TECHNOECONOMIC ANALYSIS OF PLANT GRINDING OPERATIONS

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To Fida, Michel, and Mackie.

"A tumbling mill in a plant is a lazy-appearing colossus, with a quiet grumbling in its interior that in no way reflects the intense activity there. It contains a random multitude of bodies with no positive connection either to the shell or to each other. The way these bodies move determines how well they grind. But the operator has no direct control of their movement as he has in the other size-reduction mechanisms that have been studied. He has only a statistical control by way of the interplay of shell size and shape and speed; liner conformation; tumbling-body shape and size and size distribution; feed-particle nature, shape, size, and size distribution; moisture content; feed rate; and the coactions of all these with each other and with the external pulp circuit. His colossus demands large amounts of energy and yet is reluctant to accept all that it is offered. It consumes unconscionable quantities of expensive steel. In many cases it costs more to run than any other part of the plant. It is a constant challenge to every conscientious operator. And it yields its secrets slowly."

Professor A.F. Taggart

ABSTRACT

This study presents methodology for technical and economic performance analysis of conventional grinding circuits supported by two industrial cases. A review of the strengths and weaknesses of Bond work index and population balance modelling approaches to plant circuit analysis discloses the need to refine work index determinations, and to establish a system of more meaningful ball mill circuit performance parameters.

The statistical reliability of carefully determined rod mill work index efficiency ratings is presented. Technical and economic performance improvements through rod mill speed and feed water rate adjustments are presented.

Parameters which separately characterize overall ball mill circuit classification and breakage efficiency are presented. Their relationships with circuit design and operating variables and overall circuit efficiency and economy are discussed. The particle size dependence of flotation is used to estimate the value of the grinding circuit product, and thus to integrate grinding into the overall economics of a flotation concentrator.

RÉSUMÉ

Cette étude présente une méthodologie qui permet l'analyse technique et économique des circuits de broyage conventionnels; deux études de cas illustrent cette méthodologie. L'auteur démontre par une évaluation des forces et faiblesses de l'emploi de l'indice de broyabilité et des modèles dits "d'équilibre des populations" pour l'étude des circuits de broyage, le besoin d'améliorer la mesure de la broyabilité et de définir de nouveaux paramètres permettant de mieux évaluer la performance des circuits de broyage à boulets.

On estime la fiabilité statistique d'indices de broyabilité à tiges mesurés avec soin. On quantifie des améliorations du rendement de broyeurs à tiges obtenues en ajustant la vitesse de rotation et le débit d'eau à l'alimentation.

On définit pour les circuits fermés de broyeurs à boulets, deux paramètres qui permettent de quantifier d'une part l'efficacité du système de classification et, d'autre part, l'efficacité de fragmentation. On établit le lien entre ces paramètres et la conception de circuits, les variables d'opération, et l'efficacité et l'économie globale du circuit. La valeur du produit broyé est évaluée à partir de la relation entre la performance à l'étape de séparation subséquente (la flottation) et la granulométrie du produit du circuit de broyage; on obtient ainsi un modèle économique complet du broyage dans un concentrateur à flottation.

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THESIS STRUCTURE AND CONTRIBUTIONS OF OTHERS

Owing to the broadness of the subject matter of this investigation, a number of reviews related to rod and ball milling in mineral concentration plants were carried out, including design and operation of rod mills, classification effects in closed circuit ball milling, interfacing of grinding and flotation operations, and process control of conventional ball milling. This review work formed part of the documentation for the grinding circuit study projects at the two industrial plants, and has been incorporated into the relevant sections of the thesis. Summaries of supplemental plant operating data found in the literature on rod milling and ball mill circuit control practices are given in the appendices. The other appendices provide plant descriptive information and details of the testwork performed at each of the case study operations.

Dr. J.A. Finch provided extensive input and reference material on flotation, section 4.3, as well as valuable discussion and direction on methodology to collect and analyse the data on operating rod mills in industrial plants, section 3.1 and Appendix C. Drs. M.H. Moys, J.A. Finch, and A.R. Laplante all had extensive input through discussion to arrive at the process control strategy for ball milling described in section 4.4.

Assistance with the grinding and particle sizing testwork at McGill University was provided by Mr. M. Leroux, Mr. C.J.

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NOTATION

c	Gy's composition factor
Cs	mill critical speed, fraction or percentage
C.L., N	circulating load ratio
d	Gy's particle size factor
d50c	hydrocyclone 50% separation size, corrected
D	mill diameter, inside liners
Dc	hydrocyclone inside diameter
Di	hydrocyclone inlet diameter
Do	hydrocyclone vortex finder diameter
Du	hydrocyclone apex diameter
E	line voltage
Eff.	motor efficiency
EF4	feed size efficiency factor, rod milling
EF6	reduction ratio efficiency factor, rod milling
f	Gy's liberation factor
F	80% passing feed size, Bond laboratory work index test
Fo	optimum feed size, 80% passing
F.L.	full load, motor power
F80	80% passing size of grinding circuit feed, by mass
g	Gy's size distribution factor
Grp	grams per revolution in Bond laboratory work index test
h	hydrocyclone free vortex height
I	amperage
kW	kilowatt draw of mill at the pinion
kWr	kilowatt draw per unit mass of rods in the mill
K	constant
K50	50% passing size of any mass distribution
K80	80% passing size of any mass distribution
l	Gy's liberation factor
m	sharpness of separation, Plitt's hydrocyclone model
M	solids mass flowrate
Ml	lot mass
Ms	sample mass
p.f.	motor power factor
P	80% passing product size, Bond laboratory work index test
P_1	screen closing size, Bond laboratory work index test
P80	80% passing size of grinding circuit product, by mass
Q	distance from inside top of mill to charge level
Qc	hydrocyclone volumetric flow rate
R	rod diameter
Rr	reduction ratio
Rro	optimum reduction ratio
R.P.M.	revolutions per minute

s.f.	motor service factor
S.G.	specific gravity
S_E^2	variance due to experimental error
S_S^2	variance due to sampling error
S_T^2	total variance
S^2 (FE)	Gy's fundamental sampling error
Vp	fraction media filling of mill volume
W	work applied per unit mass
Wio	Bond operating work index
W.I.	Bond laboratory work index
X	new feed rate of solids to circuit
ρ	mill charge bulk density
ρs	density of solids
ρl	density of liquid

CHAPTER 1

INTRODUCTION

1.1 GENERAL REQUIREMENTS FOR ACHIEVING EFFECTIVE PERFORMANCE FROM PLANT GRINDING OPERATIONS

In addition to striving to meet production quotas, the process engineer in a mineral concentrator must deal with a multitude of interrelated issues to achieve effective performance from the grinding operations. These issues may be broadly categorized into one or more of the following three areas of plant process engineering.

1. Design, or the selection of suitable design and operating variables in order to achieve both desired product characteristics and efficient circuit operation. Examples would be selection of hydrocyclone dimensions, grinding media size, pump speed, and grinding mill water addition rate.
2. Control, to maintain the desired circuit product specifications, as well as the operating conditions which provide efficient circuit performance throughout the inevitable variations in the circuit inputs. Examples would be control of the circuit product sizing, cyclone feed sump level, and ball mill feed percent solids.
3. Measurements and data collection, or development of the information required to assess overall circuit performance to verify that production, design, and

control objectives are being met, and to assess the effects of either externally imposed or intentional experimental changes to the circuit. Examples would include determination of circuit mass flow rates, size distributions, ore grindability, grinding power and media consumption, and operating costs.

Note that lack of attention to any one of these areas can significantly diminish the effectiveness of the others.

Numerous factors or paramaters may be used to characterize and model grinding circuit performance. These range from simple process stream characteristics (such as size distribution 80 percent passing size, mass flowrate, or solids specific gravity), to complex mathematical functions describing unit process operations (such as hydrocyclone selectivity, or ball mill specific rates of breakage). In order to be of greatest practical merit in the industrial operation, the parameters which are used to characterize and model grinding circuit performance should satisfy a number of criteria.

1. They should be conceptually meaningful, and preferably simple enough to be understood and used by all concerned personnel.
2. They should be measurable, or it must be possible to calculate them from available measurements, to a reasonable level of accuracy. Ideally, the statistical reliability should be known. For comparative purposes, relative (rather than absolute) quantities are of greatest significance.

3. The system of technical paramaters can only be described in the specific terminology of the equipment and processes. However, it must be possible to relate the technical parameters to the economics of the grinding process, as well as the overall economics of the plant. This is a necessity to determine optimum process conditions, as well as to develop reliable cost-benefit information so that process improvements requiring capital expenditures can be evaluated.
4. Once a desirable performance condition or parameter value has been identified, it must be possible to obtain it either directly or by manipulation of available design, operating, or control variables to achieve the desired result.

Steady-state grinding circuit modelling systems, when used for the purpose of industrial grinding circuit performance evaluation and improvement, may be judged in terms of how effectively they satisfy these needs.

1.2 EXISTING GRINDING CIRCUIT MODELLING SYSTEMS: STRENGTHS AND WEAKNESSES

1.2.1 General

The technical literature provides two schools by which evaluation of the steady-state performance of conventional grinding circuits may be addressed. Historically the first of these, generally termed the "Bond" (1952) methodology after its well known founder, quickly became and remains today the standard pragmatic basis for characterizing the grindability of ores and for quantifying the overall performance of rod and ball mills. The second, referred to as "population balance modelling" or "grinding as a rate process" is acknowledged to have been conceptually introduced by Epstein (1948), but became popular in the comminution research field with the growth of data processing capabilities in the late nineteen-sixties, and has now established itself as the traditional approach to the study of grinding.

1.2.2 The Bond Methodology

1.2.2.1 The Bond Law of Comminution

Industrial grinding circuit efficiency can only, at best, be described in relative terms, since "true" efficiency, such as measured by net changes in surface energy before and after breakage, is but a small fraction of the applied work, most of it being dissipated as heat. With mechanical power as the means, and size reduction as the objective, the relative efficiency for reduction of a given material from the same feed size to the same product size can be seen to be linearly proportional to the mass broken (directly), or the power consumed (inversely). Changes in the material composition or feed and product size distributions require more complex treatment to assess size reduction efficiency.

Bond (1961) made the observation that size distributions, as naturally produced through stages of comminution, were highly consistent, and therefore could be reasonably accurately represented by a single point, such as the 80% passing size.

Secondly, from numerous observations, he derived the following approximate empirical relationship between the work applied (per unit mass) in any particular size reduction circuit with one comminution device (crusher, rod mill, or ball mill), and the amount of size reduction achieved:

$$W = \text{constant} \left(\frac{1}{P80^{0.5}} - \frac{1}{F80^{0.5}} \right)$$

W = work per unit mass
F80 = 80% passing size of feed
P80 = 80% passing size of product

Defining the constant as the work required to go from a feed size F80 equal to infinity to a product size P80 of 100 um results in the familiar "Bond operating work index" equation:

$$W = Wio \left(\frac{10}{P80^{0.5}} - \frac{10}{F80^{0.5}} \right)$$

W = work, kw-hr per tonne
F80 = 80% passing size of feed µm
P80 = 80% passing size of product µm
Wio = Bond operating work index

This equation is the mathematical expression of Bond's law of comminution, which states that the feed and product sizes are related to the per unit work applied in inverse proportion to their (80% passing) sizes to the power 0.5, and in proportion to a constant (Wio) for the material in that type of size reduction circuit. From one set of operating data (feed and product sizes, and mill power draw), Wio can be calculated. It is then possible to predict changes in product size, say, for a given change in power draw, assuming Wio does not change either with the reference product size or because of a change in the ore. The Bond law thus makes it possible to account for variations in feed and product sizes when carrying out comparative efficiency determinations for a particular stage of comminution on a given material.

1.2.2.2 Grinding Mill Sizing

Bond also made use of bench-scale laboratory tests (one for rod milling, and one for ball milling) which could be used to estimate the value of Wio (at a specific product size) for the material tested when ground in a full scale 2.44 metre or 8 foot (nominal) diameter overflow mill operating under a given set of standard conditions. By equating the work applied in the 2.44 metre mill to the number of revolutions to obtain the same size reduction (that is, the same product size) in the bench-scale circuit, he provided a means of mill scale-up, and established a simple means of quantifying the relative resistance of breakage of different materials. This ore characteristic, obtained by a standardized feed preparation and laboratory grinding procedure, is termed the Bond laboratory ball mill (or rod mill) work index, Wi.

For grinding equipment selection, the application engineer's basic objective when using the Bond methodology is to satisfy the relationship:

Power required for grinding = Power draw of the mill.

For convenience, and to allow for differences in motor efficiency, the mill pinion shaft was selected as the standard reference point (losses in the gear and pinion, and mill bearings in large mills are negligible). The left hand side of the above equation (in the standard 2.44 metre plant-scale mill) is calculated from the Bond work index test. A mill (or mills) of a suitable length, steel loading and speed is then selected to draw the total power requirement in the right hand side of the equation.

For mills other than 2.44 metre in diameter, and other "non-standard" operating conditions, certain correction factors have to be applied to the estimated power for grinding, or else the reasonable accuracy of the procedure is lost. These have been modified and updated as additional data has been collected, and as milling practices evolved with time. A recent list includes the following (Rowland, 1982);

1. nature of the crushing circuit (open or closed) before the rod mill;
2. dry grinding;
3. open circuit ball milling;
4. mill diameter;
5. oversize feed;
6. extreme product fineness (below 74 µm);
7. extreme reduction ratios.

Appropriate correction factors are then applied for sizing industrial equipment for a broader range of conditions.

Note that this procedure was developed for sizing of industrial-scale equipment from a bench scale grinding test, and that the correction factors reflect Bond's perceptions of sources of inefficiency (or lack of conformance with his basic law) from a large number of laboratory tests and plant operating data. Other factors (for example, for circulating load, classifier efficiency, media size, pulp dilution) were identified by Bond (1953), but because of difficulties with their measurement, application, or just to simplify the mill selection procedure, they were not introduced.

1.2.2.3 Grinding Circuit Performance Evaluation

An overall measure of grinding circuit efficiency can be obtained by comparing the Bond laboratory test work index to the calculated operating work index of the plant circuit on the same feed material (Rowland, 1976). Since higher circuit efficiency yields a lower operating work index, the test work index in the numerator over operating work index in the denominator yields a relative grinding efficiency, with 1.0 or 100% corresponding to perfect conformance with Bond's scale-up criteria. Relative higher or lower efficiency ratings would then compare favourably or unfavourably with the Bond standard.

A "corrected" operating work index can be calculated by factoring out the inefficiency factors mentioned above for direct comparison with the test work index (Rowland, 1973 and 1976). This adjusted figure, however, is less meaningful for performance evaluation as it no longer pertains to the actual operating efficiency of the circuit. Although the uncorrected figure is preferred and used throughout this discussion, either will suffice if used consistently.

The basic strategy for using the Bond work index approach to improve grinding circuit efficiency is described in work by Rowland (1976). Plant and laboratory tests are first carried out to determine the operating and test work indices, and an efficiency rating is calculated for the circuit. Then, a process design or operating change is implemented (or one is observed in a parallel circuit), for example, a different mill speed. The efficiency is then remeasured. Should the circuit show improved relative work index efficiency, the change would be taken as a positive one.

1.2.3 Population Balance Modelling

1.2.3.1 General Concepts

Treatises on the population or size-mass balance modelling approach to the study of the steady-state performance of grinding circuits are numerous. The reader is referred here to two excellent summary descriptions by Austin et al (1982) and Herbst and Rajamani (1982), and to the more extensive work by Lynch (1977) and by Austin et al (1984). The purpose of this account is to outline the basic principles and methods involved so that their utility in the context of plant grinding circuit performance evaluation and improvement can be assessed. The ball mill circuit models for wet grinding and hydrocyclone classification are described and critiqued here. Although similar concepts are applied to rod mill modelling (Calcott and Lynch, 1964; Grandy and Fuerstenau, 1970; Mular and Henry, 1971; Fournier and Smith, 1972; Heyes et al, 1973; Shoji and Austin, 1974; Lynch, 1977), their development and use has been far less extensive.

The ball mill is considered a reactor in which breakage of particles is conceived to be a rate process. A given size distribution of material is fed to the mill. The particle sizes are broken at different rates, according to the ore's resistance to breakage and the breakage environment. The breakage rate is heavily size dependent, as shown in Figure 1-1, and normally considered to be first-order, with respect to the amount of the particular size class present in the mill. When a particle breaks, it does so according to a fracture pattern which yields a characteristic array of daughter fragments. The size distribution of these fragments is called the breakage function (Figure 1-2). Breakage and rebreakage of particles occurs until the material is discharged from the mill. Their residence time characteristic is determined by the volume and the mixing characteristics of the slurry contained in the mill, and is usually modelled by combinations of plug flow and/or fully mixed tanks in series (Figure 1-3). The size distribution of the solids discharge is thus determined by the sizing of the feed, breakage rates, the breakage function, and the residence time distribution.

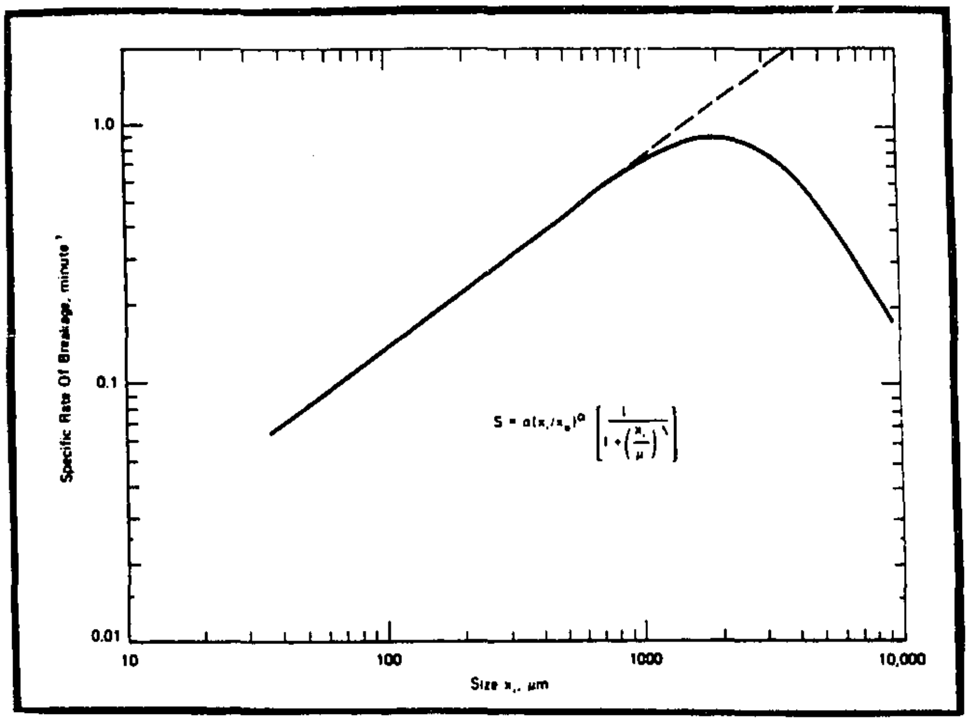


Figure 1-1. The Breakage Rate Function (Austin et al, 1984)

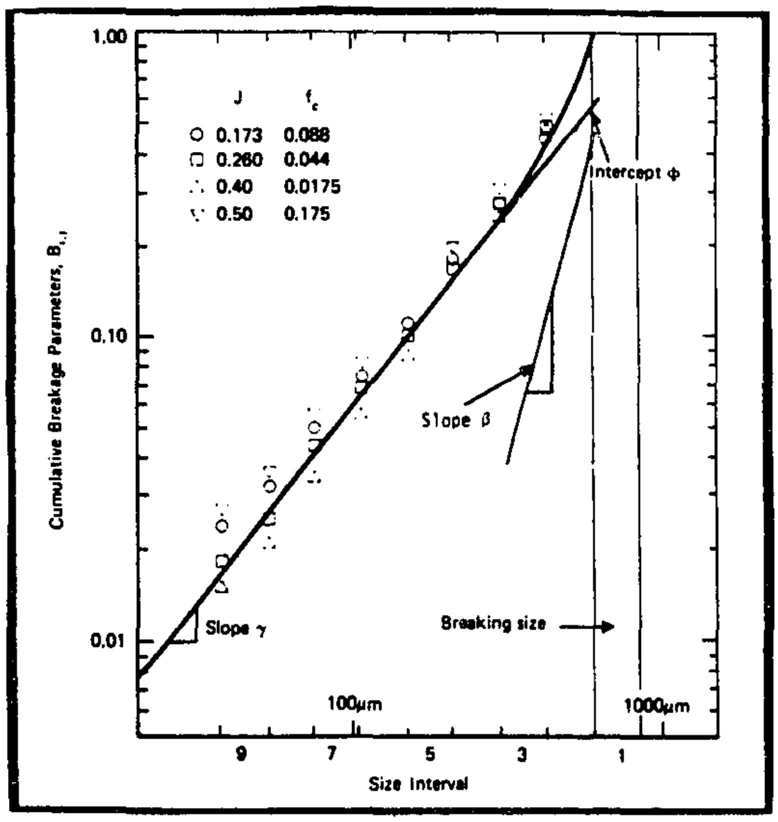


Figure 1-2. The Breakage Function
(Austin et al, 1984)

In closed circuit grinding with hydrocyclones, the classifier separation performance is characterized by the selectivity curve (Figure 1-4). It describes the probability that a particle of a given size in the cyclone feed will report to the cyclone underflow. The shape and position of this curve is determined by hydrocyclone dimensions and feed characteristics and conditions, and permits calculation of the cyclone product mass flows and size distributions from a given feed.

1.2.3.2 Applications

Industrial plant applications of steady state population balance modelling techniques can be divided into three levels or categories according to the purpose and depth of investigation. The first would involve model parameter estimation for circuit problem diagnosis, or to test the effect of a change in an operating or design variable. The purpose is to examine one or to compare two or more sets of steady state operating data to identify conditions associated with improved performance. The second is circuit simulation for optimization, in which interactions between continuous ranges of design and operating conditions and process performance parameters are built into a system of mathematical models which attempt to mimic the real system. Many variables and combinations of variables can then be examined off-line to once again seek conditions of improved performance. For both levels of investigation, improved plant performance ultimately means

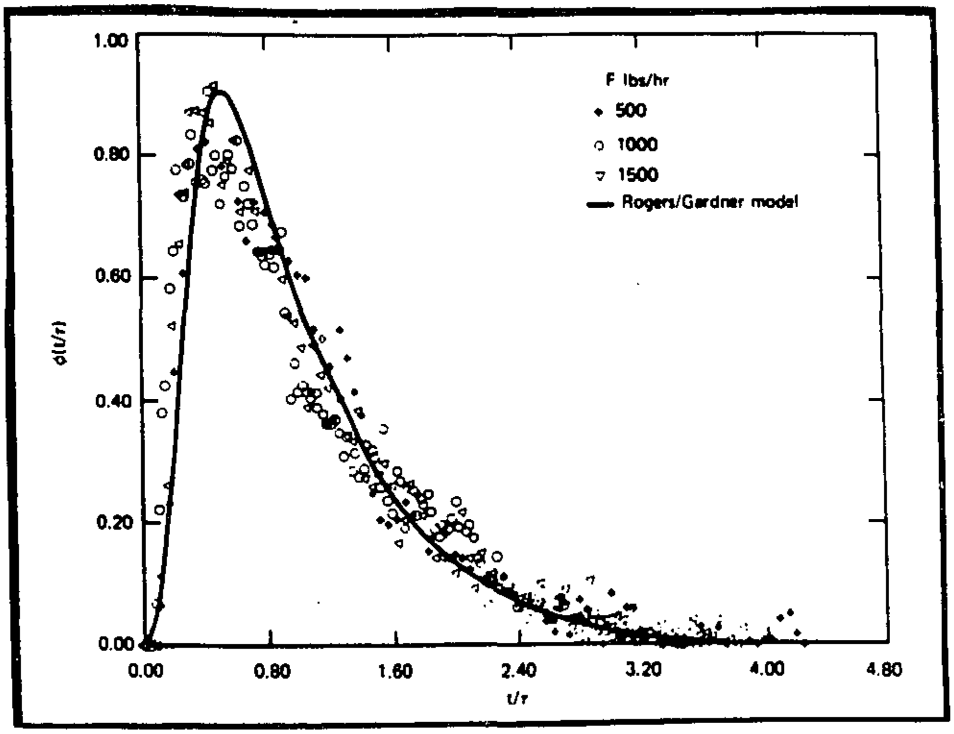


Figure 1-3. Ball Mill Residence Time (Austin et al, 1984)

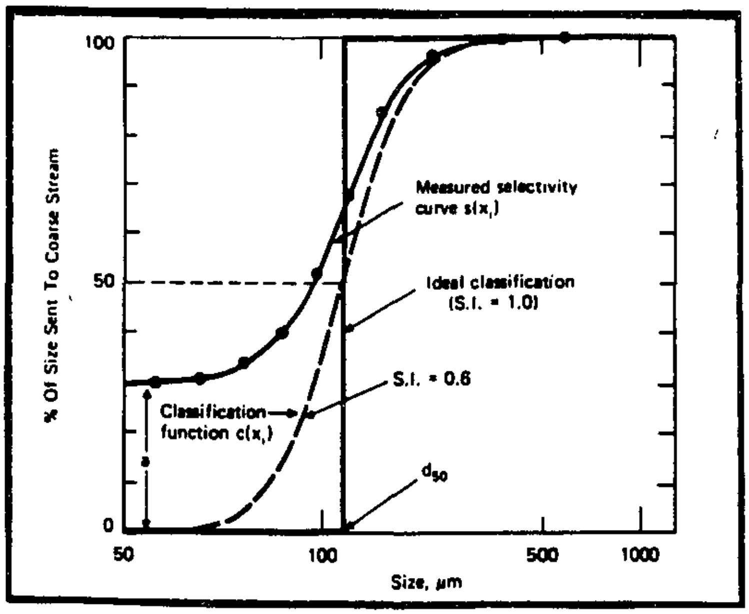


Figure 1-4. Hydrocyclone Selectivity Curve (Austin et al, 1984)

higher tonnage, a finer grind, or improved grinding circuit product quality (less oversize or slimes present). What may be considered as a third level of sophistication is the use of rate process parameters and simulation for scale-up design and analysis of industrial grinding circuits.

Level I: Parameter Estimation

Ball mill circuit model parameters for circuit performance evaluation are obtained from sampling data from one or more surveys. Mill feed and product size distributions are measured. The mill residence time characteristic may be measured using a liquid tracer test, or calculated from estimation of the hold-up volume and mixing characteristics of the mill. The breakage functions can be determined by short duration batch grinding tests on the material in the laboratory, or assumed to fit normalized breakage functions which are characteristic for the ore type. The breakage rate function is then back-calculated.

Hydrocyclone model parameters are calculated from the mass balance of solids, water, and solids size distributions around the classifier. The water and feed solids bypass fraction can be estimated, and cut-size and sharpness of classification parameters determined for the reduced cyclone separation curve.

The following are examples of the first type of work, which often involves trying to detect a change in the breakage rate function which would indicate an improved environment for grinding inside the mill: use of smaller balls in a secondary mill at Mt. Lyell was shown to cause breakage rates to increase by an average of 70% (Hartley et al, 1983); Spring et al (1984) showed that calculated breakage rates remained virtually constant over the normal operating range of circuit feed size, feed rate, and mill density; Kelsall et al (1972) showed that improved classification resulting from rearrangement of the ball mill circuits from parallel to series resulted in a reduction of oversize to flotation despite what was an unexpected negative shift in the breakage rates for both mills. Laplante (1987) demonstrated a poor flow split to two ball mills in parallel operation through back-calculation of the mill mean residence time from assumed equivalent breakage rate and breakage functions.

Level II: Circuit Simulation

At the second level of investigation, simulation for optimization, three steps are involved in the overall strategy. First, models of the unit processes (the ball mill and the hydrocyclone) are established. This may involve extensive experimentation, or less exhaustive estimation of parameters from operating data to fit accepted, known models. Second, the models are combined into a circuit simulator. Third, the simulator is used to predict circuit performance when design or operating variables are altered, and thus to find ways to improve circuit performance.

What is believed to be the first use of such a plant grinding circuit simulator was by Lynch and colleagues at Mount Isa Mines (Lynch et al, 1967). They showed by simulation that changing three parallel ball mill circuits to two primary circuits and one secondary circuit in series would result in improved performance. The change was implemented, and it became possible to increase tonnage by 10 percent at the same product grind size. Both the scope and published description of this classic work are almost unique in that simulator based recommendations were actually implemented in the plant with apparent success.

Cameron et al (1971) used computer simulation of the Broken Hill South grinding circuits to produce evidence of the capability to handle increases in fresh feed rate. Digital simulation of the Gibraltar grinding circuit was carried out to evaluate the effect of feed rates, the dimensions and number of operating hydrocyclones, and alternative circuit arrangements (Allen et al, 1973). Hydrocyclone and flowsheet modifications were investigated in a similar fashion at Heath Steele Mines (Hodouin et al, 1978). A general circuit simulator has also been used to evaluate the effect of improved classification, circuit arrangement changes, and the use of a coarse flotation cell inside a grinding circuit at Mount Isa (Manlapig et al, 1985).

Level III: Scale-Up

The use of scaled-up breakage rate and breakage function parameters from laboratory batch tests combined with computer simulation to characterize the performance of industrial circuits has been proposed by at least two research groups. Herbst and Rajamani (1982) assume breakage functions to be invariant for constant mill speed, ball load, and particle load conditions, and that breakage rate constants are directly proportional to the specific power input. Selecting the size of a new mill then can be reduced to meeting power draft requirements, and they revert to the same mill power draw relationships used and developed by Bond (1961) and Rowland (1982). Austin et al (1982) correct the breakage function from small-scale tests according to the laboratory versus plant ball size mix, and breakage rates are scaled using factors for mill diameter, ball sizes, mill speed, and ball and solids loading. In both cases, batch tests are performed either dry or wet, according to the plant application.

1.2.4 Summary of Advantages and Disadvantages

The major advantages of the Bond approach to plant grinding circuit evaluation can be summarized as follows.

