOAD"MUSHMENU",R
10 'PROGRAM TO INPUT MILL PARAMETERS
10 DIM B(31)
20 KEY OFF:COLOR 7,0:CLS
30 N$="N":CV$="CURRENT VALUE =":UV$="UPDATED VALUE ="
40 DISCS$(1)="FEED RATE (TONNES PER HOUR)"
50 DISCS$(2)="GRATE SIZE (INCHES)"
70 DISCS$(3)="CONFIGURATION OF THE CIRCUIT (1=CONVENTIONAL, 2=MUSHER)"
75 DISCS$(4)="MILL DISCHARGE 7. SOLIDS"
80 DISCS$(5)="FRESH ORE FAST ABRASION THICKNESS (MM)"
230 ' *************** INPUT DATA SUBROUTINE ***************
240 CLS:COLOR 0,7:PRINT
250 PRINT " THIS PROGRAM IS FOR INPUTTING MILL OPERATING PARAMETERS "
260 PRINT " PLEASE SELECT THE METHOD OF DATA ENTRY: "
270 COLOR 7,0:PRINT
280 PRINT " 1 - USE ONLY THE DATA STORED IN FILE MILL.BAS"
290 PRINT
300 PRINT " 2 - INPUT ALL NEW DATA"
310 PRINT
320 PRINT " 3 - USE SOME OF THE STORED DATA WITH THE REST"
330 PRINT " BEING INPUTTED"
340 PRINT
350 I$=INKEY$:IF I$="" THEN 350
360 IF I$="1" THEN 920
370 IF I$="3" THEN 390
380 IF I$="2" THEN 450 ELSE 350
390 OPEN "I",#I."MILL.BAS"
400 FOR I=1 TO 31
410 INPUT #1,B(I)
420 NEXT I
430 CLOSE #1
440 IF I$="3" THEN 490
450 CLS:COLOR 0,7:PRINT
460 PRINT " SINCE ALL DATA ARE TO BE ENTERED "
470 PRINT " BE SURE TO ANSWER 'Y' TO EACH QUESTION "
480 FOR I=1 TO 2000:NEXT I:PRINT
490 CLS:COLOR 0,7:PRINT
500 LOCATE 1,20:PRINT " INPUT THE FOLLOWING INFORMATION "
510 COLOR 7,0:LOCATE 3,1:PRINT "DESCRIPTION"
520 LOCATE 3,30:PRINT "CURRENT VALUE"
530 LOCATE 3,50:PRINT "UPDATE VALUE ? ('Y' OR 'N')"
540 FOR J=1 TO 5
550 LOCATE 2*J+4,1:PRINT DISCS$(J)
560 LOCATE 2*J+5,30:PRINT USING "#####.#####";B(J)
570 LOCATE 2*J+5,50
580 U$=INKEY$:IF U$="" THEN 580
590 IF U$="B" AND J=1 THEN BEEP:GOTO 490
600 IF U$="B" THEN LOCATE 2*J+4,1:PRINT STRING$(80," "):PRINT
610 .IF U$="N" OR U$="n" THEN 630
620 LOCATE 2*J+5,50:PRINT UV$:INPUT " ",B(J)
630 NEXT J
640 CLS:COLOR 0,7:PRINT
650 LOCATE 1,20:PRINT " INPUT THE FOLLOWING INFORMATION "
660 COLOR 7,0:LOCATE 3,1:PRINT "DESCRIPTION"
670 LOCATE 3,30:PRINT "CURRENT VALUE"
680 LOCATE 3,50:PRINT "UPDATE VALUE ? ('Y' OR 'N')"
690 FOR J = 1 TO 26
700 PRINT USING "& # # # " ; " FEED % IN SIZE NO.", J
710 LOCATE (J - 1) MOD 13 + 5, 1: PRINT USING "###.###" ; B(J + 5)
720 LOCATE (J - 1) MOD 13 + 5, 50
730 U$ = INKEY$: IF U$ = "" THEN 730
740 IF U$ = "B" AND J = 1 THEN BEEP: GOTO 640
750 IF U$ = "B" THEN J = J - 2: GOTO 790
760 IF U$ = "N" OR U$ = "n" THEN 790
770 LOCATE (J - 1) MOD 13 + 5, 50: PRINT "UPDATED VALUE = "; INPUT " ", B(J + 5)
780 NEXT J
790 NEXT J
800 CLS: PRINT "HIT 'C' TO CONTINUE OR 'R' TO RESTART THE DATA SECTION"
810 U$ = INKEY$: IF U$ = "" THEN 810
820 IF U$ = "R" THEN 490 ELSE IF U$ <> "C" THEN BEEP: GOTO 800
830 " UPDATE DATA FILE"
840 CLS: COLOR 0, 7
850 LOCATE 12, 12: PRINT " NOTE: DATA FILE NOW BEING UPDATED ">
860 COLOR 7, 0
870 OPEN "O", #1, "A: MILL.BAS"
880 FOR I = 1 TO 31
890 WRITE #1, B(I)
900 NEXT I
910 CLOSE #1
1000 LOAD "MUSHMENU", R
APPENDIX C

Calculation of Fast Breakage Rate in Terms of Average Breakage Rate for Different Damage Thicknesses
Table C1 - Calculation of $R_f/R$ under Standard Conditions with Fast-breaking Thickness of 1000 μm

<table>
<thead>
<tr>
<th>Size</th>
<th>R</th>
<th>$(1-b_{11})$</th>
<th>$\phi$</th>
<th>Feed %</th>
<th>Contents %</th>
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$$
\phi = \frac{\text{Size}^3 - (\text{Size} - 2 \times \text{FT})^3}{\text{Size}^3}
$$

where FT is the fast-breaking thickness in microns (1000 in this case).

$$
\phi'R_f = \frac{\phi \times 515 \times \text{Feed %}}{(1-b_{11}) \times 502.3 \times \text{Contents %}}
$$

where 515 is the feed rate in kg/hr and 502.3 is the mill contents in kg.

$$
R = \phi'R_f + (1-\phi')R_s = \phi'R_f + \frac{R_f}{16} - \frac{\phi'R_f}{16}
$$

because $R_f = 16 \times R_s$. 
Table C2 - Calculation of $R_f/R$ under Standard Conditions with Fast-breaking Thickness of 500 μm

<table>
<thead>
<tr>
<th>Size</th>
<th>$R$</th>
<th>$(1-b_{11})$</th>
<th>$\phi$</th>
<th>Feed %</th>
<th>Contents %</th>
<th>$\phi'R_f$</th>
<th>$R_f/R$</th>
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$$\phi = \frac{\text{Size}^3 - (\text{Size}^2 - 2 \times FT)^3}{\text{Size}^3}$$

where $FT$ is the fast-breaking thickness in microns (500 in this case).

$$\phi'R_F = \frac{\phi \times 515 \times \text{Feed} \%}{(1-b_{11}) \times 502.3 \times \text{Contents} \%}$$

where 515 is the feed rate in kg/hr and 502.3 is the mill contents in kg.

$$R = \phi'R_F + (1-\phi'R)S = \phi'R_F + \frac{R_F}{16} - \frac{\phi'R}{16}$$

because $R_F = 16 \times R_S$. 