1. It is simple. Calculation of the operating work index requires the determination of only 4 data, the circuit solids feed rate, the feed 80% passing size, the mill power draw at the pinion, and the product 80% passing size.
2. It provides a measure of overall grinding efficiency. The ratio of the work index as determined in a standard laboratory test circuit compared to the

operating work index of the plant circuit provides a single, overall efficiency rating of the plant circuit. The resistance to breakage of the ore is taken into account. This provides a clear overall performance objective and means of measurement of efficiency improvements.

3. It provides a standard for comparison of both the grindability characteristics of different ores, and for the operating efficiency of different industrial installations.
4. It provides a useful link to grinding economics. The Bond formulation is based on mill power draw, which in turn is controlled primarily by the level of the steel load in any given fixed-speed mill. Since power consumption and media consumption together dominate direct, variable grinding costs, there is a direct link to plant commercial performance. Savings in power and steel consumption provide the basis for estimation of the minimum value of an efficiency improvement. Exploiting an efficiency improvement by increasing tonnage or by obtaining a finer grind, when feasible, may provide a greater economic benefit to the plant.

The major disadvantages to the Bond approach are as follows.

1. It is too simple to provide significant insight into what is a complex process. Stemming from its first

two strengths is the obvious shortcoming that, as a "lumped" parameter, the relative work index efficiency does not identify or characterize all the specific design and operating variables which affect circuit performance. Some important factors, such as media utilization, classification efficiency, and slurry rheology are not even taken into account.

2. Its level of accuracy is not clearly defined. Rowland (1973) has compared 16 laboratory tests and corrected operating work indices for operating ball mill circuits (Table 1-1). These data display a 95% confidence interval of approximately plus or minus 20% on estimated versus actual mill power requirements based on the Bond scale-up method. However, the possible sources of error (sampling, experimental, and scale-up) and means to minimize them are not well defined. It is clearly important to know and to minimize relative efficiency measurement error for efficiency changes to be detected and quantified.

The two primary advantages of population balance modelling techniques are quite clear.

1. It has scope for dealing with a complex process in a complex fashion. A number of process performance parameters are utilized which may help troubleshoot and suggest methods to improve circuit performance. Dealing with complete size distributions throughout

Table 1-1. Comparison of Corrected Operating and Laboratory Test Work Indices (Rowland, 1973)

Corrected Operating Work Index	Grindability Test Work Index	Ratio of Operating to Test Work Index
11.4	12.8	0.89
9.81	10.13	0.97
10.08	10.16	0.99
13.00	12.5	1.04
13.03	12.2	1.07
12.31	12.27	1.00
10.77	10.1	1.07
9.63	10.2	0.94
14.5	14.61	0.99
11.48	8.9	1.29
10.71	11.2	0.96
9.77	11.2	0.87
5.34	5.99	0.89
5.96	6.26	0.95
11.78	13.34	0.88
13.17	13.18	1.00
	Mean	0.9875
	Variance	0.0098

the entire grinding circuit, rather than single point values for only the circuit feed and discharge, is an enormous step forward. Numerous operating and design variables can be incorporated, and the complicated interactions between them dealt with by sophisticated mathematical models and computational methods.

2. It has powerful potential for controlled, off-line experimentation. Background noise associated with unmeasured or short-term random variations makes interpretation of data from plant experiments very difficult. The computer simulator provides a clean,

noiseless, completely operator controlled environment in which even fairly rudimentary unit models can demonstrate the general trend of effects from changes in the process inputs. This usefulness as an instructional medium applies more in the general sense than it does to site specifics.

Clearly, the need for more detailed process performance characterization than provided by the Bond methodology is at least partly met by population balance techniques.

There are three major disadvantages of population balance modelling of ball milling, in particular with reference to plant circuit simulation for optimization and the related need for scaling of rate process parameters.

1. It is conceptually and computationally complex.
2. The level of uncertainty of simulation results is both significant and undefined.
3. There is no readily apparent overall measure of grinding efficiency or direct association with grinding costs, as exist with the Bond approach.

Development and use of a plant circuit simulator requires the combination of two expert skills. Both the process unit operation models and the computational methods to estimate circuit steady-state conditions must be understood in terms of their capabilities and limitations in simulating the real circuit. The high degree of sophistication and complexity of the task of circuit simulation requires skills that are not readily available to many operations.

There are multiple sources of error in grinding circuit simulation, and their magnitudes are not well defined. Sampling and experimental error are of course present in any methodology. However, it is the nature of the models themselves that is of greatest concern. For example, ball mill and hydrocyclone simulation models normally are not developed to deal with heterogeneous materials on a mineral by mineral basis. The individual ball mill parameters may also be inaccurate. Liquid residence time characteristics are assumed to apply to the solids, although preferential retention of coarse, heavy particles in some ball mills has been observed and documented (Davis, 1945; Myers and Lewis, 1950; Austin et al, 1982). The breakage function, measured in a laboratory mill, must either be assumed invariant, or scaled to plant size equipment. This implies that the relative proportion of active breakage mechanisms (e.g., impact, abrasion, and chipping) and their characteristic product size distributions can be extrapolated to those in the environment of the full-scale equipment. Breakage rates, when back calculated from plant operating data on the basis of estimated or assumed breakage functions and mill residence time characteristics, absorb all the inherent errors to make all three parameters suit the mill feed and product size distribution data (or vice-versa). If breakage rates are determined in the laboratory, as suggested by Austin et al (1984), then the problem of scale-up once again arises.

Relative back-calculated breakage rates may serve as a useful parameter for rough circuit performance evaluation. They can also be used to provide rough estimates of the ball mill discharge size distribution under varied mill feed conditions. This is because the results of the "forward" calculation are insensitive to the exact form of the breakage function or residence time characteristic used, as long as the relative mean residence time is reasonable. Even greatly simplified formulations can produce reasonable results (Laplante et al, 1986).

However, the comparisons between sets of back-calculated breakage rates involves a substantial fundamental uncertainty. The breakage rate function must be recognized as a lumped parameter which is dependent on both the efficiency of the process and the grindability characteristic of the ore itself. Just as the test work index of the ore must be factored out of the operating work index before the circuit efficiency can be estimated, the grindability characteristic of the ore must be factored out of the breakage rate function before a reasonably accurate measure of the efficiency of the breakage process can be derived. Otherwise, error due to ore grindability variations is inherent.

An important case in point is the original work by Lynch et al (1967) at Mt. Isa Mines mentioned earlier. Upon reviewing the operating data provided, Kelsall and Stewart (1971) have concluded that a major change in the ore was the only probable cause for a significant change in the breakage rates after the circuit reconfiguration. They concluded that "to date, there is no method of predicting accurately the change in performance of a complex grinding circuit which will result from a major change in circuit layout, and confirmation of any apparent change must include a separate assessment to establish whether or not the grindability of the ore remained unchanged."

The need for independent verification of ore grindability suggests once again the need for laboratory measurements of breakage rates. The problem is that an accurate full-scale modelling system is then required to verify scale-up of breakage rates, or accurately known scale-up of breakage rates is required to verify the full-scale modelling system. At the present level of their development, then, the group of parameters used in population balance modelling are collectively unverifiable. Because there are still a number of possible sources of significant error in the simulation process (e.g., the models, ore variations, and scale-up), it may be difficult to pinpoint the causes of discrepancies between predicted and actual plant results as shown by experience at Broken Hill South (Kelsall et al, 1972) and more recently at Brunswick Mining (del Villar et al, 1985).

The process unit models needed for accurate ball mill circuit simulation thus require further development to become an effective tool for the plant engineer. Even though present sampling and mass balancing techniques may provide highly dependable plant operating data, procedural standards for parameter estimation and for model and simulator building have therefore not yet been established.

1.3 DISCUSSION AND DEFINITION OF STUDY OBJECTIVES

1.3.1 General Approach for Technoeconomic Analysis

The observed lack of widespread industrial application of population balance modelling as methodology for technical and economic improvement of grinding operations was first thought to be largely associated with the difficulty of relating the technical parameters used in the modelling system to plant economic performance. A combined approach, using both the Bond work index analysis and computerized circuit population balance modelling, in particular for closed-circuit ball milling, was seen to be at least a partial solution to this problem. Circuit operating data could be obtained for both modelling systems simultaneously. It was therefore planned to construct a circuit simulator which would be used in the usual way to find ways to improve grinding circuit performance. When a process change indicated improved grinding performance on the simulator (a finer grind or higher tonnage, or some combination of both, at the same mill power draw), the associated improvement in work index efficiency would then be calculated to yield a single quantitative measure of improved grinding performance. This measure would also permit calculation of savings in grinding power and steel consumption, assuming that the improved efficiency was utilized to reduce the mill power draw and charge level at the original tonnage and grind. In this way, at least the minimum economic benefit could be estimated. If the efficiency improvement was utilized to obtain a finer grind or higher tonnage, it would only be on the basis of an even better economic return. Even though the effects of grinding circuit changes on plant metallurgical results may be difficult to evaluate, the minimum economic return could be established based on operating cost reductions. It may be possible in this way to justify the capital expenditures required to implement grinding circuit improvements in the plant.

The focus on operating cost reductions was also supported by broad experience at the case study operations, where tonnage was generally limited by underground mining operations. Changes in operating costs are typically much easier to measure and to associate with specific plant changes than are changes in plant metallurgical performance, which is influenced by a multitude of factors. Dollar for dollar, operating cost reductions are therefore considered by management to be much more meaningful than claimed increased revenues due to improved plant metallurgy (i.e., grade or recovery).

This emphasis on grinding cost reductions was not to say that the effects of grinding circuit operation on plant metallurgical performance could be ignored. For the economic analysis to be complete, in addition to all associated operating costs, the value of each unit of the grinding circuit product as flotation circuit feed must be taken into account. One approach to this problem has been suggested by Trahar (1981), who carried out widespread investigations of particle size by size behaviour in both laboratory batch and plant flotation cells. From this he recommended that grinding and classification should be designed to place as much valuable mineral as possible within the size range associated with high flotation rates. Behavior of discrete size fractions in flotation was also adopted by Apling et al (1982) in their development of a combined comminution-classification-flotation model. Hence, it was decided to develop size by size flotation circuit performance as part of the data collection in this investigation. This and other approaches to interfacing of grinding and flotation operations are discussed in greater detail in chapter 4.

1.3.2 Grinding Circuit Modelling

Review of the Bond and population balance modelling approaches for circuit performance evaluation revealed several problems with present grinding circuit steady-state modelling, as discussed earlier. First, circuit performance is normally evaluated by comparisons of steady-state operating conditions and corresponding calculated model and performance parameters determined from as few as two short-term plant surveys. When the ore can vary from one test to the next, the statistical reliability of the laboratory and operating work indices from the test periods must be known. As well, the error must be small enough so that small changes in efficiency can be detected. Otherwise, extensive or long-term testing may be required to demonstrate an effect. Rowland (1976) used the results from 14 grindability tests and four months of composite corrected operating work index data from four grinding circuits to arrive at an estimated 7 percent average higher efficiency for the higher of two mill speeds (see Table 1-2). Note that during any given month, the measured efficiency of one of the lower speed mills occasionally equaled or even exceeded that of a higher speed unit. Estimation and minimization of the error associated with work index determinations was therefore selected as one of the objectives of this study.

Table 1-2. Rowland's Work Index Efficiency Comparisons for Ball Mills at Different Speeds (1976)

	68% of Critical Speed		73% of Critical Speed	
	Mill A	Mill B	Mill C	Mill D
January	96.5%	81.8%	99.2%	-
February	103.0	109.9	109.9	-
March	85.0	99.2	112.0	90.8
April	103.6	102.3	103.4	113.5
Average	97.7		104.8	

A second and much broader problem is that of relating specific design and operating variables to the overall performance of the grinding circuit. For the open circuit rod mill, it was felt that the traditional empirical approach of investigating cause and effect relationships between changes in design and operating variables and overall circuit efficiency could prove fruitful as long as the circuit efficiency measurements were accurate enough to detect the effects of the changes. Economic effects could then be evaluated based on the capital or operating cost of the change itself, plus cost changes due to the change in efficiency. For example, changing to a smaller rod size might increase rod milling efficiency, but increase rod costs (per unit weight delivered to the plant) and consumption rate per unit of mill power draw. However, the improved efficiency would permit operating at a reduced charge level and power draw to achieve the same grind, with associated reductions in both power and steel consumption. The net economic effect would depend on the quantitative values of each cost.

For closed circuit ball milling, it was seen that detailed analysis of the effects of changes in design and operating variables was extremely limited using the simple empirical approach of Bond. As well, circuit simulation using current population balance modelling techinques did not appear to be a satisfactory alternative. However, during this work, it was discovered that it may be possible to characterize overall ball mill circuit classification efficiency with a single parameter, which itself could be related to either individual circuit design and operating variables and overall circuit efficiency. It was subsequently discovered that once circuit classification efficiency was isolated, overall breakage efficency could likewise be quantified, and that the two together determine the overall circuit efficiency. The premise that this intermediary level of ball mill circuit modelling can be used to link overall circuit performance with specific circuit design and operating variables is discussed in greater detail in chapter 4. Presentation of this system is the primary objective of this dissertation.

CHAPTER 2

THE INDUSTRIAL CASE STUDY OPERATIONS

2.1 OVERVIEW OF PLANT TESTWORK PERFORMED

Owing to the general nature of the objectives of this study, it was advantageous to investigate more than a single operation. Therefore, the work was carried out in conjunction with two plant case studies, at Les Mines Selbaie in Joutel, Quebec, and at Kidd Creek Mines in Timmins, Ontario.

Both these plants use rod and ball milling following multi-stage crushing before flotation. The Selbaie circuit is simple in basic design, using one stage of ball milling, followed by the flotation of a single copper concentrate with no regrinding. The Kidd Creek circuit utilizes secondary ball milling after rougher copper flotation, but before zinc flotation, as well as both copper and zinc regrind milling. This circuit is further complicated by a number of recycle stream options.

General circuit information and historical operating data available at each plant was collected and reviewed (Appendices A and B). This was done to consolidate circuit design information, to carry out a preliminary assessment of circuit performance based on available information, to establish existing data sources, and to identify grinding costs so that areas of investigation could be prioritized. General circuit surveys were then carried out at each location. These were needed to establish data procurement procedures, as well as to provide further basic operating information, although performance criteria were not yet defined. The selected information objectives were therefore of the most general nature, guided by traditional circuit sampling practices. These surveys are described in sections 3 and 4 of this chapter. Data was also gathered on the size by size behaviour of minerals in the flotation circuits of each operation.

Subsequent plant testwork was aimed at specific circuit performance assessments relative to key design and operating variables which were identified and prioritized through the review work. Most of this work concentrated on the rod mills, as the front end of the grinding circuit was deemed the logical starting point for implementing changes. A combined rod milling experimentation program was carried out at the two plants, as described in Chapter 4. Subsequent sampling work on the ball milling circuits was limited to testing specific aspects of hydrocyclone performance.

2.2 GENERAL GRINDING CIRCUIT SURVEY SAMPLING AND ANALYSIS PROCEDURES

2.2.1 Circuit Sampling and Sample Analysis

Initial sampling runs (number 1 and 2) of the Selbaie and Kidd Creek concentrator grinding circuits were carried out in November and December, 1985. Preparatory work had included the following:

a. review of sampling techniques;
b. definition of data requirements;
c. selection of sample point locations, and modifications, where necessary;
d. determination of sample analyses to be performed;
e. determination of sample size, and number of cuts required;
f. sample cutter design, testing of sample extraction procedures, and cutter re-design, where necessary;
g. determination of mill power draw;
h. calibration checks on instrumentation;
i. investigation of circuit instability (cyclone surging);
j. design of overall sample run for minimum manpower;
k. final preparation of record sheets, sample containers, equipment, etc.

The purpose here is to provide a summary description of the grinding circuit data acquisition system.

2.2.2 Sample Cutters

General rules applied for sample cutter design and usage are as follows (Gy, 1974; Gy, 1982):

1. The ideal sample cut should be taken straight across the complete stream of material, at a constant speed.
2. The size of the cutter opening should be a minimum of 3 times the largest particle, but no less than 10 mm.

3. The maximum cutter speed, for a cutter of minimum slot width (i.e., 3 times max. particle size) is about 0.4 meters per second. This speed may be increased in proportion to the actual versus minimum cutter width.
4. The cutter receptacle must be large enough so it does not overflow when used at a suitable speed.

Several practice cuts were taken and the cutters thoroughly emptied (without rinsing) at each sample location before each run. They were likewise emptied (without rinsing) after each cut (Restarick, 1976; Smith, 1982) throughout the run. If any sample cut was not felt to be satisfactory for any reason, it was dumped and re-taken.

2.2.3 Sample Size

The minimum sample size, in general, is that amount required for;

(a) statistical validity,

(b) analysis to be performed, or,

(c) practical cutter design and use.

For determination of the basic characteristics of the material (i.e., size distribution, percent solids, specific gravity, and grindability), samples made up from a minimum of 3 or 4 cuts with properly designed cutters were found to be large enough to satisfy (a) and (b) for each of the sample points around the grinding circuit.

2.2.4 The Sample Run

The objective of the sample run is to generate one set of circuit operating data. Circuit stability (or "steady state") is required to obtain data which is collectively representative of circuit performance at any given time. In addition, a large number of randomly timed cuts is desirable for the most statistically reliable results. Based on successful experience of others, sampling periods of from one to four hours could be recommended (Mular and Larsen, 1983; Houdoin et al, 1978; del Villar, 1985; Cameron et al, 1971). In the interest of devising a practical, two-person sampling routine for the complete circuit, a two hour period, in which 8 sample cuts were taken from each location, was selected. Sump levels and instrumented flows were monitored for steadiness for several hours before surveys were commenced. As a further check for steady state, separate samples (number 1 and 2) were made up from the four cuts during each hour. If checks on the size distribution of key indicators (rod mill discharge and cyclone overflow) were found to be consistent, the other pairs of samples were combined for one set of analyses. If the key indicators were not consistent, a decision would be taken whether to treat each set of samples for each hour separately, or to scrap the run.

Complete circuit test runs were carried out using prepared record forms and involved the maximum amount of sampling and analysis work from a single run, including percent moisture determinations, screen analysis, size by size assays, solids specific gravities, and grindability testing of circuit feeds.

For most tests, screening to 38 μm was considered adequate. Finer sizing, to 5 or 10 um, was carried out by microsieving or cyclosizing, as deemed necessary.

2.2.5 Sample Analyses

All sample analyses except for grindability testing, sub-sieve sizing, and a limited amount of assaying was carried on site by plant personnel. Percent solids of each complete sample were obtained from the net wet and dry weights of samples taken. Sample containers were kept sealed prior to weighing to avoid moisture loss, and gross and net weights reported. Gross sample weights (container plus contents) were measured as soon as possible after the sample run.

In general S.G. of solids checks were carried out on all composite samples of rod mill discharge, surge tank discharge, ball mill discharge, and cyclone overflow. These tests were done on residual samples from riffling before screen analysis, after further splitting. Standard S.G. bottles were too small for anything coarser than cyclone overflow samples, unless previously pulverized.

Except for the exceptionally coarse material size at the rod mill feed, all samples were wet screened to 38 μm (400 mesh) after careful riffling down to a suitable sub-sample size of about 150-200 grams, and then dry screened down to (and including) 400 mesh. The combined (wet plus dry) minus 400 mesh was retained for further sizing (micro-sieving) when necessary, as well as assaying of all sizes, when called for. The standard Tyler series of screens was used, as follows;

1.05" (26.5 mm)	20 mesh (850 μm)
.074" (19.0)	28 mesh (600 μm)
.525" (13.2 mm)	35 mesh (425 μm)
.371" (9.5 mm)	48 mesh (300 μm)
3 mesh (6.7 mm)	65 mesh (212 μm)
4 mesh (3.35 mm)	100 mesh(150 μm)
6 mesh (3.35 mm)	150 mesh(106 μm)
8 mesh (2.36 mm)	200 mesh(75 μm)
10 mesh (1.70 mm)	270 mesh(53 μm)
14 mesh (1.18 mm)	400 mesh(38 μm)

Sieves were labeled, and the same set used for all screen analyses work. Weights, as well as percentages, of screen analyses were reported so that sample size was known for all data. The screen opening size in mm or μm as indicated on each screen as various makes may have slightly different opening sizes than those indicated.

Rod mill feed screen analysis started with a 6 to 10 kg sample, which was then treated as follows. After weighing and drying the complete sample for % moisture determination, it was riffled (if necessary) to a 6 to 10 kg sample for screen analysis. Then,

a) The sample was dry screened to 6 mesh. The -6 mesh was riffled down to about 1 kg.

b) The 1 kg. of -6 mesh was dry screened from 6 down to 35 mesh. The -35 mesh was riffled down to about 200 grams.

c) The 200 grams of -35 mesh was wet screened on 400 mesh, and then the oversize dry screened from 35 to 400 mesh.

As with screen analysis on all samples, all undersize material from wet sieving was saved to be dried and weighed. Any -400 mesh from dry screening was added to the wet screening undersize. When finer sizing was carried out, it was done by cyclosizing or by micro-sieving at 25, 15, 10 and 5 um (Finch and Leroux, 1982).

If the rod mill feed material contained an excessive amount of fines, it was pre-washed in a bucket, and the fines dried, weighed, and screened.

When required, separate samples for rod mill and/or ball mill grindability testing were taken at the rod mill feed and rod mill discharge, respectively. Approximately 8 to 10 cuts (over the 2 hour period) of rod mill feed provided 40 kg of material, enough for the Bond rod mill tests, and back up for additional or comparative testing. At the rod mill discharge, about 12 to 15 cuts are needed for 20 kg (dry) of material for Bond ball mill testing (including some back up material). During the first test runs, extra rod mill discharge material was taken so the grindability test apparatus at each site could also be calibrated against the standard Bond ball mill test equipment at McGill University.

Once steady state conditions were verified by checking the cyclone overflow and rod mill discharge size distributions, grindability samples were shipped to McGill.

Note that grindability test samples were completely separate and in addition to those taken for percent solids and size distribution. Consequently, the rod mill feed size distribution was not necessarily determined on the other sample if a rod mill grindability test was to be carried out, as it includes screen analysis.

2.2.6 Discussion

In the effort to take every practical step to ensure the reliability of data collected with the available staff and other fixed resources, almost a year of development work was carried out before any sampling of the circuits. In addition, a system of self-checks was utilized throughout the steps in the generation of the data. For size distribution and mass flow data around the grinding circuits these included:

a. calibration of mass flow instruments immediately before (or after) sample runs;

b. monitoring of mass flows both before and during the runs to ensure stability of flow well within the normal range of accuracy of the instruments;

c. initial size distribution and percent solids checks on separate "key indicator" samples to ensure stable circuit performance throughout the sampling period;

d. use of identical screening procedures and the same sieves for sizing of all samples from the same sample run;

e. use of computerized mass balance routines to check for discrepancies in the size and mass flow data.

When a visible circuit upset occurred during a sampling campaign, the run was abandoned and restarted after several more hours of what appeared to be stable operation. Fortunately, none of the major sampling campaigns had to be entirely rejected because of discrepancies in the "key indicator" samples. However, cyclone feed samples taken with great difficulty at Selbaie were found to be unreliable from the mass balance procedures, and so this data was rejected. In general, however, the minor amount of adjustment in all the other raw data shows that it can be used for circuit performance evaluation with a high level of confidence in its reliability.

2.3 LES MINES SELBAIE CIRCUIT DESIGN AND OPERATION

2.3.1 Process Flowsheet

The process flowsheet for the B zone grinding circuit at Les Mines Selbaie is shown on drawing number 11112-01 (Figure 2-1). This is a conventional two-stage grinding circuit consisting of a rod mill in open circuit followed by a ball mill in closed circuit with hydrocyclone classifiers. Rod

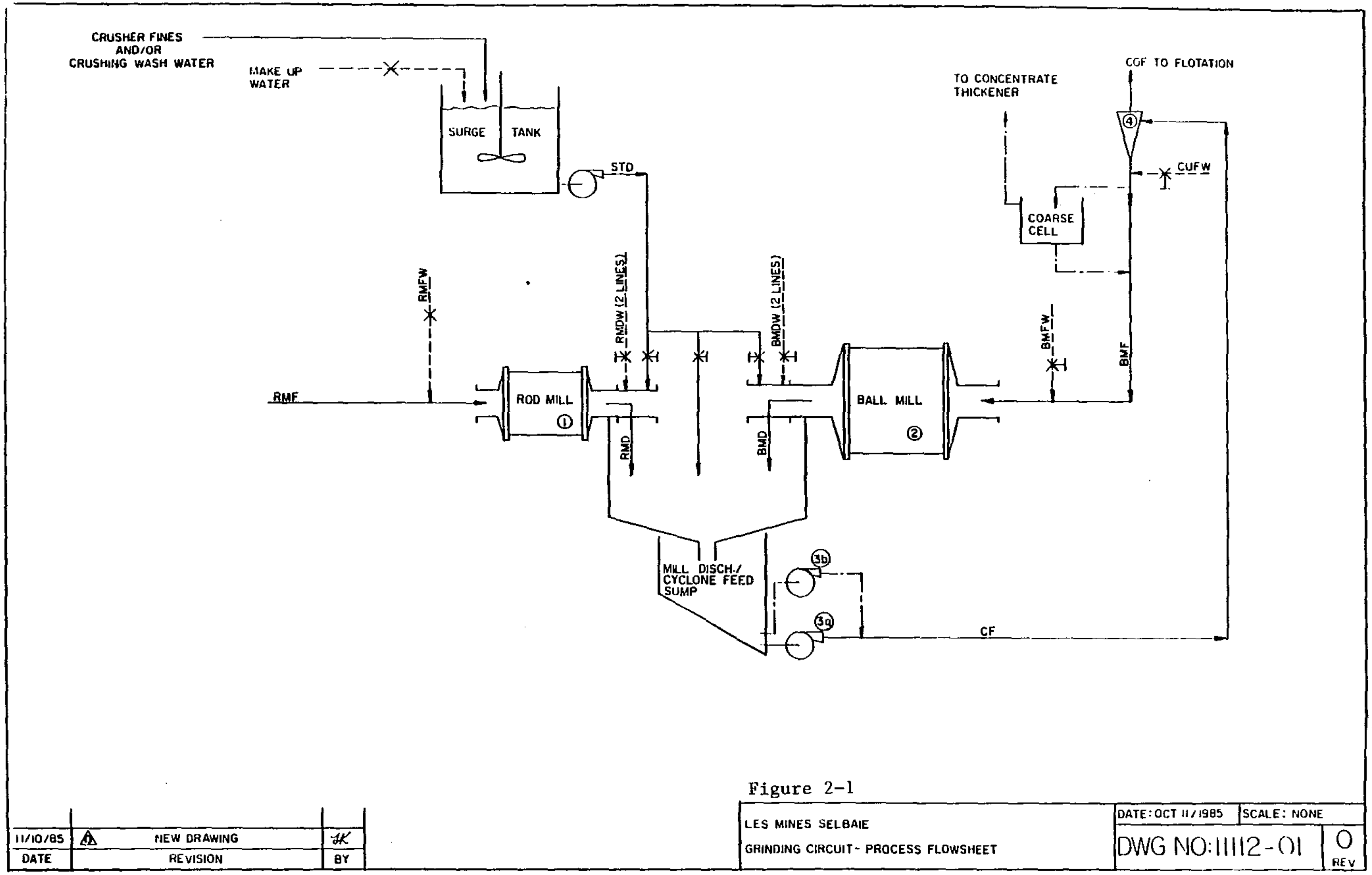

CRUSHER FINES
AND/OR
CRUSHING WASH WATER
MAKE UP
WATER
SURGE
TANK
STD
TO CONCENTRATE
THICKENER
COF TO FLOTATION
CUFW
COARSE
CELL
RMFW
RMDW (2 LINES)
BMDW (2 LINES)
BMFW
BMF
RMF
ROD MILL
BALL MILL
RMD
BMD
MILL DISCH./
CYCLONE FEED
SUMP
CF
LES MINES SELBAIE
GRINDING CIRCUIT- PROCESS FLOWSHEET
DATE: OCT 11/1985
SCALE: NONE
DWG NO: 11112-01
0
REV
11/10/85
NEW DRAWING
DATE
REVISION
BY

Figure 2-1

mill discharge (RMD) is combined with ball mill discharge (BMD) to form the feed to the hydrocyclones (CF).

Rod mill feed (RMF) is prepared by closed circuit crushing to all passing a 12.7 mm (0.5 in.) slotted deck screen. Fresh water can be added at either of the mill feed or discharge points (RMFW, RMDW, BMFW, BMDW) or at the cyclone underflow (CUFW). However, the major source of cyclone feed sump water is by way of the surge tank discharge (STD), which contains make up water, as well as crushing wash water and fines from a spiral classifier when the crushing plant is operating.

The cyclone overflow (COF), which is the final grinding circuit product, is fed to a conditioning tank before separation of a single copper concentrate in a conventional multistage flotation circuit. There is no regrinding, although another grinding mill exists and may be used for this purpose. Cyclone underflow can be directed to a flotation cell inside the grinding circuit to produce a coarse concentrate, which is added to the conventional flotation circuit concentrate stream before thickening, if desired. However, this flotation cell was not operated extensively during the course of this study.

The normal operating tonnage for rod mill feed material is 65 dry metric tons per hour. Fines from the crushing plant (operating approximately 12 hours per day) increase the total average tonnage to 73 dry metric tons per hour, for a total of 1650 tons daily with a 6% down time allowance. The general grind size objective is approximately 70% minus 75 um (200

Tyler mesh) or an 80% passing size of approximately 100 um in the cyclone overflow product.

Note the original plant design capacity was 1500 tons per day, but a daily throughput as high as 1750 tons has been achieved for extended periods when it was necessary to make up for lost tonnage. Operating at a reduced tonnage may also be necessary at certain times because of the unavailability of ore.

Details on the grinding circuit equipment, process instrumentation and control, and historical performance and cost data are given in Appendix A .

2.3.2 Size by Size Recovery of Copper in Flotation

Size distribution and assay data is routinely collected for monthly composite samples of flotation circuit feed, tailings, and concentrate, down to a size of 45 um. The residual -45 um samples from September to December, 1984 were micro-sieved down to 5 um at McGill University, and then returned to the Selbaie assay laboratory for analysis. The -45 um November tailings sample was found to be contaminated, so this data was discarded. A summary of the size distributions and copper assays is given in Table 2-1. Note that these data were for the period prior to replacement of the flotation cells during the later mill expansion.

Table 2-2 and Figure 2-2 summarize the size by size copper recoveries for the four months. The mean recoveries for each size fraction were taken as the basis for the size versus recovery curve shown.

Table 2-1. Les Mines Selbaie Monthly Composite Flotation Size and Copper Assay Data

A. September, 1984

Size Class	Feed		Tailings		Concentrate	
	Wt. %	% Cu	Wt. %	% Cu	Wt. %	% Cu
+250 um	1.2	0.49	1.2	0.44	0.1	7.4
+150	7.4	0.49	8.6	0.23	1.5	9.8
+125	6.0	0.66	7.0	0.19	1.8	13.5
+ 90	9.1	1.07	10.5	0.16	4.0	17.0
+ 75	5.7	1.95	6.2	0.12	4.1	20.8
+ 45	15.2	3.65	15.2	0.09	20.5	24.4
+ 25	15.7	4.40	15.3	0.06	23.2	27.5
+ 15	14.1	3.66	13.2	0.06	17.7	28.2
+ 10	6.4	3.34	5.6	0.08	7.4	28.4
+ 5	9.9	2.31	9.6	0.10	11.0	28.1
- 5	9.3	2.58	7.6	0.42	8.7	27.9
Total	100.0		100.0		100.0	

B. October, 1984

Size Class	Feed		Tailings		Concentrate	
	Wt. %	% Cu	Wt. %	% Cu	Wt. %	% Cu
+250 um	1.6	0.70	1.7	0.51	0.1	12.0
+150	7.8	0.62	8.6	0.23	1.5	12.0
+125	6.2	0.92	7.3	0.20	1.8	15.8
+ 90	9.2	1.51	10.5	0.16	4.3	20.3
+ 75	5.2	2.83	5.7	0.14	4.6	23.3
+ 45	15.4	4.50	15.4	0.11	21.6	26.6
+ 25	14.3	4.77	15.1	0.07	22.8	27.4
+ 15	14.1	3.77	13.1	0.07	17.5	27.6
+ 10	6.0	3.46	5.6	0.07	6.9	27.8
+ 5	11.6	2.45	8.9	0.14	10.6	27.4
- 5	8.6	2.81	8.2	0.36	8.3	27.4
Total	100.0		100.0		100.0	

C. November, 1984

Size Class	Feed Wt. %	Feed % Cu	Tailings Wt. %	Tailings % Cu	Concentrate Wt. %	Concentrate % Cu
+250 um	2.3	0.67	2.2	0.50	0.2	14.3
+150	8.9	0.83	11.3	0.29	2.2	15.8
+125	6.3	1.36	5.9	0.22	2.5	21.2
+ 90	8.8	2.22	11.1	0.18	5.8	24.0
+ 75	5.3	3.41	5.8	0.14	5.7	27.0
+ 45	14.1	4.63	13.7	0.12	21.4	29.2
- 45	54.3	-	50.0	-	62.2	-
Total	100.0		100.0		100.0	

D. December, 1984

Size Class	Feed Wt. %	Feed % Cu	Tailings Wt. %	Tailings % Cu	Concentrate Wt. %	Concentrate % Cu
+250 um	1.2	0.56	1.3	0.52	0.1	13.6
+150	7.0	0.56	7.6	0.22	1.5	13.6
+125	5.6	0.96	6.4	0.17	1.9	16.8
+ 90	8.4	1.66	9.7	0.15	4.7	19.9
+ 75	5.3	2.58	6.0	0.13	5.1	22.7
+ 45	14.9	3.84	16.6	0.10	16.9	24.8
+ 25	14.0	3.71	15.1	0.07	27.6	25.8
+ 15	15.5	2.94	13.2	0.06	15.9	26.3
+ 10	7.3	2.74	6.2	0.07	7.4	26.8
+ 5	12.2	1.95	10.0	0.13	10.1	27.2
- 5	8.7	2.49	7.9	0.40	8.8	28.3
Total	100.0		100.0		100.0	

Table 2-2. Summary of Size by Size Copper Recoveries at Selbaie, Sept. - Dec. 1984 (Percent)

Size Range (μm)	Sept.	Oct.	Nov.	Dec.	Mean
+250	10.9	28 (approx)	27.8	7.4	18.5
-250 +150	54.3	64.1	66.3	61.7	61.6
-150 +125	72.2	79.3	84.7	83.1	79.8
-125 + 90	85.9	90.1	92.6	91.7	90.1
- 90 + 75	94.4	95.6	96.4	95.5	95.5
- 75 + 45	97.9	98.0	97.8	97.8	97.9
- 45 + 25	98.9	98.8	-	98.4	98.7
- 25 + 15	98.6	98.4	-	98.2	98.4
- 15 + 10	97.9	98.2	-	95.5	97.2
- 10 + 5	96.0	94.8	-	93.8	94.9
- 5	85.0	88.3	-	85.1	86.1

Each of the average copper recovery values was plotted against the arithmetic mean size of the sizing screen openings. The recovery values may be applied to any given copper distribution to obtain an estimate of the overall copper recovery for that particular feed. Note that although recovery values varied widely in the coarser size fractions, they were very consistent below about 100 um where most of the copper value is present. This curve follows the typical shape for sulphide ores, with losses expected in the coarse and fine ends due to flotation difficulties associated with very coarse and very fine particles.

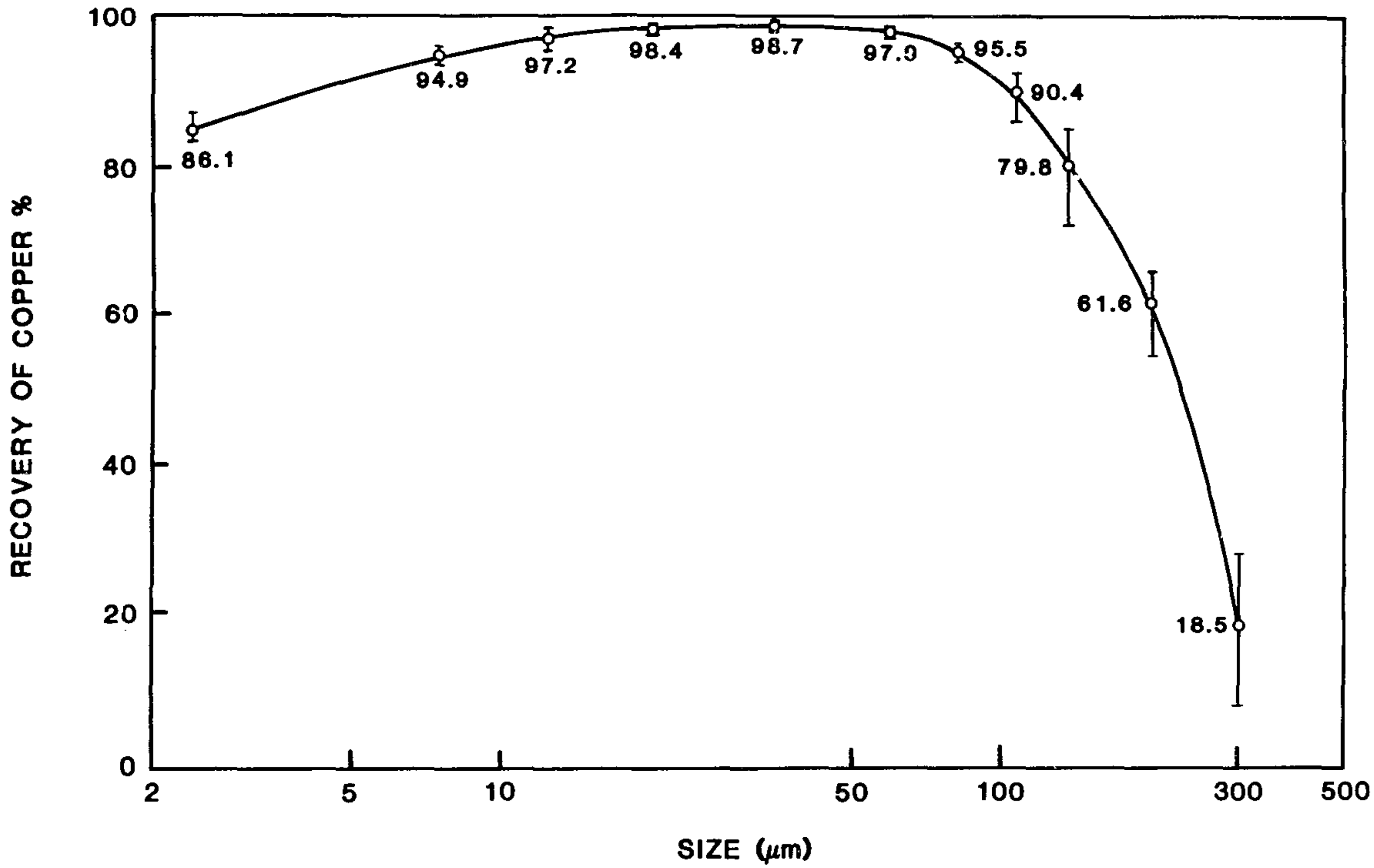


Figure 2-2. Les Mines Selbaie Copper Size-Recovery Data for Sept.-Dec. 1984

2.3.3 General Circuit Sampling (Surveys No. 1 and 2)

2.3.3.1 Survey Data

<u>Survey No. 1</u>

This was a complete grinding circuit sampling survey carried out on November 12, 1985, while the crushing plant was operating and bringing an influx of fines into the grinding circuit through the surge tank.

The samples were first weighed, dried, and then re-weighed to determine percent solids, as summarized in Table 2-3. Individual rod mill overflow samples were then riffled and sized to check for circuit stability throughout the sampling period (Table 2-4A). Based on the close agreement obtained, the other samples were combined, riffled, and then screen analysed, (Table 2-4B).

Specific gravity tests were carried out on the cyclone feed and cyclone underflow samples, giving values of 2.91 and 2.94 respectively.

Bond work index testing was carried out on the rod mill feed and discharge samples collected for this purpose, as well as comparative (batch) ball mill grindability tests at the mine site, as described later. The rod mill feed size distribution from the grindability test is shown in Table 2-4C.

Table 2-3. <u>Selbaie Survey No. 1 - Wet and Dry Sample Weights</u>

Sample	Wet Weight (g)	Dry Weight (g)	% Solids	Ave. % Solids
COF 1-1	5025.1	2147.8	42.74	42.18
COF 1-2	3450.2	1427.1	41.36	
RMD 1-1	4377.5	3537.2	80.80	80.86
RMD 1-2	4293.4	3473.9	80.91	
BMD 1-1	5283.4	3926.4	74.32	75.07
BMD 1-2	8011.8	6053.9	75.56	
CUF 1-1	8214.4	6359.3	77.42	77.28
CUF 1-2	7682.2	5924.9	77.13	
CF 1-1	3309.5	2307.3	69.72	70.65
CF 1-2	2385.7	1716.2	71.94	
RMF 1-1	14849.2	14005.6	94.32	94.15
RMF 1-2	16997.1	15978.6	94.01	
STD 1-1	4046.6	842.1	20.81	18.25
STD 1-2	4159.9	655.4	15.76	

Table 2-4A. Selbaie Survey No. 1 - Size Distributions

Size (μm)	Cyclone Overflow 1-1 Weight (g)	Weight (%)	Cum.% Pass	Cyclone Overflow 1-2 Weight (g)	Weight (%)	Cum.% Pass
+212	5.55	4.12	95.88	2.86	3.13	96.87
+150	12.10	8.98	86.90	7.59	8.31	88.56
+106	13.77	10.21	76.69	9.13	10.00	78.56
+ 75	12.01	8.91	67.78	8.05	8.81	69.75
+ 53	10.45	7.75	60.03	7.19	7.87	61.88
+ 38	11.49	8.52	51.51	8.06	8.83	53.05
- 38	69.44	51.51		48.45	53.05	
Total	134.81			91.33		
Start Wt.	136.4			93.1		

Screen Size (μm)	Rod Mill Discharge 1-1 Weight (g)	Weight (%)	Cum.% Pass	Rod Mill Discharge 1-2 Weight (g)	Weight (%)	Cum.% Pass
+3360	1.23	0.61	99.39	0.74	0.34	99.66
+2360	5.18	2.59	96.80	5.29	2.45	97.21
+1700	12.95	6.47	90.33	13.47	6.24	90.97
+1180	19.46	9.73	80.60	20.99	9.72	81.25
+ 850	22.54	11.27	69.33	23.75	11.00	70.25
+ 600	20.50	10.25	59.08	22.12	10.25	60.00
+ 425	20.54	10.27	48.81	20.95	9.71	50.29
+ 300	12.21	6.10	42.71	13.77	6.38	43.91
+ 212	10.65	5.32	37.39	10.10	4.68	39.23
+ 150	9.91	4.95	32.44	10.90	5.05	34.18
+ 106	7.90	3.95	28.49	8.64	4.00	30.18
+ 75	6.00	3.00	25.49	6.67	3.09	27.09
+ 53	4.69	2.34	23.15	5.61	2.60	24.49
+ 38	4.49	2.24	20.91	5.78	2.68	21.81
- 38	41.80	20.91		47.06	21.81	
Total	200.05			215.84		
Start Wt.	203.9			219.1		

Table 2-4B. Selbaie Survey No. 1 Size Distributions

Screen Size (µm)	Cyclone Underflow Weight (g)	Cyclone Underflow Weight (%)	Cyclone Underflow Cum. Pass. (%)	Ball Mill Discharge Weight (g)	Ball Mill Discharge Weight (%)	Ball Mill Discharge Cum. Pass. (%)	Cyclone Feed Weight (g)	Cyclone Feed Weight (%)	Cyclone Feed Cum. Pass. (%)	Crusher Fines Weight (g)	Crusher Fines Weight (%)	Crusher Fines Cum. Pass. (%)
+4750	0		100	0		100	0		100			
+3360	1.93	1.02	98.98	2.03	1.26	98.74	1.77	0.70	99.30			
+2360	4.91	2.60	96.38	2.35	1.46	97.28	5.60	2.20	97.10			
+1700	9.63	5.09	91.29	3.78	2.35	94.93	12.52	4.93	92.17			
+1180	12.76	6.75	84.54	5.64	3.50	91.43	19.11	7.52	84.65			
+ 850	13.91	7.35	77.19	6.23	3.87	87.56	19.51	7.68	76.97			
+ 600	15.62	8.26	68.93	8.64	5.37	82.19	19.63	7.73	69.24			
+ 425	18.45	9.75	59.18	11.98	7.44	74.75	20.57	8.10	61.14			
+ 300	21.66	11.45	47.73	16.20	10.07	64.68	20.16	7.93	53.21			
+ 212	19.76	10.45	37.28	16.53	10.27	54.41	17.81	7.01	46.20	6.95	8.39	91.61
+ 150	18.39	9.72	27.56	18.15	11.28	43.13	19.04	7.49	38.71	6.53	7.88	83.73
+ 106	13.81	7.30	20.26	15.16	9.42	33.71	16.08	6.33	32.38	7.77	9.38	74.35
+ 75	9.28	4.91	15.35	11.02	6.85	26.86	12.27	4.83	27.55	7.20	8.69	65.66
+ 53	5.34	2.82	12.53	6.94	4.31	22.55	8.73	3.44	24.11	6.01	7.25	58.41
+ 38	3.98	2.10	10.43	5.84	3.63	18.92	8.98	3.53	20.58	6.38	7.70	50.71
- 38	19.73	10.43		30.46	18.93		52.32	20.59		42.01	50.71	
Total	189.16	100.00		160.95	100.00		254.10	100.00		82.85	100.00	
Start Wt	190.3			163.4			257.1			84.6		

Table 2-4C. Selbaie Survey No. 1, Rod Mill Feed Size Distribution

Size (μm)	Weight (%)	Cum. % Passing
26,500	0	100.0
19,000	0.55	99.45
13,200	11.35	88.10
9,500	21.05	67.05
6,700	14.16	52.89
4,750	9.72	43.17
3,350	7.73	35.44
2,360	6.32	29.13
1,700	4.10	25.02
1,180	4.38	20.64
850	2.66	17.98
600	2.56	15.42
425	2.37	13.05
- 425	13.05	
	100.0	

Survey No. 2

This was a complete grinding circuit sampling survey carried out on November 13, 1985, while the crushing plant was not operating, and only water entered the circuit through the surge tank.

The samples were first weighed, dried and then re-weighed to determine percent solids, as summarized in Table 2-5. Individual rod mill discharge and cyclone overflow samples were then riffled and sized to check for circuit stability throughout the sampling period (Table 2-6A). Based on the close agreement obtained, the other samples were combined, riffled, and then screen analysed (Table 2-6B).

Specific gravity tests were carried out on the cyclone feed and cyclone underflow samples, giving values of 2.90 and 2.93 respectively.

Bond laboratory rod and ball mill grindability tests on rod mill feed and rod mill discharge samples, as well as comparative batch grindability tests at the mine site, are described later. The rod mill feed size distribution is shown in Table 2-6C.

The minus 38 um portion of all samples except the rod mill feed and cyclone feed were riffled, where necessary, and then micro-sieved. The results are given in Table 2-7.

After sizing down to 5 µm, and pulverizing samples of particles larger than 53 µm (270 mesh), the samples were numbered, weighed (after splitting), and submitted to the Centre de Recherches Minerales in Quebec City for Cu, Zn, Pb and Fe assaying by atomic adsorption. Sample weights were determined only for the purpose of estimating sampling error, if needed, and do not correspond to size distribution characteristics. The results are given in Tables 2-8 and 2-9.

2.3.3.2 Mass Balance and Data Adjustment

The mass flow rates around the grinding circuit are part of the basic information required for characterization of circuit performance. The flows which cannot be obtained by direct measurement must be calculated by the mass balance equations which apply to the circuit. Due to various sources of error, each size class in the size distribution data from the sampling runs provides a different estimate of the mass split at the hydrocyclone (i.e., the circulating load). The

Table 2-5. Selbaie Survey No. 2 - Wet and Dry Sample Weights

Sample	Wet Weight (g)	Dry Weight (g)	% Solids	Ave. % Solids
COF 2-1	4553.0	1920.4	42.18	42.03
COF 2-2	3583.0	1499.3	41.84	
RMD 2-1	3464.3	2820.1	81.40	81.31
RMD 2-2	5381.2	4370.4	81.22	
BMD 2-1	5258.2	3909.5	74.35	74.30
BMD 2-2	5666.5	4207.5	74.25	
CUF 2-1	7492.7	5766.1	76.96	76.91
CUF 2-2	6957.4	5347.0	76.85	
CF 2-1	1695.1	1066.3	62.90	66.58
CF 2-2	2221.3	1541.4	69.39	
RMF 2-1	12051.9	11361.5	94.27	94.32
RMF 2-2	13815.0	13035.4	94.36	

Table 2-6A. Selbaie Survey No. 2 - Size Distributions

Size (μm)	Cyclone Overflow 2-1 Weight (g)	Weight (%)	Cum.% Pass	Cyclone Overflow 2-2 Weight (g)	Weight (%)	Cum.% Pass
+212	3.68	3.28	96.72	2.94	3.12	96.88
+150	9.73	8.68	88.04	8.21	8.72	88.16
+106	11.73	10.47	77.57	9.83	10.44	77.72
+ 75	10.70	9.55	68.02	8.91	9.46	68.26
+ 53	9.04	8.07	59.95	7.84	8.33	59.93
+ 38	10.46	9.34	50.61	8.80	9.35	50.58
- 38	56.71	50.61		47.63	50.58	
Total	112.05			94.16		
Start Wt.	113.6			96.5		

Screen Size (μm)	Rod Mill Discharge 2-1 Weight (g)	Weight (%)	Cum.% Pass	Rod Mill Discharge 2-2 Weight (g)	Weight (%)	Cum.% Pass
+3360	.58	.36	99.64	1.29	.62	99.38
+2360	4.37	2.70	96.94	5.34	2.57	96.81
+1700	10.54	6.52	90.42	13.40	6.45	90.36
+1180	15.56	9.63	80.79	20.65	9.93	80.43
+ 850	17.81	11.02	69.77	21.88	10.53	69.90
+ 600	16.28	10.07	59.70	20.71	9.96	59.94
+ 425	15.15	9.38	50.32	18.65	8.97	50.97
+ 300	10.01	6.19	44.13	13.09	6.30	44.67
+ 212	7.46	4.62	39.51	9.71	4.67	40.00
+ 150	8.25	5.11	34.46	10.55	5.08	34.92
+ 106	7.05	4.36	30.04	9.05	4.35	30.57
+ 75	5.80	3.59	26.45	7.51	3.61	26.96
+ 53	4.91	3.04	23.41	6.29	3.03	23.93
+ 38	4.94	3.06	20.35	6.45	3.10	20.83
- 38	32.89	20.35		43.29	20.83	
Total	161.60			207.86		
Start Wt.	163.7			211.2		

Table 2-6B. Selbaie Survey No. 2 Size Distributions

Screen Size (µm)	Cyclone Underflow Weight (g)	Cyclone Underflow Weight (%)	Cyclone Underflow Cum. Pass. (%)	Ball Mill Discharge Weight (g)	Ball Mill Discharge Weight (%)	Ball Mill Discharge Cum. Pass. (%)	Cyclone Feed Weight (g)	Cyclone Feed Weight (%)	Cyclone Feed Cum. Pass. (%)
+4750	0		100	0		100	0		100
+3360	1.72	1.03	98.97	1.67	0.66	99.34	0.67	0.44	99.56
+2360	4.00	2.40	96.57	3.70	1.45	97.89	2.54	1.66	97.90
+1700	7.75	4.65	91.92	5.97	2.34	95.55	6.30	4.13	93.77
+1180	10.27	6.16	85.76	8.17	3.21	92.34	9.58	6.28	87.49
+ 850	11.50	6.96	78.86	9.24	3.63	88.71	10.32	6.76	80.73
+ 600	13.27	7.96	70.90	12.58	4.94	83.77	10.60	6.75	73.78
+ 425	15.71	9.42	61.48	17.85	7.01	76.76	11.17	7.32	66.46
+ 300	18.55	11.13	50.35	24.10	9.46	67.30	11.39	7.46	59.00
+ 212	17.18	10.30	40.05	24.83	9.75	57.55	10.83	7.10	51.90
+ 150	17.26	10.35	29.70	29.66	11.65	45.96	12.32	8.07	43.83
+ 106	13.20	7.92	21.78	24.85	9.76	36.14	10.81	7.08	36.75
+ 75	9.38	5.61	16.17	19.26	7.56	28.58	8.71	5.71	31.04
+ 53	5.72	3.43	12.74	12.54	4.92	23.66	6.46	4.23	26.81
+ 38	4.33	2.60	10.14	11.56	4.54	19.12	6.36	4.17	22.64
- 38	16.92	10.15		48.72	19.13		34.53	22.63	
Total	166.73	100.00		254.70	100.00		152.59	100.00	
Start Wt	168.4			257.2			154.6		

Table 2-6C. Selbaie Survey No. 2, Rod Mill Feed Size Distribution

Size (µm)	Weight (%)	Cum. % Passing
26,500	0	100.0
19,000	0.38	99.62
13,200	13.03	86.59
9,500	22.22	64.37
6,700	15.26	49.11
4,750	10.10	39.01
3,350	7.00	32.01
2,360	5.48	26.53
1,700	3.72	22.81
1,180	3.64	19.17
850	2.17	17.00
600	2.08	14.92
425	1.90	13.02
300	1.36	11.66
212	1.14	10.52
150	1.08	9.44
106	1.06	8.38
75	1.06	7.32
53	1.07	6.25
38	0.93	5.32
- 38	5.32	
	100.0	

Table 2-7. Micro-sieving of Samples from Selbaie Survey No. 2

Sample: Rod mill discharge 2-1

Starting weight	24.44 g
Sized weights	
-38 +25 µm	5.97 g
-25 +15 µm	4.63
-15 +10 µm	4.34
-10 + 5 µm	3.47
- 5 µm	4.75
Total	23.16 g

Sample: Ball Mill discharge 2

Starting weight	24.24 g
Sized weights	
-38 +25 µm	7.05 g
-25 +15 µm	5.29
-15 +10 µm	3.90
-10 + 5 µm	3.08
- 5 µm	4.20
Total	23.52 g

Sample: Cyclone underflow 2

Starting weight	16.73 g
Sized weights	
-38 +25 µm	5.28 g
-25 +15 µm	3.10
-15 +10 µm	2.65
-10 + 5 µm	1.98
- 5 µm	2.97
Total	15.98 g

Sample: Cyclone overflow 2

Starting weight	22.76 g
Sized weights	
-38 +25 µm	7.04 g
-25 +15 µm	4.08
-15 +10 µm	3.73
-10 + 5 µm	2.22
- 5 µm	4.83
Total	21.90 g

Table 2-8. Sample Weights Submitted for Assaying, Selbaie Survey No. 2

Size Class (µm)	RMD 2-1 No.	RMD 2-1 Wt. (g)	BMD 2 No.	BMD 2 Wt. (g)	COF 2 No.	COF 2 Wt. (g)	CUF 2 No.	CUF 2 Wt. (g)
+3360	1.	4.6	19.	4.9	-	-	48.	5.2
+2360								
+1700	2.	9.7	20.	5.7	-	-	49.	7.4
+1180	3.	12.1	21.	7.8	-	-	50.	9.7
+ 850	4.	10.5	22.	8.8	-	-	51.	11.1
+ 600	5.	9.2	23.	11.9	-	-	52.	12.9
+ 425	6.	9.2	24.	17.4	-	-	53.	15.1
+ 300	7.	9.2	25.	23.7	-	-	54.	18.0
+ 212	8.	5.4	26.	24.4	37.	3.4	55.	16.3
+ 150	9.	7.7	27.	10.0	38.	9.3	56.	17.1
+ 106	10.	6.5	28.	13.4	39.	11.1	57.	12.1
+ 75	11.	5.0	29.	18.9	40.	10.3	58.	9.0
+ 53	12.	4.3	30.	12.3	41.	8.9	59.	5.6
+ 38	13.	4.5	31.	11.2	42.	10.1	60.	4.2
+ 25	14.	5.7	32.	7.0	43.	7.0	61.	5.2
+ 15	15.	4.3	33.	5.2	44.	4.0	62.	3.0
+ 10	16.	4.2	34.	3.8	45.	3.6	63.	2.5
+ 5	17.	3.4	35.	3.0	46.	2.2	64.	1.9
- 5	18.	4.8	36.	4.2	47.	5.0	65.	2.9

Table 2-9. Assay Results, Selbaie Survey No. 2

Sample No.	% Cu	% Zn	% Pb	% Fe
1	1.46	0.60	0.05	5.54
2	2.36	0.97	0.04	6.06
3	2.24	1.07	0.05	5.86
4	2.22	1.03	0.05	5.72
5	2.09	0.85	0.04	5.40
6	2.91	1.18	0.05	6.20
7	3.59	1.51	0.09	6.74
8	4.30	1.76	0.10	7.64
9	4.64	1.87	0.10	7.60
10	4.98	1.89	0.11	7.48
11	4.96	1.89	0.11	7.28
12	5.02	1.87	0.13	7.36
13	6.03	2.22	0.13	7.36
14	4.68	1.81	0.11	6.40
15	3.55	1.44	0.09	5.68
16	2.94	1.22	0.10	7.44
17	2.28	0.93	0.10	9.10
18	2.34	0.94	0.15	10.4
19	0.82	0.12	0.02	4.04
20	0.94	0.25	0.01	4.72
21	1.39	0.42	0.03	5.34
22	1.41	0.70	0.03	5.36
23	1.69	0.70	0.06	5.56
24	2.17	0.90	0.05	6.16
25	2.40	1.08	0.05	6.06
26	3.32	1.42	0.07	6.74
27	4.89	2.10	0.09	8.56
28	8.92	3.63	0.17	12.7
29	10.5	3.85	0.23	14.4
30	9.12	3.37	0.33	14.2
31	8.08	2.92	0.38	11.4
32	5.68	2.20	0.26	8.36
33	4.24	1.65	0.15	6.30
34	3.72	1.51	0.13	7.44
35	2.91	1.23	0.13	9.52
36	3.08	1.29	0.19	11.6
37	0.32	0.13	0.02	6.46
38	0.50	0.22	0.02	6.96
39	1.22	0.56	0.02	8.26
40	3.13	1.36	0.04	5.54
41	5.13	1.99	0.06	7.48
42	5.88	2.24	0.10	8.04
43	4.35	1.72	0.08	6.60
44	3.83	1.55	0.10	6.06
45	3.34	1.31	0.09	6.94

Table 2-9

(Cont'd)

Sample No.	% Cu	% Zn	% Pb	% Fe
46	2.72	1.12	0.10	9.24
47	2.65	1.08	0.17	10.9
48	0.81	0.28	0.02	4.48
49	1.16	0.55	0.02	4.48
50	1.76	0.74	0.04	5.52
51	1.77	0.81	0.03	5.66
52	1.98	0.96	0.04	5.54
53	2.28	0.96	0.06	5.84
54	2.66	1.14	0.06	6.28
55	3.53	1.53	0.07	7.28
56	6.42	2.80	0.10	10.5
57	11.1	4.45	0.19	13.9
58	12.9	4.65	0.27	17.1
59	11.3	3.80	0.41	16.7
60	9.52	3.30	0.61	13.7
61	6.50	2.48	0.41	9.48
62	4.50	1.75	0.29	6.66
63	3.67	1.47	0.20	7.40
64	3.02	1.27	0.16	9.40
65	2.83	1.19	0.19	11.2

best estimate of the circulating load is the value which provides the minimum total adjustment of raw data.

The mass balancing process fulfills three purposes. First, it uses all the raw data to provide the most reliable estimate of mass flowrates which cannot be measured directly (in particular, the circulating load). Second, it provides suitable adjustments to the data so that it is all consistent, based on the criterion of minimum total adjustment of the raw data. Thirdly, it provides a quantitative measure of the reliability of the sampling and analysis work, and helps to identify poor or erroneous values.

The computer program used for mass balancing of the size distribution data is based on the method described by Smith and Ichiyen, 1973. The size by size assay data on streams inside the grinding circuit were not incorporated into the balance since they were not needed. The raw and adjusted data from the computer printouts for runs no. 1 and 2 are shown in Tables 2-10 and 2-11.

Note that due to poor accessability, the cyclone feed samples could not be cut across the entire stream. The mass balance calculations revealed that this sample was in fact poor (see Table 2-12). However, since cyclone feed could be reliably calculated from the other size distribution data, it was possible to omit this location from the sampling routine.

2.4 KIDD CREEK MINES CIRCUIT DESIGN AND OPERATION

2.4.1 Process Flowsheet

The process flowsheet for B circuit at Kidd Creek Mines Limited is shown in attached drawing number 11111-01, Figure 2-3. Rod mill feed is prepared by closed circuit crushing to all passing either a 19 mm rod-deck or 40 by 23 mm slotted rubber deck screens. The normal feed rate is 135 dry metric tons per hour. This is a conventional rod and ball milling circuit preceding copper rougher flotation, followed by secondary and copper regrind ball milling, closed-circuited with secondary and cleaner copper flotation, with final copper tails being directed to the zinc flotation circuit. There is

Table 2-10. Selbaie Survey No. 1

Raw and Adjusted Size Data

Circulating Load (%) = 288

Product Identification:

1 Rod Mill Discharge Plus Surge Tank Discharge
2 Ball mill Discharge
3 Cyclone Underflow
4 Cyclone Overflow

Unadjusted Size Data

Mesh	Micrometers	1	2	3	4
6	3360	.38	1.26	1.02	0
8	2360	2.01	1.46	2.6	0
10	1700	5.08	2.35	5.09	0
14	1180	7.76	3.5	6.75	0
20	840	8.9	3.87	7.35	0
28	600	8.19	5.37	8.26	0
35	420	7.98	7.44	9.75	0
48	300	4.98	10.07	11.45	0
65	210	5.68	10.27	10.45	3.63
100	150	5.58	11.28	9.72	8.64
150	105	5.06	9.42	7.3	10.11
200	74	4.19	6.85	4.91	8.86
270	53	3.43	4.31	2.82	7.81
400	37	3.52	3.63	2.1	8.67
-400	-37	27.26	18.92	10.43	52.28

Adjusted Size Data

Mesh	Micrometers	1	2	3	4
6	3360	.35	1.06	1.18	.01
8	2360	2.08	1.59	2.32	-.01
10	1700	5.32	2.56	4.41	-.01
14	1180	7.93	3.63	6.38	0
20	840	9.03	3.96	7.09	0
28	600	8.2	5.38	8.23	0
35	420	7.91	7.27	10.02	0
48	300	4.96	9.92	11.64	0
65	210	5.65	10.02	10.71	3.64
100	150	5.55	11.01	9.92	8.7
150	105	5.04	9.22	7.43	10.19
200	74	4.16	6.67	5.01	8.96
270	53	3.43	4.33	2.81	7.79
400	37	3.56	3.75	2.04	8.48
PASSING		26.82	19.63	10.81	52.24

Table 2-11. Selbaie Survey No. 2

Raw and Adjusted Size Data

Circulating Load (%) = 333

Product Identification:

1 Rod Mill Discharge
2 Ball mill Discharge
3 Cyclone Underflow
4 Cyclone Overflow

Unadjusted Size Data

Mesh	Micrometers	1	2	3	4
6	3360	.49	.66	1.03	0
8	2360	2.64	1.45	2.4	0
10	1700	6.48	2.34	4.65	0
14	1180	9.78	3.21	6.16	0
20	840	10.78	3.63	6.9	0
28	600	10.01	4.94	7.96	0
35	420	9.18	7.01	9.42	0
48	300	6.24	9.46	11.13	0
65	210	4.64	9.75	10.3	3.2
100	150	5.1	11.65	10.35	8.7
150	105	4.36	9.76	7.92	10.46
200	74	3.6	7.56	5.61	9.5
270	53	3.03	4.92	3.43	8.2
400	37	3.08	4.54	2.6	9.34
-400	-37	20.59	19.12	10.14	50.6

Adjusted Size Data

Mesh	Micrometers	1	2	3	4
6	3360	.51	.75	.9	-.01
8	2360	2.67	1.5	2.3	0
10	1700	6.62	2.42	4.41	0
14	1180	9.79	3.21	6.15	0
20	840	10.79	3.64	6.38	0
28	600	10.01	4.94	7.95	0
35	420	9.12	6.89	9.63	0
48	300	6.23	9.37	11.24	0
65	210	4.64	9.8	10.24	3.2
100	150	5.09	11.53	10.44	8.72
150	105	4.36	9.76	7.32	10.46
200	74	3.03	7.46	5.67	9.55
270	53	3.0	4.95	3.41	8.18
400	37	3.0	4.51	2.61	9.38
PASSING		20.47	19.26	10.23	50.54

Table 2-12. Selbaie - Survey No. 2

Calculated vrs Sample Size Distribution Data for Cyclone Feed

Size Class (µm)	Individual Percent Retained Calc.	Individual Percent Retained Sampled	Individual Percent Retained Diff.	Cumulative Percent Retained Calc.	Cumulative Percent Retained Sampled	Cumulative Percent Retained Diff.
3360	0.71	0.44	0.27	0.71	0.44	0.27
2360	1.79	1.66	0.13	2.50	2.10	0.40
1700	3.43	4.13	-.70	5.93	6.23	- .30
1180	4.73	6.28	-1.55	10.66	12.51	-1.85
850	5.29	6.76	-1.47	15.95	19.27	-3.32
600	6.11	6.95	- .84	22.06	26.22	-4.16
425	7.38	7.32	.06	29.44	33.54	-4.10
300	8.64	7.46	1.18	38.08	41.00	-2.92
212	8.62	7.10	1.52	46.70	48.10	-1.4
150	10.05	8.07	1.98	56.75	56.17	0.58
106	8.51	7.08	1.43	65.26	63.25	0.01
75	6.57	5.71	0.86	71.85	68.96	2.87
53	4.51	4.23	0.28	76.34	73.19	2.87
38	4.18	4.17	0.01	80.52	77.36	3.16
PAN	19.48	22.63	-3.15	100.00	100.00	0

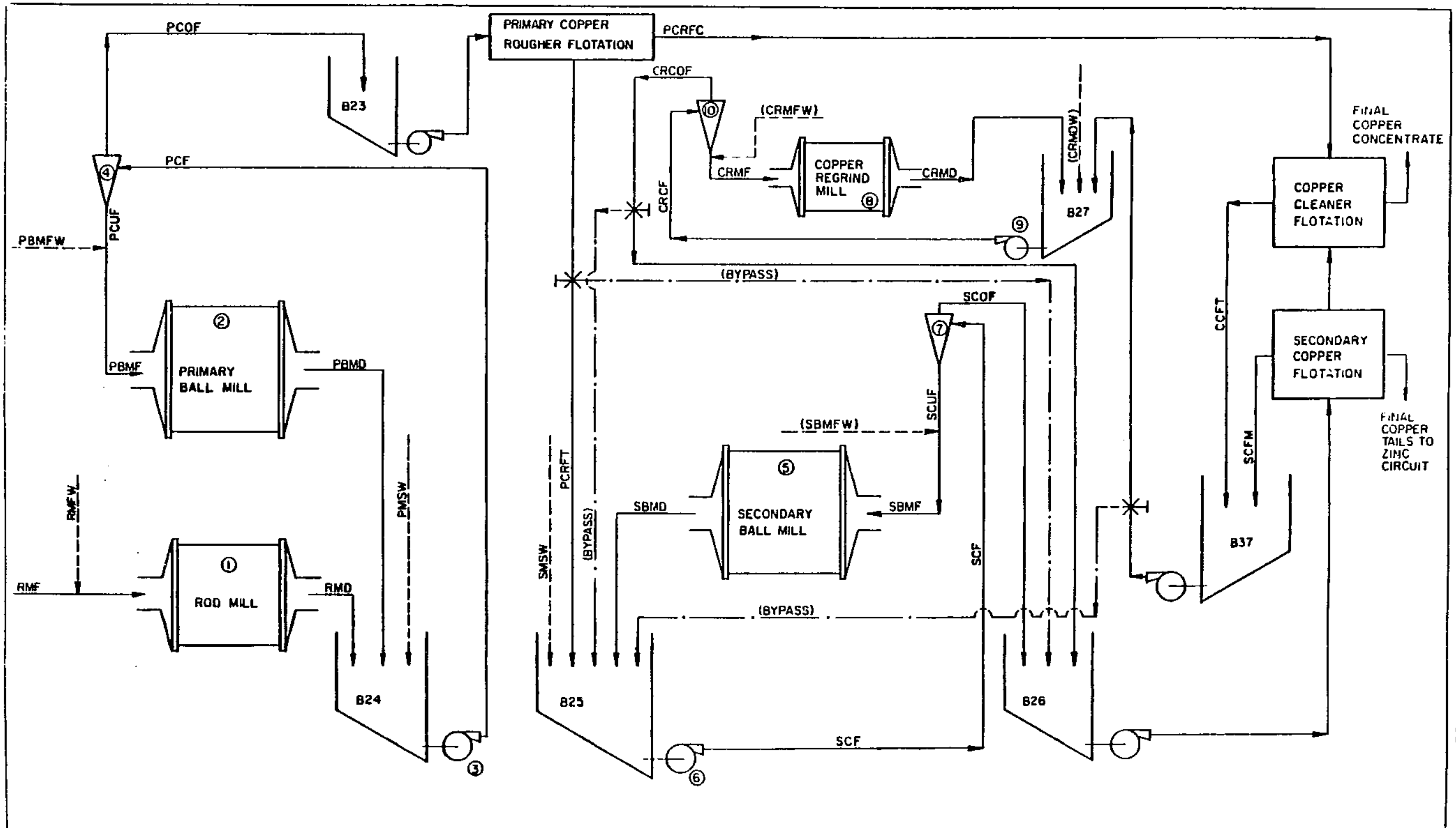


Figure 2-3

DATE	REVISION	BY
.09/85	△ NEW DRAWING	JK

KIDD CREEK MINES LTD. LINE B
GRINDING CIRCUIT - PROCESS FLOWSHEET

DATE: SEPT 16/1985 | SCALE N.T.S.
DWG NO: 1111-01 | REV 0

also a zinc regrind ball mill (not shown). The primary grinding product size is targeted at 50 percent minus 45 µm, and the secondary flotation feed at 78 percent passing 45 µm.

The various process flows on the attached drawing are represented as follows.

RMF:	Rod Mill Feed
RMFW:	Rod Mill Feed Water
RMD:	Rod Mill Discharge
PMSW:	Primary Mill Sump Water
PBMF:	Primary Ball Mill Feed
PBMD:	Primary Ball Mill Discharge
PCF:	Primary Cyclone Feed
PCUF:	Primary Cyclone Underflow
PBMFW:	Primary Ball Mill Feed Water
PCOF:	Primary Cyclone Overflow
PCRFC:	Primary Copper Rougher Flotation Concentrate
PCRFT:	Primary Copper Rougher Flotation Tailings
SMSW:	Secondary Mill Sump Water
SBMF:	Secondary Ball Mill Feed
SBMD:	Secondary Ball Mill Discharge
SCF:	Secondary Cyclone Feed
SCUF:	Secondary Cyclone Underflow
SBMFW:	Secondary Ball Mill Feed Water
SCOF:	Secondary Cyclone Overflow
SCFM:	Secondary Copper Flotation Middlings
CCFT:	Cleaner Copper Flotation Tailings
CRCF:	Copper Regrind Cyclone Feed

CRCOF: Copper Regrind Cyclone Overflow

CRMF: Copper Regrind Mill Feed

CRMFW: Copper Regrind Mill Feed Water

CRMD: Copper Regrind Mill Discharge

CRMDW: Copper Regrind Mill Discharge Water

Note the existence of three bypass options. Primary copper rougher flotation tailings may be directed, in whole or in part, to either the secondary ball mill circuit or secondary copper flotation. The regrind circuit product (cyclone overflow) may be directed, in whole or in part, to either of the same locations. Middlings and tailings from copper secondary and cleaner flotation may be directed to the regrind or secondary ball milling circuit.

2.4.2 Size by Size Behavior of Cu and Zn Minerals in Flotation

The overall split of solids to the copper circuit concentrate and tailings streams, plus copper and zinc assays on the size fractions of these samples taken during general surveys no. 1 and 2, were used to generate the size versus recovery curves shown in Figures 2-4 and 2-5. The recoveries are plotted against the arithmetic mid-point of the size class based on the screen opening or cyclosizer cut size, after adjusting for the mineral specific gravity and cyclosizer operating conditions. The copper mineral was assumed to have a specific gravity of 4.2 (chalcopyrite), and the zinc mineral 4.0 (sphalerite), which was not enough to make any significant difference in the size interval or its mid-point, as shown below.

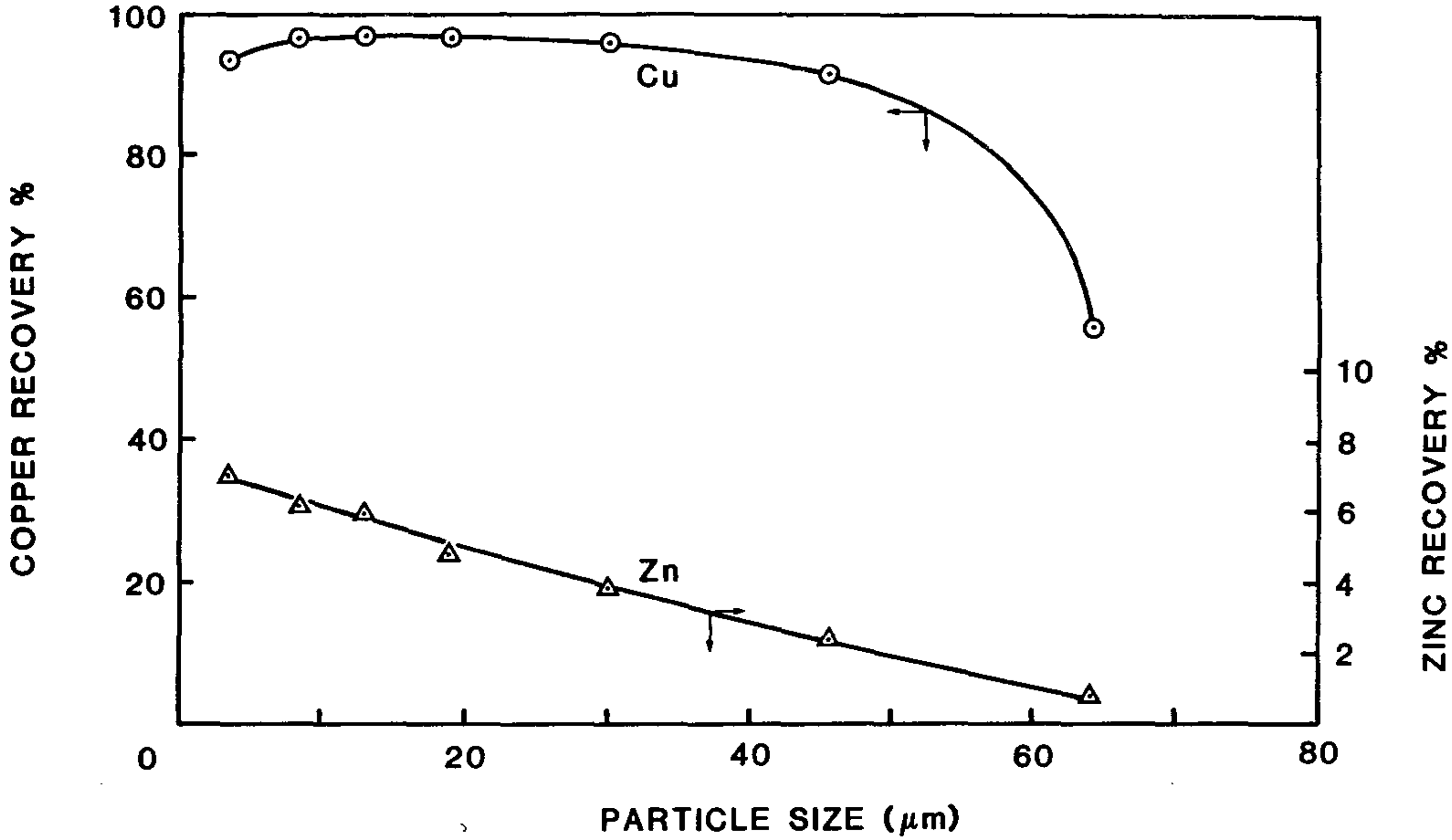


Figure 2-4. Cu and Zn Recovery to Copper Concentrate, Kidd Creek Survey No. 1.

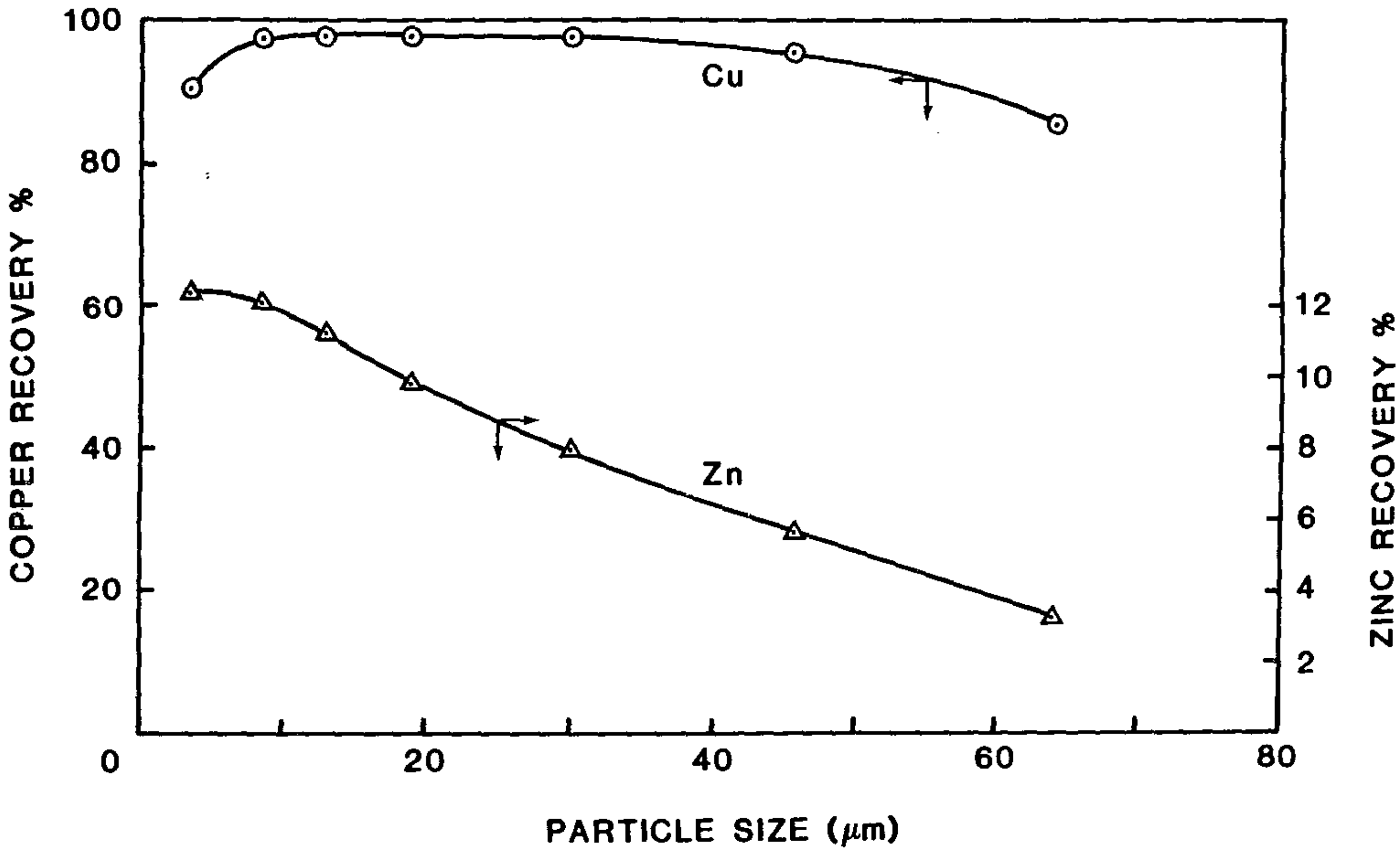


Figure 2-5. Cu and Zn Recovery to Copper Concentrate, Kidd Creek Survey No. 2.

Table 2-13. Screen and Cyclosizer Size Intervals for Cu and Zn Minerals

Size Interval, Copper	Size Interval, Zinc (µm)	Mid-Point (µm)
+53 µm (270 mesh)	+53	64 (est.)
-53 +38 µm (400 mesh)	-53 +38	45.5
-38 +22 µm (cones 1 and 2)	-38 +22	30
-22 +16 µm (cone 3)	-22 +16	19
-16 +10 µm (cone 4)	-16 +10	13
-10 + 7 µm (cone 5)	-10 + 7	8.5
- 7 µm (passing cone 5)	-7	3.5

Note that the solids passing cone 5 were not assayed, but the copper and zinc values were calculated using the overall -400 mesh assay and the sum of the metal content in the other fractions.

The copper recovery curves in Figures 2-4 and 2-5 exhibit the typical shape encountered in sulphide flotation, with losses in the coarse and fine regions. Note that substantial variance is often noted in the calculated recovery of the coarsest particle sizes, probably at least in part due to the small sample size in the coarsest fractions. Zinc recovery to copper concentrate increases significantly with finer particle size.

Sizing and assaying of 1984 monthly composite samples for both the copper and zinc flotation circuits were carried out on site to determine the size vs. recovery characteristics of 'A' ore (Scheding, 1985). The results from the data analysis for the copper circuit are shown in Figure 2-6. The recovery curves are very similar to those determined from the short

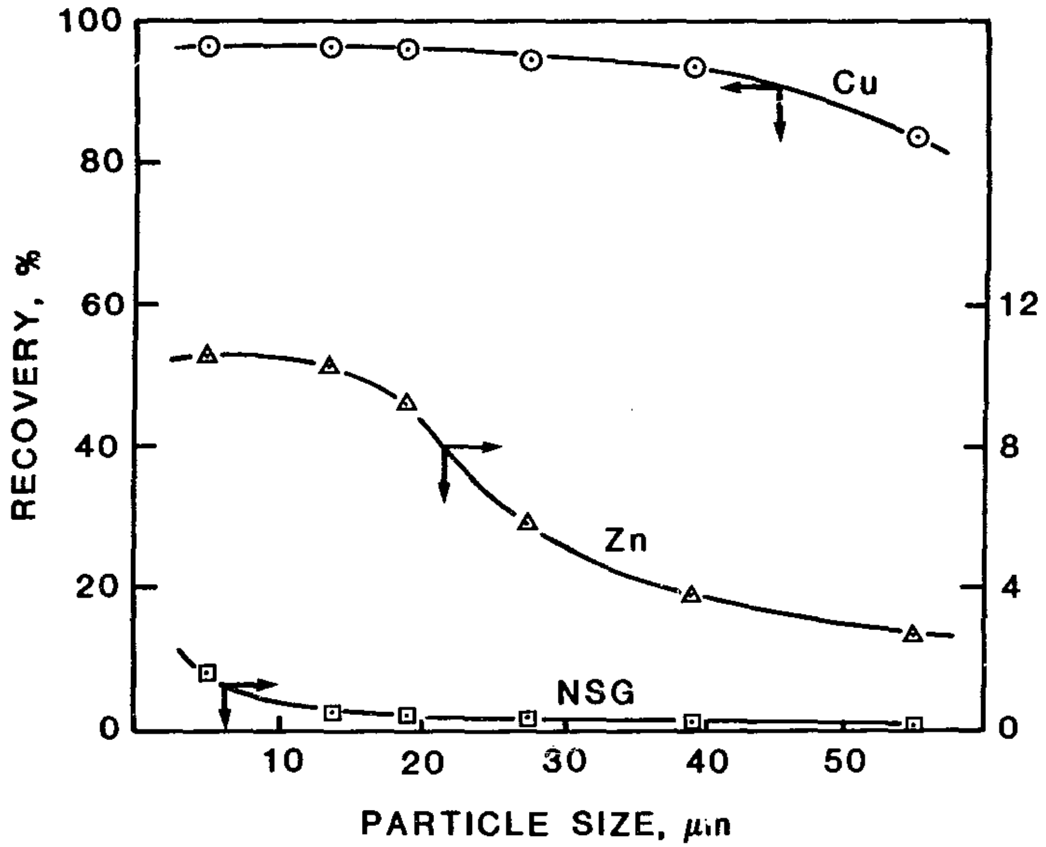


Figure 2-6. Cu, Zn and Non-Sulphide Gangue Recovery to Copper Concentrate, Kidd Creek 1984 Monthly Composite Samples.

duration surveys, although for unknown reasons the drop off in copper recovery in the fine particle range is not observable.

When all the size recovery data from the above plant test are considered, the way in which the minerals of different sizes distribute themselves to the concentrator products, either copper concentrate, zinc concentrate, or final tailings, can be calculated. This is shown for copper and zinc in Table 2-14. Such data could be used to estimate overall plant recoveries from the mineral distribution by size in the grinding circuit product.

Table 2-14. Distribution of Minerals by Size in Concentrator Products, Kidd Creek 1984 Composite Samples (Approximate)

A. Cu Distributions (%)

Size Class (μm)	To Cu. Conc.	To Zn Conc.	To Final Tails	Total
+43	84.1	4.5	11.4	100.0
25-43	93.7	1.6	4.7	100.0
18-25	94.2	2.5	3.3	100.0
11-18	96.0	1.9	2.1	100.0
8-11	96.2	1.6	2.2	100.0
0-8	96.3	1.5	2.2	100.0

B. Zn Distributions (%)

Size Class (μm)	To Cu. Conc.	To Zn Conc.	To Final Tails	Total
+43	2.7	77.1	20.2	100.0
25-43	3.8	83.4	12.8	100.0
18-25	5.9	89.0	5.1	100.0
11-18	9.2	86.3	4.5	100.0
8-11	10.4	85.7	3.9	100.0
0-8	10.6	76.9	12.5	100.0

The absolute values of mineral recoveries can be seen in the general upward or downward shifting of the size recovery curves. Note, for example, the difference in the level of zinc recoveries to copper concentrate in Figures 2-4 and 2-5. This may be due to the nature of the ore, the flotation process chemistry, or a host of other causes. However, the size region of maximum recovery can be seen to remain more or less constant, despite these shifts. This suggests that it is possible to assess the relative effects of a change in grinding circuit product size characteristics despite a variety of other complex interactions in the flotation process.

2.4.3 General Circuit Sampling (Surveys No. 1 and 2)

2.4.3.1 Survey Data

<u>Survey No. 1</u>

This was a complete sampling survey of the grinding and copper flotation circuits carried out on December 18, 1985, with all the product from the copper regrind circuit being recirculated to the secondary ball mill. All the primary copper rougher flotation tailings were also directed to secondary ball milling.

The samples were first weighed, dried and then re-weighed to determine percent solids, and specific gravity checks were performed on a number of subsamples, as summarized in Table 2-15. Samples were sub-sampled by riffling, as necessary, for screen analyses and assays were performed on each size fraction. The size distribution and copper and zinc assay data are summarized in Tables 2-16A through M. Note that small weights on the coarser screen sizes were combined for assaying.

All the minus 38 µm samples were sent to McGill University where they were tested for specific gravity and then cyclosized. They were then returned to the plant for size by size assay determinations. The results are given in Table 2-17.

Bond laboratory rod mill and ball mill work index tests were also carried out, as well as on-site grindability testing, as described later.

Table 2-15. Kidd Creek Survey No. 1 Percent Solids and Specific Gravities

Sample	Dry Weight (g)	% Solids by Weight	Solids Specific Gravity
RMF	18,901	97.75	-
RMD-1	5,298	81.05	3.3
RMD-2	5,960	81.17	-
PBMF	6,845	75.51	3.2
PBMD	14,750	76.42	3.3
PCOF-1	1,615	51.42	3.4
PCOF-2	1,665	51.48	-
PCRFC	-	-	3.8
PCRFT-1	2,044	51.02	3.3
PCRFT-2	2,018	50.75	-
SBMF	4,568	71.53	-
SBMD	8,885	71.60	3.6
SCOF-1	1,957	33.93	3.6
SCOF-2	1,662	33.51	-
CRCOF	1,850	17.43	-
Copper Concentrate	-	-	-
Copper Tailings	-	-	2.9

Table 2-16. Kidd Creek Survey No. 1 Size and Assay Data

Table 2-16A. Survey No. 1 Size and Assay Data: RMF

Mesh	Screen Size	Size Distribution		Assays	
(Tyler)	(μm)	Ind.%	Cum.% Pass.	% Cu	% Zn
1	25,400	3.99	96.01	0.62	7.01
0.742	19,000	5.01	91.00	1.47	6.51
0.530	13,200	15.84	75.16	2.56	4.89
0.371	9,500	15.94	59.22	2.60	6.40
3	6,700	10.46	48.76	2.83	5.70
4	4,750	7.88	40.88	2.49	5.27
6	3,350	6.22	34.66	3.52	6.25
8	2,360	4.46	30.20	3.17	6.81
10	1,700	3.61	26.59	3.28	6.86
14	1,180	3.52	23.07	2.84	5.99
20	850	2.58	20.49	3.02	6.42
28	600	2.01	18.48	3.23	6.18
35	425	1.88	16.60	3.57	6.61
48	300	1.59	15.01	3.73	6.66
65	212	1.40	13.61	4.33	6.92
100	150	1.31	12.30	4.67	7.12
150	106	1.15	11.15	5.00	7.58
200	75	1.47	9.68	5.33	8.08
270	53	0.96	8.72	5.60	8.77
400	38	1.30	7.42	5.77	9.28
-400	- 38	7.42		4.00	6.86
		100.0			

Table 2-16B. Survey No. 1 Size and Assay Data: RMD

Mesh (Tyler)	Screen Size (µm)	Size Distributions: Sample No. 1 Ind.%	Sample No. 1 Cum.% Pass.	Sample No. 2 Ind.%	Sample No. 2 Cum.% Pass.	Combined Assays % Cu	Combined Assays % Zn
1	25,400						
0.742	19,000						
0.530	13,200						
0.371	9,500						
3	6,700	0.00	100.0	0.00	100.0		
4	4,750	0.19	99.81	0.30	99.70	2.24	4.83
6	3,350	1.58	98.23	1.24	98.46		
8	2,360	4.69	93.54	4.31	94.15	2.44	5.23
10	1,700	8.73	84.81	8.16	85.99	2.49	5.61
14	1,180	12.08	72.73	12.45	73.54	2.76	6.01
20	850	10.65	62.08	10.80	62.74	2.83	6.09
28	600	8.11	53.97	8.40	54.34	2.83	5.97
35	425	7.25	46.72	7.40	46.94	2.07	4.85
48	300	5.91	40.81	5.90	41.04	2.96	6.09
65	212	4.90	35.91	4.86	36.18	3.19	6.10
100	150	4.19	31.72	4.15	32.03	3.62	6.39
150	106	3.30	28.42	3.19	28.84	4.06	6.74
200	75	3.26	25.16	3.24	25.60	4.59	7.39
270	53	2.74	22.42	2.70	22.90	5.00	8.06
400	38	3.59	18.83	3.73	19.17	4.94	8.43
-400	- 38	18.83		19.17		3.86	6.95
		100.0		100.0			

Table 2-16C. Survey No. 1 Size and Assay Data: PBMF

Mesh (Tyler)	Screen Size (µm)	Size Distribution Ind.%	Size Distribution Cum.% Pass.	Assays % Cu	Assays % Zn
1	25,400				
0.742	19,000				
0.530	13,200				
0.371	9,500				
3	6,700	0.00	100.0		
4	4,750	0.08	99.92		
6	3,350	0.49	99.43	1.26	3.14
8	2,360	1.60	97.83		
10	1,700	3.14	94.69	1.83	3.75
14	1,180	5.28	89.41	2.10	4.41
20	850	5.77	83.64	2.20	4.74
28	600	6.08	77.56	2.40	4.94
35	425	7.71	69.85	2.11	5.46
48	300	8.85	61.00	2.83	6.41
65	212	10.52	50.48	3.52	7.79
100	150	12.07	38.41	4.51	8.86
150	106	9.40	29.01	4.89	8.46
200	75	7.47	21.54	4.79	8.27
270	53	4.46	17.08	5.10	8.60
400	38	4.27	12.81	5.11	9.00
-400	- 38	12.81		4.09	7.54
		100.0			

Table 2-16D. Survey No. 1 Size and Assay Data: PBMD

Mesh (Tyler)	Screen Size (μm)	Size Distribution Ind.%	Size Distribution Cum.% Pass.	Assays % Cu	Assays % Zn
1	25,400				
0.742	19,000				
0.530	13,200				
0.371	9,500				
3	6,700	0.00	100.0		
4	4,750	0.17	99.83		
6	3,350	0.23	99.60	1.56	3.55
8	2,360	0.70	98.90		
10	1,700	1.22	97.68	1.83	4.04
14	1,180	2.41	95.27	2.11	4.33
20	850	3.12	92.15	2.26	4.78
28	600	3.94	88.21	2.37	5.02
35	425	6.03	82.18	2.92	1.75
48	300	8.22	74.96	2.67	5.67
65	212	10.79	63.17	3.27	6.85
100	150	12.96	50.21	3.95	7.75
150	106	10.54	39.67	4.39	7.74
200	75	8.75	30.92	4.39	7.73
270	53	5.57	25.35	4.71	8.20
400	38	6.10	19.25	4.67	8.46
-400	- 38	19.25		3.90	7.22
		100.0			

Table 2-16E. Survey No. 1 Size and Assay Data: PCOF

Mesh (Tyler)	Screen Size (μm)	Size Distributions Sample No. 1 Ind.%	Sample No. 1 Cum.% Pass.	Sample No. 2 Ind.%	Sample No. 2 Cum.% Pass.	Combined Assays % Cu	Combined Assays % Zn
1	25,400						
0.742	19,000						
0.530	13,200						
0.371	9,500						
3	6,700						
4	4,750						
6	3,350						
8	2,360						
10	1,700			0	100.0		
14	1,180	0	100.0	0.03	99.97		
20	850	0.01	99.99	0.03	99.94	0.44	0.79
28	600	0.06	99.93	0.11	99.83		
35	425	0.77	99.16	0.87	98.96		
48	300	2.81	96.35	2.82	96.14	0.38	0.51
65	212	5.26	91.09	5.00	91.14	0.56	0.87
100	150	7.17	83.92	6.93	84.21	1.21	2.54
150	106	7.82	76.10	7.87	76.34	2.49	5.22
200	75	10.05	66.05	10.02	66.32	3.44	6.60
270	53	7.76	58.29	7.75	58.57	4.07	7.67
400	38	10.88	47.41	11.17	47.40	4.41	8.09
-400	- 38	47.41		47.40		3.69	7.03
		100.0		100.0			

Table 2-16F. Survey No. 1 Size and Assay Data: PCRFC

Mesh (Tyler)	Screen Size (μm)	Size Distribution Ind.%	Size Distribution Cum.% Pass.	Assays % Cu	Assays % Zn
1	25,400				
0.742	19,000				
0.530	13,200				
0.371	9,500				
3	6,700				
4	4,750				
6	3,350				
8	2,360				
10	1,700	0	100.0		
14	1,180	0.01	99.99		
20	850	0.01	99.98		
28	600	0.02	99.96		
35	425	0.09	99.87	23.2	4.81
48	300	0.32	99.55		
65	212	0.52	99.03		
100	150	0.20	98.83		
150	106	0.05	98.78		
200	75	0.27	98.51	26.9	3.10
270	53	1.46	97.05	28.0	3.09
400	38	9.17	87.88	27.9	3.28
-400	- 38	87.88		23.7	4.99
		100.0			

Table 2-16G. Survey No. 1 Size and Assay Data: PCRFT

Mesh (Tyler)	Screen Size (μm)	Size Distributions Sample No. 1 Ind.%	Sample No. 1 Cum.% Pass.	Sample No. 2 Ind.%	Sample No. 2 Cum.% Pass.	Combined Assays % Cu	Combined Assays % Zn
1	25,400						
0.742	19,000						
0.530	13,200						
0.371	9,500						
3	6,700						
4	4,750						
6	3,350						
8	2,360						
10	1,700	0	100.0	0			
14	1,180	0.06	99.94	0			
20	850	0.05	99.89	0.01	99.99		
28	600	0.14	99.75	0.11	99.88	0.55	0.93
35	425	1.04	98.71	1.05	98.83		
48	300	3.41	95.30	3.27	95.56	0.45	0.51
65	212	5.84	89.46	5.61	89.95	0.59	0.92
100	150	7.71	81.75	7.53	82.42	1.18	2.50
150	106	8.21	73.54	8.19	74.23	2.45	5.25
200	75	10.14	63.40	10.06	64.17	3.57	6.98
270	53	8.47	54.93	8.53	55.64	3.94	7.77
400	38	11.20	43.73	11.15	44.49	3.63	8.60
-400	- 38	43.73		44.49		1.34	7.47
		100.0		100.0			

Table 2-16H. Survey No. 1 Size and Assay Data: SBMF

Mesh (Tyler)	Screen Size (μm)	Size Distribution Ind.%	Size Distribution Cum.% Pass.	Assays % Cu	Assays % Zn
1	25,400				
0.742	19,000				
0.530	13,200				
0.371	9,500				
3	6,700				
4	4,750				
6	3,350				
8	2,360	0.00	100.0		
10	1,700	0.01	99.99		
14	1,180	0.01	99.98	0.67	1.05
20	850	0.02	99.96		
28	600	0.08	99.88		
35	425	0.56	99.32		
48	300	1.97	97.35	0.56	0.75
65	212	4.59	92.76	0.60	0.90
100	150	9.16	83.60	1.32	2.65
150	106	11.94	71.66	3.39	6.86
200	75	22.14	49.52	5.88	10.20
270	53	14.66	34.86	6.80	9.60
400	38	17.07	17.79	6.48	7.80
-400	- 38	17.79		4.98	7.84
		100.0			

Table 2-16I. Survey No. 1 Size and Assay Data: SBMD

Mesh (Tyler)	Screen Size (μm)	Size Distribution Ind.%	Size Distribution Cum.% Pass.	Assays % Cu	Assays % Zn
1	25,400				
0.742	19,000				
0.530	13,200				
0.371	9,500				
3	6,700				
4	4,750				
6	3,350				
8	2,360				
10	1,700	0	100.0		
14	1,180	0.02	99.98		
20	850	0.03	99.95		
28	600	0.05	99.90	0.79	1.25
35	425	0.20	99.70		
48	300	0.86	98.84		
65	212	2.50	96.34	0.67	1.08
100	150	5.93	90.41	1.14	2.26
150	106	9.99	80.42	2.78	5.64
200	75	19.46	60.96	5.01	8.86
270	53	16.52	44.44	6.22	9.13
400	38	18.81	25.63	5.94	7.49
-400	- 38	25.69		4.68	7.39
		100.0			

Table 2-16J. Survey No. 1 Size and Assay Data: SCOF

Mesh (Tyler)	Screen Size (μm)	Size Distributions: Sample No. 1 Ind.%	Sample No. 1 Cum.% Pass.	Sample No. 2 Ind.%	Sample No. 2 Cum.% Pass.	Combined Assays % Cu	Combined Assays % Zn
1	25,400						
0.742	19,000						
0.530	13,200						
0.371	9,500						
3	6,700						
4	4,750						
6	3,350						
8	2,360						
10	1,700			0	100.0		
14	1,180			0.02	99.98		
20	850			0.02	99.96		
28	600	0	100.0	0.02	99.94		
35	425	0.04	99.96	0.01	99.93	0.84	1.49
48	300	0.05	99.91	0.05	99.88		
65	212	0.16	99.75	0.16	99.72		
100	150	0.98	98.77	0.91	98.81		
150	106	3.32	95.45	3.04	95.77	0.49	0.84
200	75	5.93	89.52	6.07	89.70	0.77	1.36
270	53	6.67	82.85	6.59	83.11	2.27	4.04
400	38	13.10	69.75	13.73	69.38	5.30	7.41
-400	- 38	69.75		69.38		4.56	7.78
		100.0		100.0			

Table 2-16K. Survey No. 1 Size and Assay Data: CRCOF

Mesh (Tyler)	Screen Size (μm)	Size Distribution Ind.%	Size Distribution Cum.% Pass.	Assays % Cu	Assays % Zn
1	25,400				
0.742	19,000				
0.530	13,200				
0.371	9,500				
3	6,700				
4	4,750				
6	3,350				
8	2,360				
10	1,700	0.00	100.0		
14	1,180				
20	850	0.01	99.99		
28	600				
35	425	0.02	99.97		
48	300	0.11	99.86	2.12	2.25
65	212	0.31	99.55		
100	150	0.59	98.96		
150	106	0.92	98.04	5.14	5.41
200	75	1.97	96.07	9.04	7.23
270	53	3.72	92.35	14.3	6.42
400	38	11.78	80.57	17.8	6.56
-400	- 38	80.57		11.5	9.81
		100.0			

Table 2-16L. Survey No. 1 Size and Assay Data: Copper Concentrate

Mesh (Tyler)	Screen Size (µm)	Size Distribution Ind.%	Cum.% Pass.	Assays % Cu	% Zn
1	25,400				
0.742	19,000				
0.530	13,200				
0.371	9,500				
3	6,700				
4	4,750				
6	3,350				
8	2,360				
10	1,700				
14	1,180	0	100.0		
20	850	0.01	99.99		
28	600	0.01	99.98		
35	425	0.01	99.97		
48	300	0.01	99.96	25.9	2.56
65	212	0.01	99.95		
100	150	0.02	99.93		
150	106	0.04	99.89		
200	75	0.23	99.66		
270	53	2.02	97.64	30.9	1.52
400	38	13.16	94.48	30.2	1.76
-400	- 38	84.48		27.7	2.88
		100.0			

Table 2-16M. Survey No. 1 Size and Assay Data: Copper Tailings

Mesh (Tyler)	Screen Size (µm)	Size Distribution Ind.%	Cum.% Pass.	Assays % Cu	% Zn
1	25,400				
0.742	19,000				
0.530	13,200				
0.371	9,500				
3	6,700				
4	4,750				
6	3,350				
8	2,360				
10	1,700	0	100.0		
14	1,180	0.02	99.98		
20	850	0.02	99.96		
28	600	0.03	99.93		
35	425	0.04	99.89	0.28	1.07
48	300	0.06	99.83		
65	212	0.19	99.64		
100	150	1.30	98.34		
150	106	4.14	94.20	0.30	0.82
200	75	8.04	86.16	0.23	1.44
270	53	8.29	77.87	0.25	4.34
400	38	14.22	63.65	0.30	7.94
-400	- 38	63.65		0.21	7.65
		100.0			

Table 2-17. Minus 38 um Cyclosizer and Cu-Zn Assay Results (Kidd Creek Survey No. 1)

Sample	Solids S.G.	Time (min)	Temp. (C)	Flowrate	Weight Percentages (%)		Assays % Cu	Assays % Zn
1. RMF	3.17	20.0	21.5	180	Cone 1	2.2	10.5	13.7
					2	8.2	9.84	15.1
					3	13.4	6.02	9.76
					4	13.8	4.80	7.86
					5	10.8	4.20	6.80
					Passing	51.6		
2. RMD	3.18	20.0	20.5	180	Cone 1	3.1	8.95	13.0
					2	9.4	7.36	12.3
					3	14.3	4.63	8.06
					4	14.5	3.88	6.79
					5	11.1	3.38	5.81
					Passing	47.6		
3. PBMF	3.33	20.0	20.5	180	Cone 1	6.1	6.81	9.98
					2	13.3	6.29	11.4
					3	16.3	4.14	7.76
					4	14.4	3.55	6.29
					5	10.1	3.76	6.68
					Passing	39.8		
4. PBMD	3.30	20.0	20.5	180	Cone 1	4.9	6.10	9.69
					2	12.4	6.19	11.6
					3	16.8	4.04	7.94
					4	15.3	3.44	6.78
					5	10.8	3.19	6.22
					Passing	39.8		
5. PCOF	3.21	20.0	21.5	180	Cone 1	3.9	6.70	10.9
					2	11.1	6.20	11.3
					3	16.1	4.10	7.76
					4	15.2	3.63	6.77
					5	11.2	3.19	5.99
					Passing	42.5		

Note: Cyclosizer calibration May 7, 1986. Base cut sizes (2.65 S.G.)
Cone no.1, 53µm; no.2, 33µm; no.3, 23µm; no.4, 15µm; no.5, 11µm

Table 2-17. (Cont'd.)

Sample	Solids S.G.	Time (min)	Temp. (C)	Flowrate	Weight Percentages (%)		Assays % Cu	% Zn
6. PCRC	3.93	20.0	21.5	180	Cone 1	2.3	24.6	2.76
					2	12.0	26.7	3.79
					3	16.4	25.4	4.41
					4	16.6	24.3	4.82
					5	12.1	24.0	5.29
					Passing	40.6		
7. PCRT	3.17	23.0	21.5	180	Cone 1	3.9	4.15	12.1
					2	10.4	3.15	13.3
					3	15.9	1.32	8.40
					4	15.1	0.74	7.10
					5	11.1	0.53	6.37
					Passing	43.6		
8. REGRIND COF	3.74	20.0	21.5	180	Cone 1	3.6	13.3	6.63
					2	11.5	14.2	9.47
					3	14.2	11.3	10.5
					4	14.6	9.66	11.6
					5	11.3	9.17	11.6
					Passing	44.8		
9. SBMF	3.59	20.0	21.5	180	Cone 1	13.9	4.83	5.59
					2	18.7	7.40	10.9
					3	16.0	4.99	8.87
					4	12.7	4.13	8.06
					5	8.3	3.79	7.66
					Passing	30.6		
10. SBMD	3.55	20.0	21.5	180	Cone 1	11.5	5.02	5.97
					2	16.9	7.01	10.3
					3	16.6	4.48	7.87
					4	13.7	3.80	7.24
					5	9.1	3.51	6.78
					Passing	32.2		
11. SCOF	3.40	21.0	21.5	180	Cone 1	4.3	8.08	10.2
					2	11.7	7.25	11.4
					3	16.2	4.56	8.40
					4	15.1	3.98	8.19
					5	11.0	3.68	7.71
					Passing	41.7		

Table 2-17 (Cont'd)

Sample	Solids S.G.	Time (min)	Temp. (C)	Flowrate	Weight Percentages (%)		Assays % Cu	% Zn
12. COPPER CONCENTRATE								
	4.06	20.0	21.5	180	Cone 1	4.1	28.2	1.83
					2	16.2	30.5	2.21
					3	17.7	28.6	2.54
					4	15.6	28.2	2.89
					5	10.7	27.9	2.91
					Passing	35.7		
13. COPPER TAILINGS								
	3.17	22.0	21.5	180	Cone 1	2.9	0.41	11.6
					2	9.8	0.29	14.1
					3	16.7	0.16	8.56
					4	16.0	0.14	7.08
					5	11.6	0.13	6.52
					Passing	43.0		

Survey No. 2

This was a complete sampling survey of the grinding and copper flotation circuits carried out on December 19, 1985. All the primary copper rougher flotation tailings were directed to secondary ball milling. The copper regrind circuit was completely closed circuited with copper secondary and cleaner flotation, with no recirculation to the secondary ball mill.

The samples were first weighed, dried and then re-weighed to determine percent solids, and specific gravity checks were performed on a number of subsamples, as summarized in Table 2-18. Samples were sub-sampled by riffling, as necessary, for screen analyses and complete assays were performed on each size fraction. The size distribution and copper and zinc assay data are summarized in Tables 2-19A through L. Assays for primary copper rougher flotation concentrate size fractions were not included.

All the minus 38 um samples were sent to McGill University where they were tested for specific gravity and then cyclo-sized. They were then returned to the plant for size by size assay determinations. The results are given in Table 2-20.

Bond laboratory rod mill and ball mill work index tests were also carried out, as well as on-site grindability testing, as described later.

Table 2-18. Kidd Creek Survey No. 2 Percent Solids and Specific Gravities

Sample	Dry Weight (g)	% Solids by Weight	Solids Specific Gravity
RMF	19,221	98.05	-
RMD-1	3,305	80.47	3.0
RMD-2	7,196	81.02	-
PBMF	15,821	76.87	3.1
PBMD	11,900	77.54	3.1
PCOF-1	1,278	51.99	-
PCOF-2	1,460	51.79	-
PCRFC	-	-	3.4
PCRFT-1	2,225	50.95	2.9
PCRFT-2	2,320	49.16	-
SBMF	3,273	73.11	3.2
SBMD	6,205	73.22	3.2
SCOF-1	2,010	46.09	2.8
SCOF-2	2,890	45.80	2.8
Copper Concentrate	-	-	3.9
Copper Tailings	-	-	-

Table 2-19. Kidd Creek Survey No. 2 Size and Assay Data

Table 2-19A. Survey No. 2 Size and Assay Data: RMF

Mesh (Tyler)	Screen Size (μm)	Size Distribution Ind.%	Size Distribution Cum.% Pass.	Assays % Cu	Assays % Zn
1	25,400	3.61	96.39	5.31	4.12
0.742	19,000	5.92	90.47	2.36	4.47
0.530	13,200	15.69	74.78	3.02	4.16
0.371	9,500	16.10	58.68	2.85	4.92
3	6,700	11.01	47.67	2.93	4.66
4	4,750	8.46	39.21	3.03	5.00
6	3,350	6.61	32.60	3.03	4.56
8	2,360	4.50	28.10	2.85	5.25
10	1,700	3.70	24.40	3.00	5.07
14	1,180	3.51	20.89	3.07	5.32
20	850	2.50	18.39	3.16	5.26
28	600	1.85	16.54	3.13	5.49
35	425	1.72	14.82	3.06	5.45
48	300	1.46	13.36	3.39	5.59
65	212	1.29	12.07	3.84	6.00
100	150	1.21	10.86	4.32	5.79
150	106	1.05	9.81	4.99	6.61
200	75	1.31	8.50	5.56	7.26
270	53	0.89	7.61	5.92	8.04
400	38	1.12	6.49	6.03	8.37
-400	- 38	6.49		4.18	6.23
		100.0			

Table 2-19B. Survey No. 2 Size and Assay Data: RMD

Mesh (Tyler)	Screen Size (μm)	Size Distributions Sample No. 1 Ind.%	Sample No. 1 Cum.% Pass.	Sample No. 2 Ind.%	Sample No. 2 Cum.% Pass.	Combined Assays % Cu	Combined Assays % Zn
1	25,400						
0.742	19,000						
0.530	13,200						
0.371	9,500	0	100.0	0	100.0		
3	6,700	0.07	99.93	0.15	99.85		
4	4,750	0.48	99.45	0.36	99.49	1.82	2.98
6	3,350	2.30	97.15	2.26	97.23		
8	2,360	6.79	90.36	6.37	90.86	2.42	3.62
10	1,700	9.34	81.02	9.50	81.36	2.72	4.44
14	1,180	12.19	68.83	12.76	68.60	2.97	4.60
20	850	9.71	59.12	10.11	58.49	2.72	4.53
28	600	7.54	51.58	7.78	50.71	2.73	4.68
35	425	6.71	44.87	6.83	43.88	2.84	4.73
48	300	5.49	39.38	5.41	38.47	3.06	4.97
65	212	4.63	34.75	4.57	33.90	3.39	5.03
100	150	4.03	30.72	3.82	30.08	3.85	5.19
150	106	3.08	27.64	3.04	27.04	4.49	5.37
200	75	3.31	24.33	3.05	23.99	4.90	5.93
270	53	2.61	21.72	2.60	21.39	5.52	6.65
400	38	3.70	18.02	3.49	17.90	5.38	7.04
-400	- 38	18.02		17.90		4.26	5.71
		100.0		100.0			

Table 2-19C. Survey No. 2 Size and Assay Data: PBMF

Mesh (Tyler)	Screen Size (μm)	Size Distribution Ind.%	Size Distribution Cum.% Pass.	Assays % Cu	Assays % Zn
1	25,400				
0.742	19,000				
0.530	13,200				
0.371	9,500				
3	6,700	0	100.0		
4	4,750	0.11	99.89		
6	3,350	0.80	99.09	2.42	4.39
8	2,360	2.65	96.44		
10	1,700	3.80	92.64	1.95	3.16
14	1,180	5.80	86.84	2.14	3.45
20	850	6.19	80.65	2.23	3.83
28	600	6.66	73.99	1.45	2.95
35	425	8.35	65.64	2.41	4.47
48	300	9.46	56.18	3.01	5.48
65	212	10.94	45.24	3.68	6.45
100	150	11.38	33.86	4.44	7.19
150	106	8.17	25.69	4.74	6.89
200	75	6.38	19.31	4.99	7.12
270	53	3.69	15.62	5.10	7.13
400	38	3.83	11.79	5.15	7.54
-400	- 38	11.79		4.20	6.17
		100.0			

Table 2-19D. Survey No. 2 Size and Assay Data: PBMD

Mesh (Tyler)	Screen Size (μm)	Size Distribution Ind.%	Size Distribution Cum.% Pass.	Assays % Cu	Assays % Zn
1	25,400				
0.742	19,000				
0.530	13,200				
0.371	9,500	0	100.0		
3	6,700	0.24	99.76		
4	4,750	0.27	99.49	1.32	1.36
6	3,350	0.81	98.68		
8	2,360	1.31	97.37		
10	1,700	1.89	95.48	1.47	2.41
14	1,180	3.11	92.37	1.67	2.87
20	850	4.12	88.25	2.04	3.80
28	600	5.12	83.13	2.24	4.17
35	425	7.38	75.75	2.22	4.18
48	300	9.04	66.71	2.65	4.92
65	212	11.12	55.59	3.33	5.87
100	150	11.08	44.51	3.96	6.29
150	106	9.18	35.33	4.35	6.36
200	75	7.83	27.50	4.57	6.47
270	53	4.98	22.52	4.75	6.96
400	38	5.38	17.14	4.88	7.08
-400	- 38	17.14		3.89	5.96
		100.0			

Table 2-19E. Survey No. 2 Size and Assay Data: PCOF

Mesh (Tyler)	Screen Size (μm)	Size Distributions Sample No. 1 Ind.%	Sample No. 1 Cum.% Pass.	Sample No. 2 Ind.%	Sample No. 2 Cum.% Pass.	Combined Assays % Cu	Combined Assays % Zn
1	25,400						
0.742	19,000						
0.530	13,200						
0.371	9,500						
3	6,700						
4	4,750						
6	3,350						
8	2,360						
10	1,700	0	100.0	0	100.0		
14	1,180	0.04	99.96	0.03	99.97		
20	850	0.07	99.89	0.05	99.92	0.37	0.47
28	600	0.28	99.61	0.26	99.66		
35	425	1.63	97.98	1.55	98.11		
48	300	3.84	94.14	3.73	94.38	0.47	0.43
65	212	6.21	87.93	6.09	88.29	0.74	0.94
100	150	7.96	79.97	7.93	80.36	1.55	2.64
150	106	8.09	71.88	8.11	72.25	2.82	4.67
200	75	9.45	62.43	9.55	62.70	3.79	5.82
270	53	7.83	54.60	7.57	55.13	4.11	6.31
400	38	10.44	44.16	10.34	44.79	4.58	6.82
-400	- 38	44.16		44.79		3.82	5.89
		100.0		100.0			

Table 2-19F. Survey No. 2 Size and Assay Data: PCRFT

Mesh (Tyler)	Screen Size (µm)	Size Distributions Sample No. 1 Ind.%	Sample No. 1 Cum.% Pass.	Sample No. 2 Ind.%	Sample No. 2 Cum.% Pass.	Combined Assays % Cu	Combined Assays % Zn
1	25,400						
0.742	19,000						
0.530	13,200						
0.371	9,500						
3	6,700						
4	4,750						
6	3,350						
8	2,360						
10	1,700	0	100.0	0	100.0		
14	1,180	0.02	99.98	0.01	99.99		
20	850	0.05	99.93	0.01	99.98		
28	600	0.31	99.62	0.14	99.84	0.30	0.37
35	425	1.76	97.86	1.11	98.73		
48	300	3.69	94.17	3.21	95.52	0.42	0.42
65	212	6.08	88.09	5.86	89.66	0.65	0.89
100	150	8.40	76.69	8.17	81.49	1.41	2.48
150	106	8.28	70.91	8.59	72.90	2.70	4.77
200	75	10.23	60.68	10.07	62.83	2.75	5.67
270	53	7.93	52.75	8.21	54.62	2.32	6.52
400	38	10.33	42.42	10.63	43.99	1.45	7.28
-400	- 38	42.42		43.99		0.55	5.77
		100.0		100.0			

Table 2-19G. Survey No. 2 Size and Assay Data: PCRFC

Mesh (Tyler)	Screen Size (µm)	Size Distribution Ind.%	Size Distribution Cum.% Pass.	Assays % Cu	Assays % Zn
1	25,400				
0.742	19,000			(not avail.)	
0.530	13,200				
0.371	9,500				
3	6,700				
4	4,750				
6	3,350				
8	2,360				
10	1,700	0	100.0		
14	1,180	0.03	99.97		
20	850	0.01	99.96		
28	600	0.01	99.95		
35	425	0.01	99.94		
48	300	0.15	99.79		
65	212	0.34	99.45		
100	150	0.38	99.07		
150	106	0.61	98.46		
200	75	2.67	95.79		
270	53	5.80	89.99		
400	38	13.45	76.54		
-400	- 38	76.54			
		100.00			

Table 2-19H. Survey No. 2 Size and Assay Data: SBMF

Mesh (Tyler)	Screen Size (μm)	Size Distribution Ind.%	Size Distribution Cum.% Pass.	Assays % Cu	Assays % Zn
1	25,400				
0.742	19,000				
0.530	13,200				
0.371	9,500				
3	6,700				
4	4,750				
6	3,350				
8	2,360				
10	1,700	0	100.0		
14	1,180	0.01	99.99		
20	850	0.04	99.95	0.53	0.80
28	600	0.28	99.67		
35	425	1.85	97.82		
48	300	4.50	93.32	0.62	0.80
65	212	7.56	85.76	0.79	1.15
100	150	10.47	75.29	1.74	3.09
150	106	11.71	63.58	3.79	6.48
200	75	16.10	47.48	4.94	8.85
270	53	13.45	34.03	3.36	6.81
400	38	13.15	20.88	3.02	7.64
-400	- 38	20.88		1.21	4.88
		100.0			

Table 2-19 I. Survey No. 2 Size and Assay Data: SBMD

Mesh (Tyler)	Screen Size (μm)	Size Distribution Ind.%	Size Distribution Cum.% Pass.	Assays % Cu	Assays % Zn
1	25,400				
0.742	19,000				
0.530	13,200				
0.371	9,500				
3	6,700				
4	4,750				
6	3,350				
8	2,360				
10	1,700				
14	1,180	(trace)	100.0		
20	850	0.02	99.98		
28	600	0.10	99.88	1.48	2.84
35	425	0.66	99.22		
48	300	1.56	97.66		
65	212	2.93	94.73	0.95	1.69
100	150	5.89	88.84	1.48	2.58
150	106	9.72	79.12	3.01	5.12
200	75	16.68	62.44	4.08	7.31
270	53	12.49	49.95	3.85	7.44
400	38	15.48	34.47	2.98	6.77
-400	- 38	34.47		2.04	5.73
		100.0			

Table 2-19J. Survey No. 2 Size and Assay Data: SCOF

Mesh (Tyler)	Screen Size (μm)	Size Distributions: Sample No. 1 Ind.%	Sample No. 1 Cum.% Pass.	Sample No. 2 Ind.%	Sample No. 2 Cum.% Pass.	Combined Assays % Cu	Combined Assays % Zn
1	25,400						
0.742	19,000						
0.530	13,200						
0.371	9,500						
3	6,700						
4	4,750						
6	3,350						
8	2,360						
10	1,700	0	100.0	0	100.0		
14	1,180	0.01	99.99	0.02	99.98		
20	850	0.01	99.98	0.03	99.95		
28	600	0.01	99.97	0.01	99.94		
35	425	0.02	99.95	0.02	99.92	0.32	0.48
48	300	0.07	99.88	0.07	99.85		
65	212	0.66	99.22	0.55	99.30		
100	150	2.66	96.56	2.44	96.86		
150	106	5.44	91.12	5.25	91.61	0.41	0.61
200	75	8.76	82.36	8.88	82.73	0.89	1.85
270	53	8.83	73.53	8.85	73.88	1.76	4.51
400	38	14.31	59.22	14.63	59.25	2.01	6.60
-400	- 38	59.22		59.25		1.24	5.59
		100.0		100.0			

Table 2-19K. Survey No. 2 Size and Assay Data:

Copper Concentrate

Mesh (Tyler)	Screen Size (μm)	Size Distribution Ind.%	Size Distribution Cum.% Pass.	Assays % Cu	Assays % Zn
1	25,400				
0.742	19,000				
0.530	13,200				
0.371	9,500				
3	6,700				
4	4,750				
6	3,350				
8	2,360				
10	1,700				
14	1,180	0	100.0		
20	850	0.09	99.91		
28	600	0.16	99.75		
35	425	0.10	99.65		
48	300	0.18	99.47	29.4	1.18
65	212	0.16	99.31		
100	150	0.14	99.17		
150	106	0.30	98.87		
200	75	2.12	96.75		
270	53	8.09	88.66	29.7	1.42
400	38	19.01	69.65	28.6	2.15
-400	- 38	69.65		25.3	4.37
		100.0			

Table 2-19L. Survey No. 2 Size and Assay Data:

Copper Tailings

Mesh (Tyler)	Screen Size (μm)	Size Distribution Ind.%	Cum.% Pass.	Assays % Cu	% Zn
1	25,400				
0.742	19,000				
0.530	13,200				
0.371	9,500				
3	6,700				
4	4,750				
6	3,350				
8	2,360				
10	1,700	0	100.0		
14	1,180	0.11	98.89		
20	850	0.02	99.87		
28	600	0.08	99.79		
35	425	0.11	99.68	0.26	0.43
48	300	0.23	99.45		
65	212	0.78	98.67		
100	150	2.59	96.08		
150	106	5.66	90.42	0.23	0.65
200	75	8.72	81.70	0.28	1.94
270	53	8.43	73.27	0.30	4.51
400	38	13.73	59.54	0.25	6.53
-400	- 38	59.54		0.19	5.60
		100.0			

Table 2-20. Minus 38 um Cyclosizer and Cu-Zn Assay Results (Survey No. 2)

Sample	Solids S.G.	Time (min)	Temp. (C)	Flowrate	Weight Percentages (%)		Assays % Cu	% Zn
1. RMF	3.11	21.5	16.5	180	Cone 1	1.3	7.96	8.23
					2	6.5	11.0	14.5
					3	13.0	5.99	8.40
					4	14.4	4.55	6.72
					5	11.1	3.90	5.63
					Passing	53.7		
2. RMD	3.10	21.5	16.5	180	Cone 1	1.3	8.37	8.89
					2	7.0	9.58	12.9
					3	13.8	5.09	7.28
					4	15.1	4.02	5.59
					5	11.8	2.22	5.40
					Passing	51.0		
3. PBMF	3.19	21.5	18	180	Cone 1	3.4	6.22	7.59
					2	10.7	7.87	12.3
					3	16.4	4.65	6.85
					4	15.5	3.93	6.08
					5	11.1	3.41	5.50
					Passing	42.9		
4. PBMD	3.21	20.0	20.5	180	Cone 1	3.8	6.37	8.39
					2	10.9	6.85	10.7
					3	16.6	4.27	6.98
					4	15.7	3.36	5.51
					5	11.4	3.10	5.04
					Passing	41.6		
5. PCOF	3.15	21.5	18	180	Cone 1	2.0	6.53	8.20
					2	8.5	7.75	11.8
					3	15.5	4.46	7.30
					4	15.9	3.72	6.02
					5	11.8	3.33	5.27
					Passing	46.3		

Note: Cyclosizer calibration May 7, 1986. Base cut sizes (2.65 S.G.)
Cone no.1, 53μm; no.2, 33μm; no.3, 23μm; no.4, 15μm; no.5, 11μm

Table 2-20 (Cont'd)

Sample	Solids S.G.	Time (min)	Temp. (C)	Flowrate	Weight Percentages (%)		Assays % Cu	% Zn
6. PCRC	3.86	21.5	16.5	180	Cone 1	1.5	21.9	2.38
					2	12.0	26.6	4.12
					3	16.4	24.9	5.25
					4	15.9	23.1	6.10
					5	11.6	21.1	6.15
					Passing	42.6		
7. PCRT	3.06	21.5	16.5	180	Cone 1	1.5	1.21	8.61
					2	7.1	1.32	14.2
					3	15.2	0.46	6.69
					4	16.2	0.30	5.54
					5	12.0	0.27	4.83
					Passing	47.7		
8. SBMF	3.28	31.5	20.5	180	Cone 1	8.4	2.04	6.16
					2	13.8	2.84	11.8
					3	16.9	1.46	6.91
					4	14.4	1.08	5.62
					5	9.8	0.96	5.00
					Passing	36.7		
9. SBMD	3.20	21.5	16.5	180	Cone 1	3.6	1.94	4.61
					2	11.0	3.90	11.3
					3	17.3	1.92	5.99
					4	16.4	1.55	5.17
					5	11.4	1.39	4.70
					Passing	40.3		
10. SCOF	3.08	20.0	20.5	180	Cone 1	2.5	2.67	9.68
					2	9.0	2.11	9.71
					3	16.7	1.27	6.48
					4	16.2	0.95	5.30
					5	11.7	0.87	6.08
					Passing	43.9		

Table 2-20 (Cont'd)

Sample	Solids S.G.	Time (min)	Temp. (C)	Flowrate	Weight Percentages (%)		% Cu	% Zn
11. COPPER CONCENTRATE								
	4.03	20.0	20.5	180	Cone 1	4.4	25.9	2.36
					2	18.0	27.0	3.15
					3	18.6	26.3	3.86
					4	15.6	25.5	4.74
					5	10.3	25.9	5.09
					Passing	33.1		
12. COPPER TAILINGS								
	3.09	20.0	20.5	180	Cone 1	1.9	0.37	10.7
					2	7.4	0.23	13.6
					3	15.6	0.11	6.57
					4	16.3	0.09	5.50
					5	12.2	0.09	4.85
					Passing	46.6		

2.4.3.2 Mass Balance and Data Adjustment

The B circuit feed rate during sample survey no. 1 was 126.6 DMTPH, as calculated from the rod mill feed totalizer readings and the percentage of moisture in the rod mill feed sample. This is also the solids feed rate to the primary ball milling circuit.

The feed to the secondary ball mill circuit consists of primary copper rougher flotation tailings plus the copper regrind circuit (cyclone overflow) product. Based on the overall copper assays calculated for the feed, concentrate, and tailings of the rougher flotation circuit, 96% of the solids report to the secondary ball mill circuit. As well, the regrind cyclone overflow rate can be calculated from the individual and combined copper assays of the two streams. Total tonnages to final copper concentrate and tailings can be calculated from their individual copper assays, and that of the primary cyclone overflow. The relative mass splits (RMF=100) and tonnages throughout the circuit were thus calculated as follows (see section 2.4.1 for nomenclature).

Table 2-21. Kidd Creek Survey No. 1 Solids Flowrates

	Relative Solids Mass (RMF=100)	Tonnage (tonnes/h)
RMF	100	126.6
PCRFT	96	121.5
PCRFC	4	5.1
CRCOF	22	27.9
SCOF	118	149.4
Cu Concentrate	10.7	13.5
Cu Tailings	89.3	113.1

For survey no. 2, the rougher copper flotation concentrate assays were not available. However, a material balance using the -400 mesh size fraction weights showed approximately the same split, that is 96% to secondary ball milling. No material from the copper regrind circuit was sent to secondary ball milling. Final copper concentrate and tailings streams flowrates were calculated as described for the earlier survey.

Table 2-22. Kidd Creek Survey No. 2 Solids Flowrates

	Relative Solids Mass (RMF=100)	Tonnage (tonnes/h)
RMF	100	131.8
PCRFT	96	126.5
PCRFC	4	5.3
CRCOF	0	0
SCOF	96	126.5
Cu Concentrate	11.7	15.4
Cu Tailings	88.3	116.4

The size-mass balance equation for the primary ball milling circuit is, for any size class:

Mass in circuit feed (i.e., rod mill discharge) + Mass in ball mill discharge	=	Mass in circuit product (cyclone overflow) + Mass in cyclone underflow

In mathematical terms:

$$((M_{RMD} \cdot X) + (M_{BMD} \cdot X \cdot N) = (M_{COF} \cdot X) + (M_{CUF} \cdot X \cdot N)$$

given M = fraction mass in the different streams,
RMD (rod mill discharge),
BMD (ball mill discharge),
COF (cyclone overflotw),
CUF (cylone underflow).

X = new feed rate of solids
N = circulating load ratio (cyclone underflow to cyclone overflow solids).

In general, the better the raw data, the simpler the method that will suffice for balancing. The primary ball mill circuit size distribution data from survey no. 1 (balanced using the computer program based on the method described by Smith and Ichiyen (1973)), are shown in Table 2-23, and reveal minor adjustment of the data. It can be concluded that the circuit operation was stable during the sampling, and that the sampling and sample handling and analysis methodology employed was effective. Note that these data also permit calculation of the cyclone feed size distribution, so sampling of this stream is not required during the sample run. The other results from the mass balance calculations for both sampling runs for primary and secondary ball milling circuits are given in Tables 2-24, 2-25, and 2-26.

Table 2-23. Raw and Adjusted Size Data, Primary Ball Mill Circuit (Kidd Creek Survey No. 1)

CIRCULATING LOAD (%): 434

PRODUCT IDENTIFICATION

1 ROD MILL DISCHARGE
2 BALL MILL DISCHARGE
3 CYCLONE UNDERFLOW
4 CYCLONE OVERFLOW

UNADJUSTED SIZE DATA

MESH	MICRONS	1	2	3	4
6	3360	1.67	.4	.57	0
8	2360	4.5	.7	1.6	0
10	1700	8.45	1.22	3.14	0
14	1180	12.26	2.41	5.28	.02
20	840	10.73	3.12	5.77	.02
28	600	8.25	3.94	6.08	.08
35	420	7.33	6.03	7.71	.82
48	300	5.9	8.22	8.85	2.82
65	210	4.88	10.79	10.52	5.13
100	150	4.16	12.96	12.07	7.05
150	105	3.24	10.54	9.4	7.84
200	74	3.25	8.75	7.47	10.04
270	53	2.72	5.57	4.46	7.75
400	37	3.66	6.1	4.27	11.03
-400	-37	19	19.25	12.81	47.4

ADJUSTED SIZE DATA

MESH	MICRONS	1	2	3	4
6	3360	1.6	.31	.68	.01
8	2360	4.42	.67	1.68	0
10	1700	8.43	1.22	3.16	0
14	1180	12.29	2.42	5.25	.02
20	840	10.81	3.16	5.65	.02
28	600	8.31	4.02	5.92	.08
35	420	7.35	6.1	7.6	.82
48	300	5.9	8.18	8.89	2.82
65	210	4.87	10.68	10.62	5.14
100	150	4.16	12.84	12.17	7.06
150	105	3.24	10.5	9.44	7.85
200	74	3.26	8.91	7.35	9.99
270	53	2.72	5.6	4.44	7.74
400	37	3.65	6.02	4.31	11.08
PASSING		18.99	19.37	12.83	47.37

Table 2-24. Raw and Adjusted Size Data, Secondary Ball Mill Circuit (Kidd Creek Survey No. 1)

CIRCULATING LOAD (%): 228

PRODUCT IDENTIFICATION

1 NEW CIRCUIT FEED
2 BALL MILL DISCHARGE
3 CYCLONE UNDERFLOW
4 CYCLONE OVERFLOW

UNADJUSTED SIZE DATA

MESH	MICRONS	1	2	3	4
6	3360	0	0	0	0
8	2360	0	0	0	0
10	1700	0	0	.01	0
14	1180	.02	.02	.01	.01
20	840	.03	.03	.02	.01
28	600	.11	.05	.08	.01
35	420	.85	.2	.56	.03
48	300	2.74	.86	1.97	.05
65	210	4.72	2.5	4.59	.16
100	150	6.31	5.93	9.16	.94
150	105	6.84	9.99	11.94	3.18
200	74	8.58	19.46	22.14	6
270	53	7.61	16.52	14.66	6.63
400	37	11.28	18.81	17.07	13.42
-400	-37	50.92	25.69	17.79	69.56

ADJUSTED SIZE DATA

MESH	MICRONS	1	2	3	4
6	3360	0	0	0	0
8	2360	0	0	0	0
10	1700	0	0	.01	0
14	1180	.02	.01	.02	.01
20	840	.03	.02	.03	.01
28	600	.11	.04	.09	.01
35	420	.85	.2	.56	.03
48	300	2.71	.84	2.01	.05
65	210	4.75	2.52	4.53	.16
100	150	6.44	6.19	8.6	.93
150	105	6.87	10.13	11.75	3.17
200	74	8.64	20.12	21.29	5.97
270	53	7.48	15.3	15.63	6.73
400	37	11.21	18.41	17.4	13.51
PASSING		50.9	26.2	18.08	69.41

Table 2-25. Raw and Adjusted Size Data, Primary Ball Mill Circuit (Kidd Creek Survey No. 2)

CIRCULATING LOAD (%): 494

PRODUCT IDENTIFICATION

1 ROD MILL DISCHARGE
2 BALL MILL DISCHARGE
3 CYCLONE UNDERFLOW
4 CYCLONE OVERFLOW

UNADJUSTED SIZE DATA

MESH	MICRONS	1	2	3	4
6	3360	2.81	1.32	.91	0
8	2360	6.58	1.31	2.65	0
10	1700	9.42	1.89	3.8	0
14	1180	12.47	3.11	5.8	.04
20	840	9.91	4.12	6.19	.06
28	600	7.66	5.12	6.66	.27
35	420	6.77	7.38	8.35	1.59
48	300	5.45	9.04	9.46	3.78
65	210	4.6	11.12	10.94	6.15
100	150	3.93	11.08	11.38	7.94
150	105	3.06	9.18	8.17	8.1
200	74	3.18	7.83	6.38	9.5
270	53	2.61	4.98	3.69	7.7
400	37	3.6	5.38	3.83	10.39
-400	-37	17.96	17.14	11.79	44.48

ADJUSTED SIZE DATA

MESH	MICRONS	1	2	3	4
6	3360	2.51	.77	1.28	.02
8	2360	6.58	1.31	2.64	0
10	1700	9.42	1.89	3.8	0
14	1180	12.56	3.15	5.69	.04
20	840	9.93	4.14	6.14	.06
28	600	7.67	5.14	6.63	.27
35	420	6.76	7.35	8.39	1.59
48	300	5.45	9.08	9.42	3.78
65	210	4.6	11.19	10.87	6.15
100	150	3.95	11.61	10.82	7.88
150	105	3.06	9.19	8.17	8.1
200	74	3.18	7.73	6.45	9.53
270	53	2.6	4.83	3.78	7.76
400	37	3.59	5.28	3.89	10.46
PASSING		18.13	17.34	12.03	44.37

Table 2-26. Raw and Adjusted Size Data, Secondary Ball Mill Circuit (Kidd Creek Survey No. 2)

CIRCULATING LOAD (%): 120

PRODUCT IDENTIFICATION

1 NEW CIRCUIT FEED
2 BALL MILL DISCHARGE
3 CYCLONE UNDERFLOW
4 CYCLONE OVERFLOW

UNADJUSTED SIZE DATA

MESH	MICRONS	1	2	3	4
6	3360	0	0	0	0
8	2360	0	0	0	0
10	1700	0	0	0	0
14	1180	.02	0	.01	.02
20	840	.03	.02	.04	.02
28	600	.22	.1	.28	.01
35	420	1.44	.66	1.85	.02
48	300	3.45	1.56	4.5	.07
65	210	5.97	2.93	7.56	.61
100	150	8.28	5.89	10.47	2.55
150	105	8.68	9.72	11.71	5.34
200	74	10.15	16.68	16.1	8.82
270	53	8.07	12.49	13.45	8.84
400	37	10.48	15.48	13.15	14.47
-400	-37	43.21	34.47	20.88	59.23

ADJUSTED SIZE DATA

MESH	MICRONS	1	2	3	4
6	3360	0	0	0	0
8	2360	0	0	0	0
10	1700	0	0	0	0
14	1180	.02	0	.01	.02
20	840	.03	.02	.04	.02
28	600	.22	.1	.28	.01
35	420	1.44	.66	1.85	.02
48	300	3.49	1.58	4.43	.07
65	210	6.02	2.95	7.46	.61
100	150	8.22	5.85	10.57	2.56
150	105	8.51	9.47	12.06	5.41
200	74	9.92	15.98	16.75	9
270	53	8.28	13.06	12.8	8.59
400	37	10.63	15.85	12.88	14.2
PASSING		43.2	34.47	20.88	59.51

CHAPTER 3

ROD MILLING

3.1 REVIEW AND EVALUATION OF ROD MILL DESIGN AND OPERATING PRACTICES

3.1.1 Introduction

This section reports the findings of a detailed review of available technical information on the operation of rod mills. A table summarizing industrial rod mill operating data collected during this review is given in Appendix C. Because of combined interactions, the effects of specific operating variables may be difficult to assess, and even at times appear erratic. It is difficult to rationalize plant testwork or adjustments when the results are highly unpredictable. The purposes of this review, therefore, were:

a) to establish what basic facts have been learned about rod mill performance;

b) to define some general guidelines for efficient rod mill design and operation; and,

c) to establish a basis for specific recommendations for plant tests or adjustments with minimum risk, and maximum potential for circuit performance improvement.

3.1.2 Historical Background of Developments in Rod Milling

According to Lowrison (1974), the use of rods rolling in a cylinder to cause size reduction was patented by Zanenko in 1870. Reports of early operating plant experiences (Crawford, 1934; Binsacca, 1934; anon, 1937; Taggart, 1945; Myers et al, 1947; Craig, 1950) reveal that rod mills were originally viewed as a virtual equivalent to the ball mill, with neither having preference for either coarse or fine grinding. Ball mills could use up to 5 inch balls to handle crusher plant discharge (Davis, 1919), while on the other hand, rod mills could be used to grind as fine as 80% passing 75 um (200 mesh). The early rod mills were normally operated at low (50 to 60 percent) critical speeds, and usually in closed circuit with classifiers, whether applied at the coarse or fine end of the grinding scale.

While ball mills were by far the more popular of the two, rod mills began to find their place in mineral processing during the 1930s. In 1945, Taggart (1945) reported on 20 rod mill installations (compared to about 100 ball mills), most of which were grinding to 3360 to 595 um (6 to 28 mesh) top size, normally in open-circuit. As a "fine crushing" machine, it was largely responsible for the obsolescense of the roll crusher, with the overwhelming advantages of higher capacity, open-circuit, wet operation, capable of producing a similarly granular product (with few fines) suitable for gravity separation. In plants requiring fine grinding, it was evolving as a suitable intermediate stage to relieve the difficulties and inefficiency of fine size reduction with crushers, and coarse grinding in ball mills. Such evolution was typefied, and fortunately well documented, by developments at the Tennessee Copper Company, largely through the work of J.F. Myers and F.M. Lewis (1943, 1946, 1946, 1953).

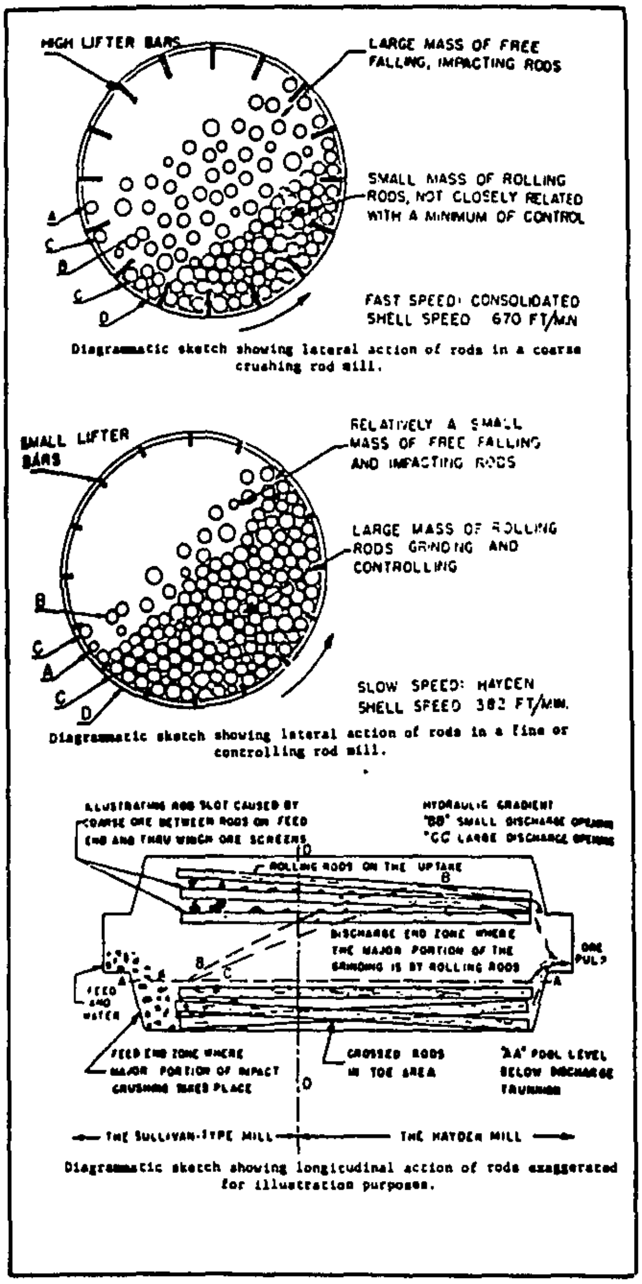
Although some other successful plant experiences were being reported (Tuck, 1938; Stahl, 1944; Strohl and Schwellenbach, 1950; Banks, 1953; McLachlan et al, 1953), rod milling was still not very common practice before 1960. The 1957 book The Milling of Canadian Ores (1957) reports less than one third (of over a hundred) of gold and base metal concentrating plants used rod mills. By 1978, however, Milling Practice in Canada (1978) indicates that this figure had risen to over 75% despite the introduction of autogenous and semi-autogenous milling during the same period. Plant descriptions in these volumes which include details on rod milling are listed in the references. Please also refer to the table of operating plant data in Appendix C.

With the growth of industrial rod milling came the extensive research efforts of one of the industry's major mill suppliers, Allis-Chalmers Corporation. Except for extensive tests with a pilot plant mill (Mitchell et al, 1954, 1955), the work was largely carried out under the leadership of F.C. Bond. Bond had collaborated earlier with J.F. Myers (1947) in his efforts to correlate rod mill plant and laboratory grinding data, and of course incorporated rod milling into his comminution "Third Theory" (Bond, 1951). He also identified factors influencing rod milling performance, recommended operating and design characteristics, and introduced correction factors for scale up to non-standard operating conditions (Bond, 1950, 1953, 1960, 1961). Rowland has carried on development of the Bond Work Index for mill sizing and circuit evaluation (Rowland and Nealy, 1969; Rowland, 1972, 1982).

Reports on a number of recent operating plant experiences were also reviewed (Steane, 1976; Zickar et al, 1981; Basnardo et al, 1984; del Villar, 1985), as were findings from some Australian experiences as reported by Lynch (1977). Work on mathematical modelling of rod mills has been carried out since 1964, but these papers provided little information on the effect of rod milling design or operating variables (Calcott and Lynch, 1964; Grandy and Fuerstenau, 1970; Mular and Henry, 1971; Fournier and Smith, 1972; Heyes et al, 1973; Shoji and Austin, 1974), concentrating mainly on development of mathematical expressions to describe the process.

3.1.3 Basic Functioning of a Rod Mill

The crushing action of the rod mass in a coarse grinding rod mill has been described by Myers (1953) and is illustrated in diagrams taken from his work in Figure 3-1. The progressive sizing of material from the feed to the discharge end of the mill by the tapered slot openings between the rods, and the resulting effective "internal classification" has been widely observed and documented (for example, see Myers and Lewis' earlier work, 1943, 1946, 1949). Apparently, the rod mass acts as an effective screen, holding back coarse paticles from progressing along the length of the mill until crushed. As well, the coarser particles tend to take the brunt of the rod impacts, and hold the rods apart, preventing the finer particles from being overground.



Diagrammatic sketch showing lateral action of rods in a coarse crushing rod mill.

Diagrammatic sketch showing lateral action of rods in a fine or controlling rod mill.

Diagrammatic sketch showing longitudinal action of rods exaggerated for illustration purposes.

Figure 3-1. Lateral and Longitudinal Rod Action (Myers, 1953)

3.1.4 Evaluation of Variables in Rod Milling

3.1.4.1 General

Energy efficiency in grinding is directly related to circuit economic performance because of the close interdependence of energy and media consumption, and their combined dominance of total direct grinding costs. Energy usage is the criterion by which grinding circuit process performance can be technically evaluated using the Bond work index approach. As well, appreciation of the nature of equipment energy consumption is required to determine the scope for changes to the power applied in each comminution stage. The following discussion therefore focuses on the effects of design or operating variables on both power draw and power efficiency, as well as giving consideration to some practical constraints for operating ease and stability.

3.1.4.2 Basics of Rod Mill Power Draw

Allis-Chalmer's estimates of rod mill power draw (Rowland, 1982) are based on an empirical formula which covers a wide range of mill dimensions, and the normal operating range of mill load of 35 to 50% of mill volume, and speeds of 50 to 80% of critical.

$$kWr = CD^{.34}\,(6.3-5.4\ Vp)\ Cs$$

kWr = kilowatts per unit weight of rods.

C = a constant, depending on the system of units.

D = mill diameter inside liners.

Vp = fraction of mill volume rod load (ground out mill).

Cs = fraction of critical speed.

The kilowatt draw of the mill is then obtained by multiplying kWr by the weight of rods in the mill, which yields:

$$kW = \left(\frac{D^2}{4} L\ Vp\rho\right)\left(C\ D^{.34}\ (6.3-5.4\ Vp)\right) Cs$$

kW = mill power draw at the pinion shaft, kilowatts.

ρ = rod charge bulk density.

L = mill effective working length.

Although different manufactures' nominal mill dimensions may vary, they are normally inside shell diameter, by effective length inside end liners at the shell.

Of note are the following observations:

1) Power draw is related directly to mill length, and, empirically to the diameter to the power 2.34. (Theoretically, this exponent should be 2.5, (Bond, 1961)).

2) Power draw is directly related to mill speed (in r.p.m., or, fraction of critical speed) over the normal range.

3) Power draw increases with charge level to a peak at approximately Vp=.58, as shown in Figure 3-2.

4) Power draw is not significantly affected by the amount of the slurry in the mill. While the presence of the

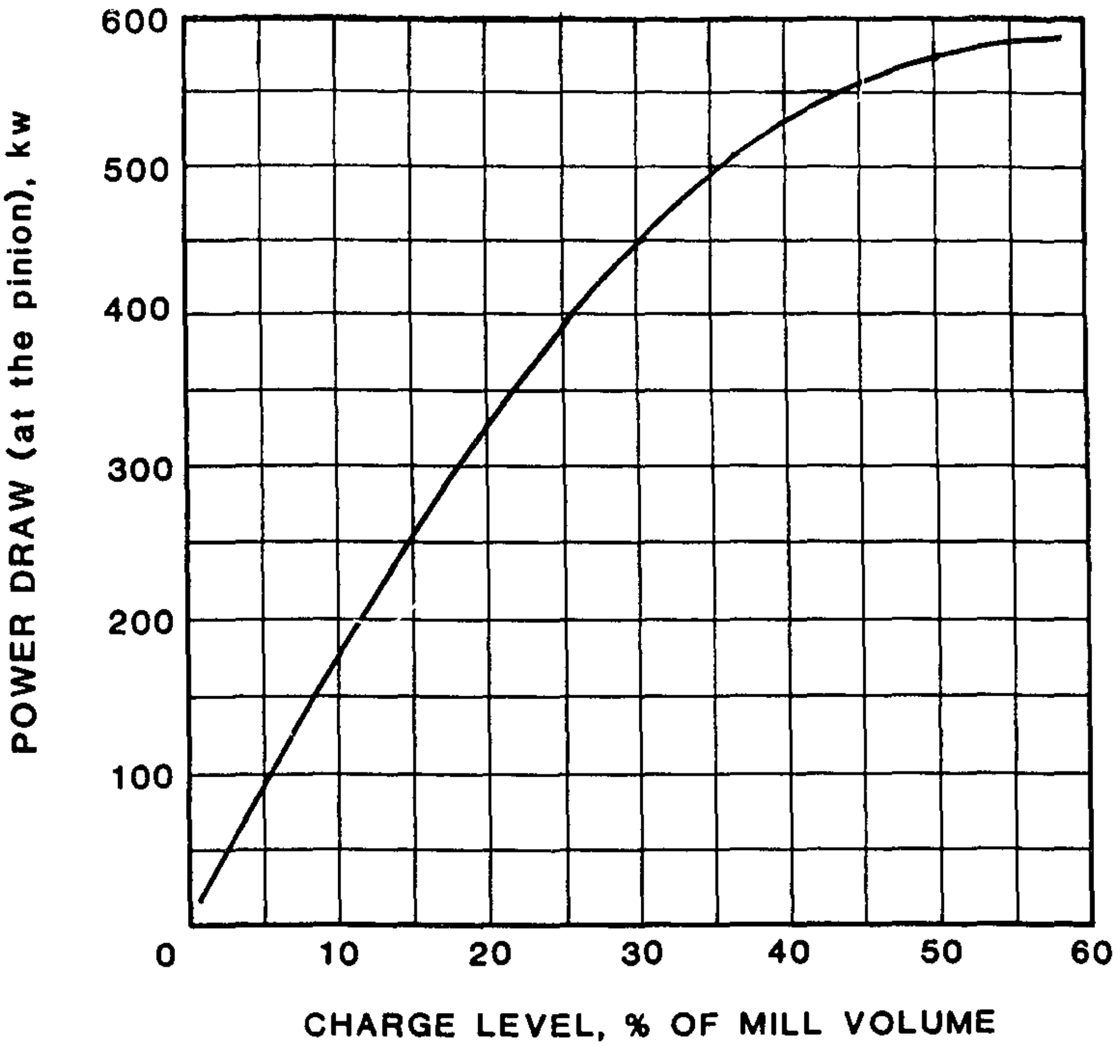


Figure 3-2. Rod Mill Power vrs. Loading (after Rowland, 1982)

slurry will affect the charge density, the charge swelling (moving the centre of gravity of the mass near the mill centre line) and added lubricity are offsetting factors. The net effect of initially adding feed to the mill is a small but noticeable reduction in power draw.

5) The bulk density of a new (or freshly culled), graded rod charge is 6247 kg/m^3 (390 lbs. per cubic foot). Due to lack of culling (i.e., removal of broken rods) and the greater tendency for rods to break in mills of increased size, the estimated bulk density of the rod charge may be reduced accordingly (Rowland, 1982).

Note that the above equation empirically relates mill power consumption over a wide range of mill sizes, on various slurries, as well as over a fairly broad range of mill speed and load levels, all combined into a single expression. Effects such as slurry density, media size, liner condition, and trunnion opening size are not included.

3.1.4.3 Rod Milling Variables

<u>Mill Dimensions</u>

A. Power Draw

There is general agreement that mill power draw is directly proportional to the mill length, and increases with mill diameter to the power 2.3 to 2.5. Larger discharge trunnion openings can increase power draw as much as 18 percent (Myers and Lewis, 1946). This may be partly due to the ease with which scrap will clear itself, which is also assisted by having conical (versus flat) mill ends (Myers, 1953), as well as the reduced buoyancy effect on grinding media with lower slurry level. Note mill diameter increases as the liners wear, but increased power draw is at least partically offset by loss of lift (McIvor, 1983).

B. Power Efficiency

Bond (1961) states that efficiency increases, in general, with mill diameter (about 10 percent from a 1.83 m to a 3.66 m mill), but there is no single plant experience to verify this. He also recommends higher length to diameter ratios for better efficiency in high reduction ratio applications.

At Tennessee Copper, a 1.83 m (6 foot) diameter mill could handle 25 mm (1 inch) feed, whereas a 1.52 m (5 foot) mill could not, although overall efficiency was not reported to be any better (Myers and Lewis, 1943). The additional work performed, in terms of surface area, did not increase as much as power draw with the larger trunnion openings (Myers and Lewis, 1946). Giving the mill a flared internal shape (wider at the feed end) to better suit the shape of the rod charge improved efficiency by 19 percent (Myers, 1953).

Rowland (1972) notes that rods break up at a larger size in larger diameter mills (51 to 76 mm rods in 4.11 m diameter, versus 25 to 51 mm rods in 3.05 m diameter), resulting in lost efficiency. He also notes that because power consumption (and design tonnage) increase exponentially with mill diameter, so does solids loading rate per unit mill volume, which at some point will limit mill production capacity.

C. Practical Considerations

Rowland (1982) and Lynch (1977) both recommend length to diameter ratios of at least approximately 1.3 to prevent rod tangles. Some plant experiences contradict this, for example;

1.83 m diam. x 1.22 m long, 58% Cs (Tuck, 1938);
3.51 m diam. x 3.66 m long, 80% Cs (Strohl and Schwellenbach, 1950);
3.05 m diam. x 3.66 m long, 68% Cs (McLachlan et al, 1953).

None of these report rod tangling problems. On the other hand, the 4.11 by 5.49 m (12 1/2 x 18 ft.) mill at Frood-Stobie, 66% Cs, initially experienced numerous tangles (Zickar et al, 1981).

Mill Speed

A. Power Draw

Numerous plant experiences and general studies show that both mill power draw and production rate increase in direct proportion with mill speed up to at least 80 percent of critical (Myers and Lewis, 1943 and 1946; Banks, 1953; Bond, 1961,; Rowland, 1982; Steane, 1976). At much beyond this limit, the charge will start to centrifuge, and power draw will level off.

B. Power Efficiency

Power draw and production rate increasing proportionately means that no change in efficiency occurs as mill speed is varied.

Tests in a 0.76 m (2 1/2 foot) diameter pilot plant rod mill (with overflow discharge) showed best efficiency with high speed (80 percent Cs) combined with high feed rate (Mitchell et al, 1955). Experience at Sullivan is unique, where a 3.51 m diameter by 3.66 m long (11 1/2 x 12 ft.) rod mill at over 80% of Cs, could handle high feed rates of extremely coarse material, even to the point of discharge of oversize, without overloading or rod tangling (Banks, 1953). Myers concluded that this indicated that higher speeds would be more effective for coarser feeds (1953).

Historically, rod mills were operated at very low critical speeds because of their early use as fine grinding machines, or to emulate the action of roll crushers which they were installed to replace. However, many successful higher speed installations, usually sped up from lower speeds, indicate that there should be little hesitation to run this equipment at similar speeds used for ball milling (see Table 3-1). A potential for reduced media consumption may exist at higher speeds as a result of reduced charge level for the same power draw. This will be discussed in more detail under the topic of media utilization.

Table 3-1. High Speed Rod Mill Installations

Installation*	Mill Size (dia.x length)	Critical Speed
Tennessee, Isabella	1.83 m x 3.66 m (6 x 12 ft)	74%
Frood Stobie	4.11 m x 5.49 m (13.5 x 18 ft)	74%
Canex Tungsten	1.83 m x 3.66 m (6 x 12 ft)	75%
East Driefontein	2.74 m x 3.66 m (9 x 12 ft)	75%
Craigmont	2.90 m x 3.66 m (9.5 x 12 ft)	75%
Tennessee, Isabella	1.83 m x 3.66 m (6 x 9 ft)	76%
Tennessee, London	1.83 m x 3.66 m (6 x 12 ft)	76%
Granby	2.44 m x 3.66 m (8 x 12 ft)	76%
Kakanda	2.13 m x 3.66 m (7 x 12 ft)	77%
Saaiplaas	2.44 m x 3.66 m (8 x 12 ft)	77%
Virtasalmi	2.21 m x 4.72 m (7.25 x 15.5 ft)	78%
Gibraltar	4.11 m x 6.10 m (13.5 x 20 ft)	78%
Sullivan	3.51 m x 3.66 m (11.5 x 12 ft)	81%
Kloof	2.74 m x 3.66 m (9 x 12 ft)	82%

* Refer to Appendix C for source references.

C. Practical Considerations

Bond recommended that rod mill speeds should be limited, decreasing as mill diameter is increased, in order to avoid excessive liner wear (Bond, 1961). However, many operating mills have been sped up to close to 80 percent of critical speed without any reports of excessive wear, or any other detrimental effects. At Sullivan

(80 percent Cs) rod consumption was extremely low (Mitchell et al, 1955). Discharge of round, rather than the normal eliptical shape, of broken rod fragments indicated a minimum of sliding action in the charge. In this installation, high mill speed afforded all the advantages of the following:

1. high power draw without having to use an excessively high charge level;
2. handling a coarse feed;
3. low metal wear.

Zickar (1981) also reported that frequency of rod tangles was reduced by operating at a higher speed (74 versus 66 percent of Cs) in conjunction with the use of larger rods.

It has been shown that it is the combination of both mill speed and liner design that induces lift to the charge, thereby together affecting mill power draw and efficiency, and that an optimum lifter spacing to height ratio is about 4 to 1 for all types of mills (McIvor, 1983). Although it appears that variations in grinding performance with liner design and condition may be too subtle to draw any general conclusions, liner condition should be noted when data is being collected, and incorporated into any power draw versus charge level evaluations. Different operators' perceptions of mill performance before and after worn liners are replaced are at variance.

Feed Preparation

A. Power Draw

There is no reported evidence that feed size has any effect on rod mill power draw. Because the solids in the rod mill are known to appreciably expand the mill charge (as observed in the charge level before and after grinding out the mill), there would likely be a tendency for power draw to be lower with coarser feeds and at higher feed rates. However, loss of power draw due to "overloading" (as sometimes reported in ball mills) or as an indication of an impending rod tangle, has not been reported.

B. Power Efficiency

Typical operating practice is to crush in closed circuit with screens to provide a top size F80 of 15,000 to 20,000 µm (1 to 1.25 in.). Deviations outside of this have often been associated with extremely poor efficiency, as measured by the Bond operating work index, especially on ores with high Bond laboratory work indices. This supports Bond's oversize feed inefficiency factor (see Table 3-2), as well as an additional overall 20% inefficiency factor introduced by Rowland for open (versus closed) circuit crushing (1982).

Table 3-2. Inefficiency Allowances for Oversize Feed (Rowland, 1982)

Bond Lab. W.I.	Feed F80: 10 mm	15 mm	20 mm
10	-	-	2%
15	-	-	18%
20	-	14%	48%

Data from Tennessee Copper (Myers and Lewis, 1943) show no change in efficiency when bin segregation resulted in a change from zero to almost 10 percent plus 25 mm (one inch) material (F80 from 11,000 to 21,000 μm), while a decrease in feed size from F80 equal to 13,000 down to 10,000 μm showed a reduction of about 20% in operating work index at National Lead (Strohl and Schwellenbach, 1950). At Brunswick Mining an increase in feed size F80 from about 12 to 14 mm actually resulted in a finer circuit product size (del Villar, 1985) and 10 to 15 percent lower operating work index.

Although ore grindability was not monitored in the above plant tests, one can generally conclude that there is a critical feed size above which rod milling efficiency is lost, most drastically when the size of feed is not controlled by screening. This critical size is smaller for harder ores. When feed size is below this critical size, variations in feed size can have varied, and possibly substantial, effects on circuit efficiency.

The presence of "fines" in open circuit rod mill feed has long been recognized as an important factor for both stable and efficient mill operation. Although addition of extra fines to rod mill feed was common practice in the past, not many details have been reported. Experience at Frood Stobie is one exception (Zickar et al, 1981). Although initially (and successfully) introduced to reduce rod tangling, the testwork showed an average reduction of

15% in operating work index over a wide range of feed rates. Note that pre-washing of crusher plant feed meant that normal rod mill feed had been low (approximately 5% minus 75 um) in natural fines content.

Work by Klimpel (1982-83) showed that the addition of fines reduced the percent solids at which production of new minus 500 um material was a maximum. Although overall efficiency was the same as at a higher density without fines addition, it could be concluded that crushing operations that produced few fines, whether because of the breakage characteristics or extreme coarseness of the feed, could be very detrimental to rod mill performance (Klimpel, 1988).

It is apparent that the presence of fines has a strong influence on material transport, and therefore the spatial distribution of solids, inside the mill. They may help mill efficiency in a number of ways;

a) by helping to carry larger ore pieces into the spaces between the rods at the mill feed end;
b) once these pieces are broken, by carrying intermediate and smaller size prices along the mill length so they can be effectively re-broken.

Note that without a) and b), a bottleneck at the feed end could control production rate, and in the extreme, result in rod tangles due to excessive spread of the charge at the feed end.

c) Near the discharge end, they provide high solids loading of fine material, so that grinding can also take place in this region. The lack of generation of fines in the mill itself may mean that there is not enough naturally produced material to take advantage of this finer grinding region.

d) Overall pulp density may be increased throughout the mill, increasing the potential for work done.

General observations, besides successful adoption of the practice at Frood-Stobie and other plants strongly support fines addition, for example;

a) The unique position of the rod mill as the first stage of material handling in which gravity is abondoned and a solids/water mixture is adopted would lead one to expect material transport problems similar to those experienced in pumping such materials.

b) The function of external classifiers, which is to lessen the net amount of fines in mill feed, and their general failure to improve the performance of coarse grinding rod mills, supports the contention that rod mills may generally function with less than an optimal amount of fines in the feed.

c) The need for a certain minimum fines content to ensure effective material transport has also been observed in ball milling (Klimpel, 1982-83).

With the historical trend towards larger, more heavily loaded, coarser grinding units, fines addition to rod mill feed offers extremely interesting potential for grinding performance improvement.

C. Practical Considerations

Excessively coarse feed may result in excessive wear on both the rods and shell liners at the feed end, and shell liners at the discharge end (Strohl and Schwellenbach, 1950). This was not noted at Sullivan, where the high mill loading and speed allowed coarse particles to pass through the mill (Banks, 1953).

Fines removal can contribute to rod tangles (Zickar et al, 1981). Fines addition, on the other hand, can be used to reduce noise level, rod and liner wear and breakage, liner bolt loosening, and generally stabilize mill performance.

While fines washing can be advantageous to crusher plant operation, it is to the detriment of rod mill operation, and can create a second problem of unstable plant operation if they are re-introduced on an intermittent basis.

Product Size

A. Power Draw

There is no observable correlation between mill power draw and product size.

B. Power Efficiency

When feed size is given, product size and reduction ratio reduce to the same basic parameter. In addition, for a given feed size and ore grindability, variations in product size (or reduction ratio) are basically related to variations in feed rate. The three may therefore be considered collectively.

The general observation that overflow rod mills operate most efficiently as coarse, rather than fine grinding units, is supported by the historical trend in their application, pilot plant results, and operating plant observations.

Operating data taken previous to 1950 (Taggart, 1945; Myers et al, 1947) are compared with data taken since then (Convey, 1957; Pickett, 1978; Rowland, 1972) below.

Table 3-3. Rod Mill Feed and Product Sizes

	Before 1950	After 1950
Typical Feed Size (F80)	5,000-25,000 µm (0.2" to 1")	16,000-26,000 µm (0.6" - 1")
Average Feed Size (F80)	12,000 µm (0.5")	20,000 um (0.75")
Typical Product Size (P80)	300-1,300 µm (48-14 mesh)	600-2,000 µm (28-8 mesh)
Average Product Size (P80)	600 µm (28 mesh)	1,400 µm (14 mesh)
Average Reduction Ratio	20/1	14/1

Both average feed and product sizes have increased considerably, with more greatly increased product size yielding an overall decrease in reduction ratio from approximately 20 to 14.

The unsuitability of rod mills for fine grinding is apparent from the large media size, and a uniquely high reduction ratio in one pass compared to other types of impact breakage comminution equipment, whether crushers, or closed circuit ball mills (Lynch, 1977). Note that Bond (1961) had no correction factor specifically for rod mill product fineness, but stated that rod mills became inefficient when operated outside the reduction ratio range of about 12 to 20. This is supported by operating data from Tennessee Copper (Myers and Lewis, 1946), where reducing the feed rate from 75 to 47 tons per hour increased the reduction ratio from 22 to 27, and the operating work index increased 40 percent. At Reserve Mining (Myers et al, 1947), reducing the feed rate from 4.4 to 2.1 tons per hour in a 0.91 m by 1.83 m (3 ft. x 6 ft.) mill correspondended to an increase in reduction ratio from 10 to 33, and an increase in operating work index of about 15 percent.

On the opposite end of the scale, at Frood-Stobie (Zickar et al 1981), an increase in feed rate reduced the reduction ratio from 15 to 8, and the operating work index increased by 40 percent. Work by Rowland (1972) also confirms inefficiency for low reduction ratios, especially for large mills with high loading rates (feed tonnage per unit of mill volume). It is noteworthy that pilot plant (overflow) rod mill tests (Myers et al, 1947) show a continuous increase in efficiency down to the lowest reduction ratio tested of 8. This suggests the limit of low reduction ratio (or high feed rate) may be approached with increasing feed per unit volume loading.

Numerous complexities make it difficult to predict results in any specific installation, even from a great deal of general operating data. However, there is strong evidence, both operational and rational, that either high or low extremes of reduction ratio are detrimental to rod milling efficiency. Fine grinding in a machine designed to crush coarse feed presents an obvious conflict. At the opposite extreme, the physical loading limitations and material transport difficulties of coarse solids in slurry form are undeniably limitations for efficient mill operation.

Slurry Density

A. Power Draw

There is no observable correlation between mill power draw and slurry density over the normal operating range. However, extremely high densities may result in reduced mill power draw (Klimpel, 1982-83).

B. Power Efficiency

Extremes in plant practice vary from about 65% solids by weight (Myers and Lewis, 1943; Tuck, 1938) or about 35% by volume, to over 80% by weight, approaching broken solids packing density of approximately 60% by volume. The latter represents the approximate limit to maintain water in the voids, and therefore approaches the limit of "wet" versus "sticky" grinding conditions. The higher density values are the most common, and have been recommended (Bond, 1953) on the basis of the maximum amount of work likely to be achieved with maximum amount of solids in the charge volume.

However, pilot plant testwork (Mitchell et al, 1954, 1955) showed that water addition rate had only a minor effect on grinding efficiency over a range of 50 to 80% solids, and only one detailed report of significant variations in rod milling performance at varying density was found (Klimpel, 1982-83). Klimpel showed that the maximum production rate of new minus 500 µm (35 mesh) material occurred at a density of 80 to 82 percent solids, in which range the mill discharge rheological character was described as pseudoplastic. At higher percent solids, the slurry displayed a yield stress, and production dropped off rapidly (note that it was also in this region that a drop in power draw was observed). At lower percent solids, the slurry character became dilatant and production fell gradually with a continuing decrease in percent solids. Maximum grinding efficiency was thus associated with optimal rheological conditions for rapid particle breakage, results similar to those found in ball milling. It was also reported by Klimpel that water control produced only marginal effects in other plants where the rod mill feed size was so coarse that the pseudoplastic region was not achievable at any density, once again implicating the potential advantage of fines addition.

Lack of an appreciable slurry feed percent solids effect in other rod mills could also be attributable to:

a) The liquid medium, with the absence of fines, is not a good vehicle for transport of coarse particles. The effect of higher flow rate may be offset by the correspondingly quicker removal of the needed fines.

b) The internal mill density may vary less than the feed (or discharge) density. Tests by Davis (1945) in a 0.91 m by 0.91 m (3 ft. x 3 ft.) ball mill showed that the average slurry density inside the mill could be appreciably higher than the feed (in one case, 72% solids versus 40% solids by weight), depending on the coarseness and density of the ore.

Owing to the simplicity with which water addition rate to the rod mill can usually be adjusted in the plant, tests should be performed to gain any advantage that can be had.

C. Practical Considerations

High mill operating densities provide operating advantages similar to those obtained by fines addition, including reduced noise and wear, and possibly breakage of rods and liners (Taggart, 1945).

Closed Circuit Grinding

A. Power Draw

No effect has been reported.

B. Power Efficiency

Historically, fine grinding rod mills were operated in closed circuit with classifiers (screens, bowls, rake and screw classifiers) and gained the same advantages as fine grinding ball mills. Operating experience at a modern rod mill installation (Basnardo, 1984) clearly shows the advantages of effective classification (both by size, in hydrocyclones, and by magnetic separation) when grinding in closed circuit with a rod mill to a final product size of 80% minus 300 um (48 mesh), although a comparison with fully open circuit operation is not mentioned.

For normal coarse grinding (or "fine crushing") applications, however, the use of external classifiers has been almost totally abandoned. As summarized by Myers and Lewis, describing testwork with a hydraulic classifier with the rod mill at Tennessee Copper (Myers and Lewis, 1946), "For more than two years, off and on, this classifier was operated with the mill, under all conceivable adjustments. In no way could it be said to benefit this fine crushing operation, and it was removed from the flowsheet."

The operating data from Tennessee Copper show that the effect of introducing a classifier into the circuit has the same direct effect as in ball milling and that is it reduces the fines content of the feed into the mill. Whereas ball mills may contain too many fines for most efficient circuit operation, the reverse may be generally true for rod mills if fines removal is never found to be an aid to better performance. Therefore, from the perspective of a reverse classification effect, fines addition as a means to improve mill performance once again appears very promising.

Media Utilization

A. Power Draw

The general relationship between mill power draw and charge level is as shown in Figure 3-2. As previously discussed, this curve should be established for each particular mill, taking into account the liner condition.

As pointed out by Bond (1960) and Rowland (1972), the overall density of the charge can be reduced by the presence of broken rod pieces, or other scrap, and this tendency will be increased with use of larger mills and larger rod sizes. Reduced breakage and amount of scrap can explain the increase in power draw (approximately 5%) observed at Tennessee Copper when a change from 76 mm to 64 mm rods was carried out, along with the possibility of higher lifting action on rods of smaller size. It is interesting that Sullivan (Banks, 1953) reverted back to 89 mm rods after trying larger ones because of the larger size and amount of broken rejects (scrap loss).

B. Power Efficiency

Because of loss of line contact between the rods, any factors which tend to increase the amount of broken rod pieces in the charge will also tend to decrease operating efficiency. In cases reviewed by Bond (1960) and Rowland (1972), associated losses in efficiency were stated to be as high as 20 to 40 percent. Bond claimed such loss could be greatly reduced by frequent charge culling. At Tennessee Copper (Myers and Lewis, 1946), only a slight improvement in performance was observed when weekly removal of all minus 19 mm rods and scrap was implemented, and the net gain was not sufficient to justify the lost operating time. Note, however, that their 2.74 metre (9 ft.) diameter mills were provided with larger than normal trunnion openings, so that accumulation of scrap may have been minimal. No change in mill power draw was reported during the test.

Bond (1950) made recommendations for the correct size of rod to be used, stating that oversize rods decrease the available contact surface area of the charge, and that undersize rods may fail to break the largest pieces in the feed. However, there appears to be only slight correlation between rod size used and feed size in plants (Convey, 1957; Pickett, 1978), indicating that the general preference for larger rods is more likely related to reduced media consumption, rather than milling efficiency.

Unfortunately, plant studies on media size versus rod mill performance are very rare. The single case that was found was once again from Tennessee Copper (Myers and Lewis, 1946). A reduction in rod make-up size from 76 to 64 mm (3 to 2 1/2 in.) resulted in;

a) a 5.4% increase in power draw;

b) a 7.8% increase in work done, as measured by new surface area;

c) a 22.2% increase in rod consumption; and

e) a reduction in the maximum feed particle size that could be broken, from about 25 mm to 19 mm.

The increase in efficiency, as calculated either by new surface area per unit power, or as indicated by the reduction in the operating work index, was only 2 to 3 percent. However, the reported feed size was 54% plus 6.73 mm with the 64 mm rods, and only 40% plus 6.73 mm with the 76 mm rods. This substantially coarse feed may have had a negative effect on mill efficiency, as indicated by Bond's reduction ratio correction factor. The reduction ratio was less than 30 with the 76 mm rods, but over 40 with the 64 mm rods.

Myers (1953) also experimented with new rods with a non-cylindrical shape (oval or flat in cross section, rather than round), to intensify the sliding action. The results on grinding performance were reported as "disastrous" and the test was quickly abandoned.

C. Practical Considerations

Any aspect of media utilization which affects its consumption rate is extremely important because it is such a large part of total grinding costs (Zieman, 1959).

As shown in Figure 3-2, the curve relating increase in power draw with increase in charge level becomes very flat as the 50% charge level is approached. Because media wear will increase roughly in proportion with charge volume, and power draw is increasing only very slowly, it is not economically favorable to operate close to this peak. This also suggests the advantage of high mill speed, rather than high charge level, to achieve desired power draw because the same power draw can be achieved with substantially less steel load in the mill. For example a 3.66 m diameter by 4.88 m long (12 by 16 foot) rod mill will draw 763 kw (1023 HP) under either of the following conditions.

	Condition A	Condition B
Critical speed (%)	66.4	75.0
Charge level (%)	45	35
Rod load weight (tonnes)	112	83
Power draw (kw)	763	763

Experience at Tennessee Copper (Myers and Lewis, 1943) provides strong evidence that increased mill speed leads to reduced rod consumption, as shown in Figure 3-3. As a

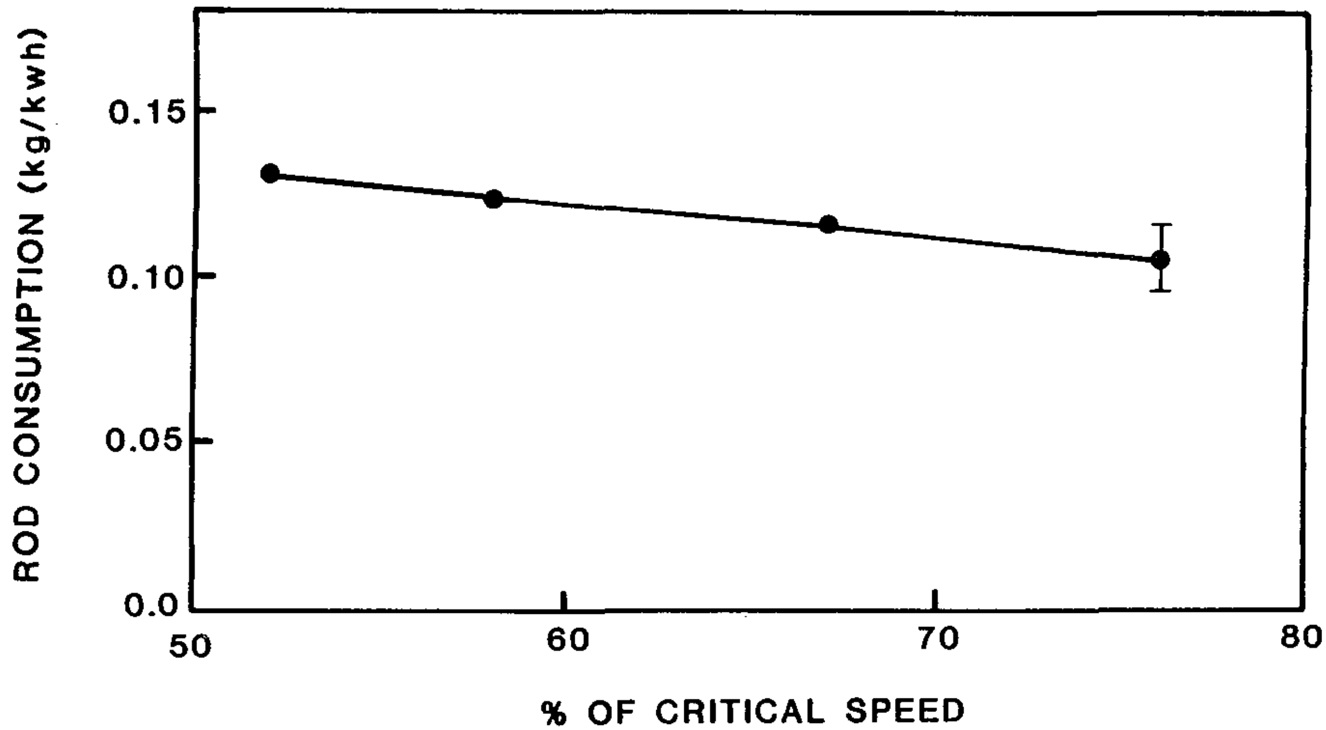


Figure 3-3. Rod Consumption vs. Mill Speed at Tennessee Copper.

further test of this concept, the rod consumption versus mill critical speed for all the plants for which data has been collected was plotted and compared, as shown in Figure 3-4. The best-fit regression line shows an average decrease in media consumption (per unit power draw) of 15 percent, for an increase in percent of critical speed of 10 points. This regression line was found to have a negative slope (that is, showing reduced media wear with increased mill speed) to a confidence level exceeding 99.9%.

Tennessee Copper also report reduced media consumption with higher feed rates (Lewis and Goodman, 1957) to the rod mill.

At Frood-Stobie (Zickar et al, 1981), use of larger rods and increasing the mill speed from 66 to 74% of Cs greatly reduced the number of rod tangles.

At Sullivan, the mill speed of 81% of Cs can probably be considered as excessive compared to "normal" rod milling practice, as it was selected (and successfully operated) on the basis of needing a well spread charge to accept the very large feed at high tonnage, and ultimately, passage of some oversize material in the discharge had to be tolerated (Banks, 1953; Convey, 1957). Despite having an extremely abrasive ore, their media consumption was unusually low. It was also noted that the shape of broken rod rejects was round, and not the typical oval shape. These facts were attributed to the lack of sliding action, and possible work hardening of the rods, as a result of the high mill speed.

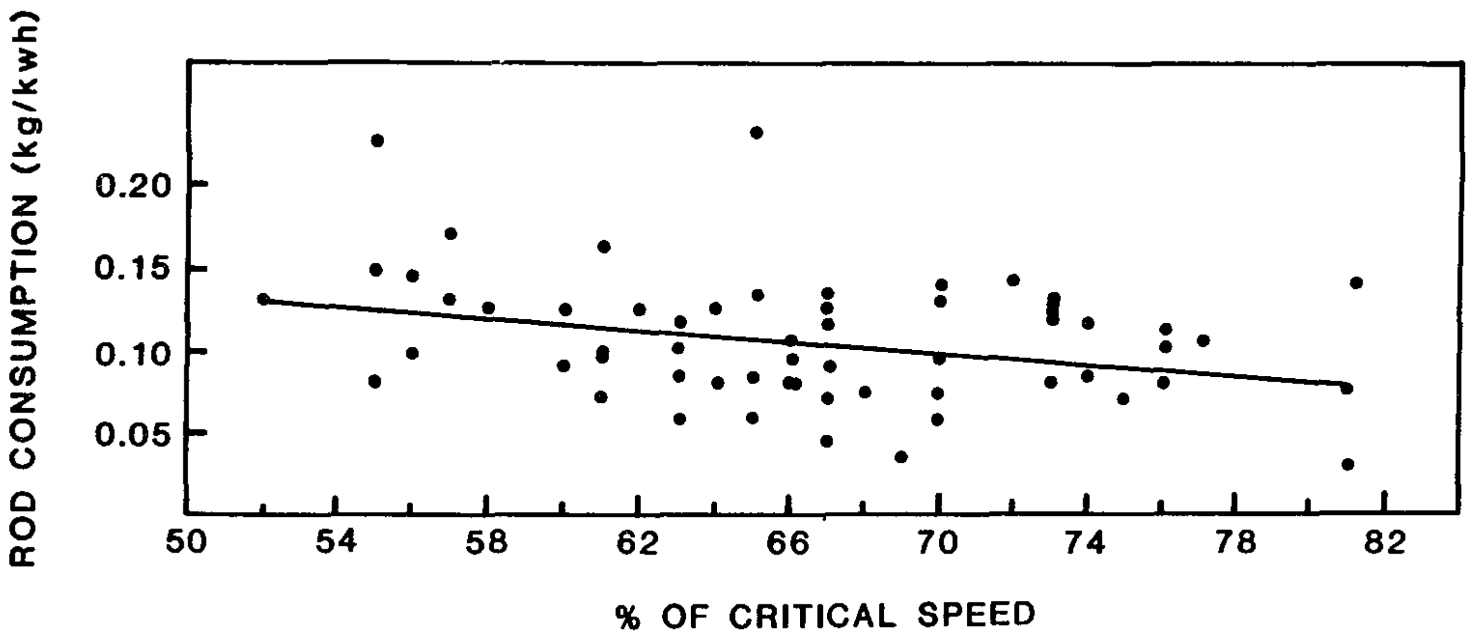


Figure 3-4. Rod Consumption vs. Mill Speed (62 Installations, 69 Data Points).

As previously noted, decreasing the rod size from 76 to 64 mm at Tennessee Copper (Myers and Lewis, 1946) resulted in an increase of 22% in rod consumption, which can be directly attributed to the larger exposed surface area of the grinding media. This strongly supports the general practice of selecting large rods, but should be considered in conjunction with mill efficiency. Rods larger than 89 mm diameter caused the size and number of broken rods to increase at Sullivan (Banks, 1953), which would not only adversely affect power draw and mill efficiency, but also increase the scrap loss component of media consumption.

Most operations report that grinding rods are added to the mill two or three times a week. Depending on the media consumption rate, this could cause a wide fluctuation in mill power draw, and operation at least part of the time near the peak of the power versus charge volume curve. Frequent rod additions should therefore be considered for both grinding cost reduction and for the stabilizing effect on both the circuit and plant performance.

Finally, rod quality plays an obvious role, both from the point of view of wear resistance (Malghan, 1981), and lost power draw, grinding efficiency, and media life due to breakage.

3.1.5 Summary and Conclusions

1) The main factors which effect rod mill power draw are the mill dimensions, speed, and charge level. Power draw

itself can have an indirect effect on mill efficiency, for example, by affecting a more favorable reduction ratio, but is more directly related to mill capacity or product size, as approximated by the Bond law.

2) Traditionally, rod mills have been operated at low percentages of critical speeds. Based on numerous successful operating plant experiences, there is no apparent reason why rod mills should not be operated at close to 80% of critical speed. As an alternative to high charge level for achieving desired power draw, this offers potential for reduction in media consumption.

3) There is a limit on the size of feed material which a given rod mill can accept without becoming inefficient, and this limit becomes smaller with smaller mills, smaller media sizes, lower percent of critical speeds, higher feed rate, and harder ores. Below this size, varying feed size can have varying effects on mill efficiency.

4) Fines in the rod mill feed may substantially assist in the transport and favorable distribution of coarse solids through the mill. It can be argued that virtually all rod mills, but especially larger ones with high loading rates, could benefit from additional fines in the feed.

5) Assuming fixed feed size, then product size, reduction ratio, and feed rate can be considered as the same basic parameter. There is a general range of product size, corresponding approximately to a reduction ratio of 10 to 20 to one, outside of which rod milling has often been observed to be inefficient.

6) When rod mill feed size is very coarse, slurry percent solids appears to have little effect on rod mill efficiency. When the feed is fine enough so that the mill slurry rheological character is sensitive to percent solids, water addition rate can have a significant effect on grinding efficiency. However, owing to the ease of water addition rate adjustments, it should always be included in plant rod mill circuit evaluation.

7) When the capability to break the coarsest feed particles rapidly enough is not encroached, a reduction in rod size will improve grinding efficiency. However, media wear will also increase because of the increased surface area of the charge. Whether such a move will be of overall benefit will depend on the specifics of each installation, including such factors as rod make-up size, scrap size, feed material size, apparent mill efficiency, and relative power and media costs.

3.2 BOND WORK INDEX ANALYSIS OF ROD MILLING

3.2.1 Introduction

As discussed in chapter 1, the effect of design and operating variables on open circuit rod milling can be assessed by comparing circuit work index efficiency, defined as the ratio of the Bond laboratory test work index to the plant operating work index, before and after a change to the circuit. When the change is made to one of a number of otherwise similar parallel circuits, a series of relative work index measurements taken over an extended period of operation can be used to verify the effect and quantify the average value of an indicated efficiency improvement. This approach was demonstrated by Rowland (1976). Another approach is to carry out a series of repeated measurements on the same circuit while the variable is alternatively changed in a progressive or "on-off" fashion. The fines addition tests at Frood-Stobie (Zickar et al, 1981) were of this nature, although operating work index variations alone were measured while maintaining a consistently controlled feed material. The purpose of repeat testing in either of the above scenarios is to generate a number of readings or measurements so that a "best estimate" of the true value of the change in performance can be calculated by averaging, with the corresponding statistical estimate of its reliability.

The purpose of investigating the reliability (or conversely, the error) associated with single work index efficiency determinations is two-fold. First, when individual data are compared with one another, only those changes which exceed the normal variations associated with the measurements are significant. Thus, it is important to know the quantitative value of the error. Secondly, when the sources of error are identified, it may be possible to eliminate or reduce some of them so that the total error is minimized. When the accuracy of the efficiency measurement is improved, smaller changes in efficiency can be detected and the need for repeat test work reduced.

3.2.2 Sources of Error

When the Bond methodology for estimation of rod or ball mill power draw requirements is used for the design of a new grinding plant, a number of sources of inaccuracy come into play, as follows:

1. Test sample representativeness;
2. Scale-up to plant size equipment from the 0.3 meter (12 inch) diameter test mill, including the use of specified correction factors for different design conditions;
3. Other non-specified factors which influence circuit performance, including classification characteristics, media sizing and condition, degree of instrumentation and control, slurry densities, mill liner condition, and circuit arrangement;
4. Estimated power draw of selected mill(s).

5. Variations in the feed and product sizing (and product composition for heterogeneous materials) between the laboratory test and plant conditions.
6. Experimental error in the test procedure and measurements.

For a new operation, samples may be obtained from exploration drill core or mine development headings. As well, relatively short and/or long term ore variations will be expected, depending on the orebody and the mining plan. These factors necessitate considerable flexibility (in power draw capability through charge level or speed adjustments) and conservatism in the mill and motor selection.

For operating plant evaluations, samples are obtained from the feed to the circuit, greatly diminishing the problem of sample representativeness. The mill power draw can also be measured, rather than estimated. The efficiency ratings will also be determined by actual feed and product size and feed rate measurements in the plant circuit.

A study by Rowland (1973), comparing the laboratory work index (of samples taken from operating circuits) with the corrected operating work index indicates that the total error involved in the Bond mill selection procedure for ball mills seldom exceeds approximately 20 percent (with 95 percent confidence limits). That is not to say that component errors may not be substantial, as they may be partly cancelled out in arriving at the total error. As well, there may be a tendency for operators to adjust circuit operation at start-up until performance comes into line with design figures. Nevertheless, the correlation between actual and predicted performance is reasonable considering the simplicity and economy of the design method. The same degree of success cannot be claimed for rod mills, with variations of 30% attributed without certainty to the presence of broken rods, or constriction of high pulp flows. Rowland's suggestion that relative rod milling inefficiency will normally be compensated by higher ball milling efficiency refers to the primary use of the Bond methodology for new circuit design, rather than existing circuit evaluation.

The following sections examine more closely two important factors in rod mill circuit efficiency evaluation using the Bond approach, namely, grinding mill power draw and Bond laboratory rod mill work index determinations.

3.2.3. Grinding Mill Power Draw Determinations

3.2.3.1 Introduction

Electrical power consumption figures are of interest for the grinding circuit evaluation for two reasons:

1. Electrical input power consumption is one of the major cost factors in grinding;
2. The output power at the mill pinionshaft is used in operating work index determinations, which in turn are used to assess overall circuit efficiency.

The purpose of this description is to outline the details required for accurate calculation of the power draw of grinding mills.

The reader is referred to general publications on motor power draw measurement (such as Pumphrey, 1959, or Janicke, 1980). The electrical input power for synchronous or induction motors may be read directly from a wattmeter, if one is provided. Alternatively, if motor input power has been measured or calculated, and motor efficiency under the given operating conditions is known, the output power can be calculated directly by the relation:

$$\text{Output Power} = \text{Input Power} \times \text{Efficiency}$$

Because grinding calculations by convention refer to mill power draw at the pinion driving the main gear, this discussion is concerned with the calculation of motor output power and mill drive losses up to the pinion.

There are basically two sources of information for motor power draw calculations, namely:

a) the electrical instruments provided; and,

b) the motor performance data on the nameplate and/or provided by the manufacturer.

3.2.3.2 Synchronous Motor Output Power

The most economical design of high power mill drives is usually with a low speed motor directly coupled to the pinion, as opposed to higher speed motors with gear reduction units on small (say, less than 300 kw) drives. The motors for the rod mills and primary and secondary ball mills at Kidd Creek are of the synchronous type.

A major advantage of synchronous motors is that they operate with a leading power factor. This provides a means to improve the overall plant power usage, since all induction motors operate with lagging power factor.

Two general methods of calculating motor output power are available. The first uses a direct measure of motor input power from a wattmeter, while the second relies on calculation of the motor input power from other instrument readings.

<u>Method 1</u>

Output Power = Input Power x Efficiency

The wattmeter reading normally provided in the control room instrument panel is a direct measurement of the electrical input power to the motor. The only additional information needed to calculate the output power is the motor efficiency under the given operating conditions.

If full operating tests were run on the motor, the manufacturer or motor file could provide the efficiency figure needed above. In most cases, no tests of this nature are performed. However, the efficiency at full load is normally either given or can be calculated from the nameplate data, and hence the efficiency at reduced load estimated.

In most cases, it is possible to calculate the full load efficiency from the nameplate data using the basic formula (Lessard, 1981):

$$\text{Eff. (at full load)} = \frac{\text{Rated Output Power} \times 746}{\sqrt{3} \times E \times I \times \text{p.f.}}$$

where E is voltage, I is amperage, and p.f. is power factor.

Normally, synchronous motor efficiency is constant down to 3/4 load, and loses 1/2 point in efficiency at 1/2 load. We can estimate the fractional loading from the ammeter reading in the motor control room compared to the full load amps of the motor, and make this correction, if necessary. Synchronous motor efficiency also varies slightly with line voltage and power factor, but the effect is negligible within 10% variations from the design ratings (Lessard, 1981).

Example - Rod mill power draw from readings taken July 25, 1985, at Kidd Creek Mines.

The wattmeter in central control was fluctuating widely, between 450 and 550 kw, with a slightly greater tendency to stay towards the high side. Using the average of a number of readings it was estimated at 510 kw.

Nameplate data:	800	H.P.
	115	amps
	4000	volts
	0.8	p.f.

From the above equation, the motor efficiency at full load is:

$$\text{Eff. (full load)} = \frac{800 \times 746}{1.73 \times 4000 \times 115 \times 0.8} = 93.7\%$$

From the motor control room ammeter reading of 99 amps (average of 3 phases), the motor is operating in excess of 3/4 load, so no correction factor is needed. Therefore:

Output = 510 kw x .937 = 478 kw, or, 641 H.P.

(Conversion factor: 1 kw = 1.341 H.P.)

Method 2

For 3 phase motors;

$$\text{Output Power (kw)} = \frac{\sqrt{3} \times I \times E \times \text{Eff.} \times \text{p.f.}}{1000}$$

Example - Same as above.

From the motor control room instruments:

$$I = 99 \text{ amps}$$
$$E = 4150 \text{ volts}$$
$$\text{p.f.} = 0.73$$

Substituting in the above equation:

$$\text{Output power} = \frac{1.73 \times 99 \times 4150 \times .937 \times .73}{1000}$$

$$= 486 \text{ kw, or, } 652 \text{ H.P.}$$

The result is in good agreement with the figure of 478 kw obtained from method 1.

3.2.3.3 Induction Motor Output Power

Method 1

Output Power = Input Power x Efficiency

The wattmeter reading normally provided in the control room once again provides the value of input power. In the case of induction motors, the efficiency at full load may once again be given, or can be calculated from the motor nameplate data. Occasionally, the motor manufacturer can also provide load versus efficiency data from a full running test.

Induction motor efficiency and power factor also vary with fluctuations in line voltage. The correction factors which should be applied can be taken from Table 3-4 (Lessard, 1981).

Table 3-4. Variation in Induction Motor Efficiency and Power Factor with Line Voltage (Approximate)

Percent Nominal	Efficiency Variation (%)			Power Factor Variation (%)		
Voltage	1/2 F.L.	3/4 F.L.	F.L.	1/2 F.L.	3/4 F.L.	F.L.
90	+ 1.5	0	- 2	+ 4.5	+ 2.5	+ 1
92	+ 1	0	- 1.5	+ 3.5	+ 2	+ 1
94	+ 1	0	- 1	+ 3	+ 1.5	+ 1
96	+ .5	0	- .5	+ 2	+ 1	+ .5
98	+ .5	0	0	+ 1	+ .5	0
100	0	0	0	0	0	0
102	- .5	0	0	- 1	- .5	- .5
104	- .5	0	+ .5	- 2	- 1.5	- 1
106	- 1	0	+ .5	- 3	- 2	- 1.5
108	- 1	0	+ .5	- 4	- 3	- 2
110	- 1.5	0	+ 1	- 5.5	- 4	- 3

Actual motor speed also varies about 1% for a 10% change in line voltage.

Example - Copper regrind mill power draw taken from readings at Kidd Creek Mines June 27, 1985.

The wattmeter reading in the central control room was 218 kw. The following performance test information (at design voltage) was given by the motor manufacturer (compliments of Brown Boveri Corp. in Montreal).

Table 3-5. Kidd Creek Regrind Mill Motor Performance Test Results

H.P.	Eff. (%)	P.F. (lagging)
436.8	93.1	.883
349.5	93.1	.866
347.7 (full load)	93.1	.868
265	92.7	.841
176	90.9	.734
86.9	84.9	.535

From the motor control room ammeter reading of 37 amps (average of 3 phases), and the motor nameplate full load current of 47 amps, the motor is running at approximately 79% of full load, or 275 H.P. From the above figures, the efficiency is close to 92.7% at the rated motor voltage.

Method 2

For 3 phase motors;

$$\text{Output Power (kw)} = \frac{\sqrt{3} \times I \times E \times \text{Eff.} \times \text{p.f.}}{1000}$$

Example - same as above.

Normally, power factor is neither given nor measured for small induction motors. Assuming it was not given in this case, we could still calculate the approximate output power, near full load conditions, as follows.

From the nameplate motor data: 350 H.P.
47 amps
4000 volts

$$\text{Eff. x p.f. (full load)} = \frac{350 \times 746}{1.73 \times 4000 \times 47}$$
$$= .803$$

Assuming we are near enough to full load so that the term (Eff. x p.f.) has not changed substantially (although we know it may have decreased slightly), substituting back into the above equation, and using the motor control room readings of 37 amps and 4150 volts:

$$\text{Output power (kw) (Approx.)} = \frac{1.73 \times 37 \times 4150 \times (.803)}{1000}$$
$$= 213 \text{ kw, or, } 286 \text{ H.P. (Approx.)}$$

However, in this case, we can use the information provided by the motor manufacturer. From Table 3-4, we see that no correction for efficiency and a 1.5 point reduction in power factor is needed to adjust for the line voltage.

Then, substituting;

$$\text{Output Power (kw)} = \frac{1.73 \times 37 \times 4150 \times .927 \times (0.841-0.015)}{1000}$$
$$= 204 \text{ kw, or, } 274 \text{ H.P.}$$

3.2.3.4 Power Draw at the Pinion Shaft

Losses in commercial size grinding mill main bearings with hydrodynamic lubrication are very small, in the order of one percent. In any case, since we are interested in the power at the mill pinion shaft, losses due to the mill bearings, windage, and at the gear and pinion mesh will be included in one figure for "mill power draw". Assuming this is constant for all mills means power draw figures may be directly compared with one another.

Losses in the drive components between the motor and the pinion are negligible except for reduction gear units, which vary from 1 to 2 percent per mesh, depending on the condition of the gears. We can therefore assume negligible losses in the synchronous motor mill drives, and 1.5 percent total loss in the regrind mill drives which each have a single reduction gear unit.

A summary of the motor data, instrument readings at Kidd Creek on June 27, 1985, and calculated power draw figures is given in Table 3-6. The discrepancy between calculated power draw using the two methods for the primary ball mill and zinc regrind mill must be attributed to instrument calibration. A calibration check was requested for all instruments prior to grinding circuit sampling.

In general, wattmeters have been found to be far less reliable than other electrical instrumentation. The detailed procedure (method 2) is therefore recommended, with wattmeter readings to be used as a check only.

Table 3-6. Mill Power Draw Measurements at Kidd Creek Mines

A. Synchronous Motors

	Rod Mill	Primary Ball Mill	Secondary Ball Mill
Motor Data:			
Speed (r.p.m.)	225	225	225
Rating (H.P.)	800	1500	1500
Service Factor	1.15	1.15	1.15
Nominal Voltage	4000	4000	4000
Full Load Current (amps)	115	214	214
Nominal Power Factor	0.8	0.8	0.8
Full Load Efficiency	93.7%	94.5%	94.5%
Instrument Readings (June 27/85):			
Wattmeter (kw)	510	870	878
Voltmeter (volts)	4150	4150	4150
Ammeter (average)	99	158	153
Power Factor Meter	0.73	0.83	0.81
Calculated Motor Output (kw) and Power at Pinion Shaft:			
Method 1.	478	823	830
Method 2.	486	890	840
Difference	1.7%	8.2%	1.3%

B. Induction Motors

	Copper Regrind Mill	Zinc Regrind Mill
Motor Data:		
Full Load Speed (r.p.m.), full load	1187	1187
Rating (H.P.)	350	350
Service Factor	1.15	1.15
Nominal Voltage	4000	4000
Full Load Current (amps)	47	47
Full Load Efficiency	93.1%	93.1%
Instrument Readings (June 27/85):		
Wattmeter (kw)	218	203
Voltmeter (volts)	4150	4160
Ammeter (average)	37	38
Calculated Motor Output (kw):		
Method 1.	202	188
Method 2.	204	210
Power at Pinion Shaft (assumes 98.5% drive efficiency) (kw):		
Method 1.	199	185
Method 2.	201	207
Difference	1.1%	12.1%

3.2.3.5 Uncertainty of Power Draw Estimates

A general study of power draw determinations for grinding mills has determined that the non-negligible sources of error are due to uncertainty of instrument calibration and uncertainty of instrument readings (Lessard, 1981). Calibration error for ammeters and voltmeters can vary widely, from 0 to 5%, depending on instrument maintenance practices. However, over a short period of time during which the grinding circuit is evaluated relative variations in instrument performance can be considered negligible. This means that the relative calibration error in a given installation is small, but that the absolute difference between mills and plants can be significant.

Instrument reading uncertainty is estimated as 1/2 of the smallest graduation (or part of graduation, when the spacing is large) that can be readily read from the scale. An oscillating or changing digital reading greatly increases the error, but these can be largely negated by taking numerous readings and averaging them over time. Thus, relative instrument readings are normally repeatable to an accuracy of about plus or minus 1 percent.

When the reading errors of multiple instrument readings are combined with the other sources of error associated with the use of the power draw formula described by method 1 above, the total relative error in calculation of motor output power is normally less that 2% (Lessard, 1981). The uncertainty associated with efficiency losses in the drive train to the mill pinion would reasonably mean a total error of no more than 3% in the mill power draw estimations.

3.2.4 Error Analysis for the Bond Laboratory Rod Mill Work Index

3.2.4.1 Introduction

The purpose of carrying out error analysis is to determine a suitable confidence interval for any numerical value. For the Bond rod mill work index test, this is a requirement for carrying out meaningful comparisons between test work index values obtained for different samples, as well as for comparing test laboratory and plant operating work index values.

The total error can be attributed to sampling error plus experimental error, such that the total variance,

$$S_T^2 = Ss^2 + S_E^2$$

where Ss^2 = variance due to sampling error

S_E^2 = variance due to experimental (preparation and analysis) error.

The laboratory test procedures, as issued by Allis-Chalmers Corporation, are included in Appendix D, along with supplementary notes on the procedures employed at McGill University. The example used is from the Bond rod mill work index test carried out on ore sampled from the rod mill feed during sampling survey no. 2 at Les Mines Selbaie. A summary of this test is given in Appendix E. Similar results would be applicable to any value obtained from a test on a properly extracted sample. The pertinent data from this test were as follows:

Primary sample size: 18 kg (approx.)
5% retained on 1.7 cm

Ore specific gravity: 2.8

Rod Mill Work Index Test
P_1 = screen closing size; 10 mesh, or 1700 µm
Grp = grams per revolution, last 3 runs; 13.3
F = test feed size (80% passing); 10,160 µm
P = test product size (80% passing); 1,310 µm
W.I. = 13.8 kwh/m.t.

Eighty percent passing sizes were calculated by log-log interpolation of the size distributions from the test.

3.2.4.2 Sampling Error

For a properly extracted sample, Gy's (1982) formula for the variance of fundamental error can be taken as a conservative estimate for the total sampling error.

$$S^2\ (FE) = c\ l\ f\ g\ d^3 \left[\frac{1}{M_s} - \frac{1}{M_l}\right]$$

c = compositon factor
l = liberation factor (normally 0.5)
f = shape factor (normally 0.5)
g = size distribution factor (normally 0.25)
d = particle size factor (5% retained, in cm)
Ms = sample size, grams
M_l = lot size, grams

The composition factor, c, reduces to the value of the specific gravity of the ore, when the above formula is applied with respect to some basic characteristic of the ore, such as laboratory work index, 80% passing size, or S.G., rather than a widely varying value, such as mineral assay. Therefore, for the primary sample of 18 kg taken during the sampling survey:

$$\text{Primary } S^2 \text{ (FE)} = (2.8)\ (0.5)\ (0.5)\ (0.25)\ (1.7)^3 \left[\frac{1}{(18{,}000)} - 0\right]$$

$$= 4.8 \times 10^{-5}$$

The actual material involved in the work index calculation is the test mill contents during the last 3 grind periods, a total of approximately 4 kg of material.

This is the secondary sample, for which:

$$\text{Secondary } S^2 \text{ (FE)} = (2.8)\ (0.5)\ (0.5)\ (0.25)\ (1.2)\left[\frac{1}{4000} - \frac{1}{18000}\right]$$

$$= 5.9 \times 10^{-5}$$

The variance of the combined sampling error is the sum of these two.

$$Ss^2 = 10.7 \times 10^{-5}$$

Solving, $Ss = 0.0103$

For the 95% confidence limits:

$$(1.96)\quad (Ss) = \pm \frac{X}{13.8}$$

Solving, $X = \pm\ 0.28$

Therefore, we can say due to sampling error, the range of the Work Index value of 13.8 is $\pm$ 0.3 kwh/t, with a confidence level of 95%.

3.2.4.3 Experimental Error

The experimental error can be estimated from the limits of error in the terms used to calculate the Work Index in the formula:

$$\text{W.I.} = 62 \Bigg/ \left[P_1^{\,0.23} \times \text{Grp}^{0.65}\left(\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}}\right)\right]$$

P_1 = closing screen size (1700 µm, or 10 mesh)
Grp = grams per revolution (13.3)
F80 = test feed size (10,160 µm), 80% passing
P80 = test product size (1,310 µm), 80% passing

Since there is little error in the value P_1, the error involved will be due to screening, weighing, and the closed circuit grinding of the ore itself. The limit error for the 95% confidence can be estimated for each of these, as follows:

a. Screening: Normal dry screening in the size ranges needed to estimate F and P is accurate to ± 0.1 percentage points on a single screen containing approximately 5 to 10 percentage points of a material size distribution (anon., 1981). This indicates a limit of error in the order of 0.1/(5 to 10), or 1 to 2%. As well, interpreting the exact location of the 80% passing point is difficult, as the coarsest screen size contains over 20% of the material in both feed and product for the test. Allowing approximately 2% error in this step yields a total limit of error for screening of $(2^2 + 2^2)^{0.5}$ = 2.8%, or say 3%. Total dust losses are very small, so may be taken as being included in this figure.

b. Weighing: Either large samples on the balance scale or smaller ones on the electronic weigh scale involve errors of much less than one-tenth of a percent. These are negligible for the purpose of total error estimation.

c. Closed-circuit laboratory grinding: The closed-circuit grind experiment involves a number of possible sources of error, including the following:

1. the number of mill revolutions,
2. classification efficiency,
3. media size or weight variations,
4. dryness of the ore,
5. segregation at mill ends,
6. dust losses.

With consistency and reasonable care practiced throughout the test, none of these can be considered significant, with the exception of the first. An error of less than one revolution is possible, or roughly 1 1/2% for 70 revolutions. A total error of $\pm$ 2% for all sources of error related to the closed circuit grinding procedure is therefore reasonable.

The variables used to calculate the Work Index and the approximate limits of error at a suitable (95%) confidence level are therefore as follows:

$$\begin{aligned} Grp &= 13.3 \pm 2\%, \text{ or } 0.27 \text{ g/rev} \\ P &= 1{,}310 \pm 3\%, \text{ or } 39\ \mu m \\ F &= 10{,}160 \pm 3\%, \text{ or } 305\ \mu m \end{aligned}$$

The rules for accumulation of error in mathematical operations are summarized as follows:

For values of A and B with limits of error of a and b respectively (i.e. A $\pm$ a, and B $\pm$ b), then, the limit or error for:

Constant x A	is	Constant x a
A + B	is	$(a^2 + b^2)^{0.5}$
A x B	is	$(B^2 \times a^2 + A^2 \times b^2)^{0.5}$
A^n	is	$n^{0.5}\ A^2 \times a$
1/A	is	a/A^2

Applying these rules to the limits of error of Grp, F80, and P80 results in the limit of error of W.I. of $\pm$ 0.52. Therefore, we can say due to experimental errors the range of the work index value of 13.8 is $\pm$ 0.5 kwh/m.t., with a confidence of 95%.

3.2.4.4 Total Error

Total error equals:

$$(0.28^2 + 0.52^2)^{0.5} = 0.59$$

The Bond rod mill work index value for this test is 13.8, $\pm$ 0.6 kwh/m.t. with a confidence level of 95%.

A more sophisticated mathematical analysis which accounts for interactions between the sources of error was carried out by Laplante et al (1988). This showed that the amount of error estimated by the above method is conservative (by a factor of about 2), and that about the same error can be expected to hold true for the ball mill work index test.

From an engineering perspective, the Bond rod mill work index test appears to be very well conceived. Sampling and experimental error are in the same order of magnitude, and the results can be taken to be statistically valid to about $\pm$ 2% with a 95% level of confidence.

3.2.4.5 Adjustment for Non-Standard Test Conditions

While the above discussion describes error associated with sampling and readings or measurements taken during the laboratory grinding test, other causes of possible variation (or bias) were investigated. These were those associated with the weight of the mill solids load during the test and new feed size distribution. A summary of the results of these and other work index tests performed during this study are given in Appendix F.

The mill load weight is determined by packing feed material into a 1250 c.c. volume. Although the same feed material was used, it was found that the mill load weight could easily vary by 150 to 200 grams, depending on the degree of packing in the cylinder or beakers. In order to determine the effect that this would have on test results, repeat tests were run on two different ore samples with substantially different mill loads in each case. The results (Table 3-7A) were virtually identical, and it was concluded that the rod mill work index test results are not sensitive to the mass of the load used during the test.

Table 3-7A. Rod Mill Work Index Test Results with Different Solids Loading

Sample	Specific Gravity	Mill Load (g)	F80 (μm)	P80 (μm)	Net g/rev.	W.I. (kwh/tonn
Selbaie 4A	2.73	2205	10,015	1,300	12.0	15.0
(same)	(same)	2337	10,015	1,326	11.9	15.0
Selbaie 4C	2.76	2160	9,675	1,330	12.4	14.9
(same)	(same)	2400	9,675	1,332	12.4	14.9

In order to test the effect of test feed fineness, a "coarse" test feed of approximately 50% minus 6 mm was tested and yielded a W.I. of 15.0 (Selbaie sample no. 4A). When the same feed was crushed to 50% minus 2.5 mm, a W.I. of 18.5 was obtained. Note that both the test feed samples met the specification of all minus 12.7 mm (Bond, 1961). Since all of this sample was depleted, a repeat test was run with a number of carefully prepared samples from Selbaie survey no. 1 with different 50% passing sizes. The same was done with samples prepared from Kidd Creek survey no. 2. The mill load was held constant in each case. The results are summarized in Table 3-7B and plotted in Figure 3-5.

Typically, rod mill feed test material prepared with a jaw crusher closed-side setting of about 13 mm (1/2 inch) has a 50% passing size of just under 6 mm. Figure 3-5 shows that no

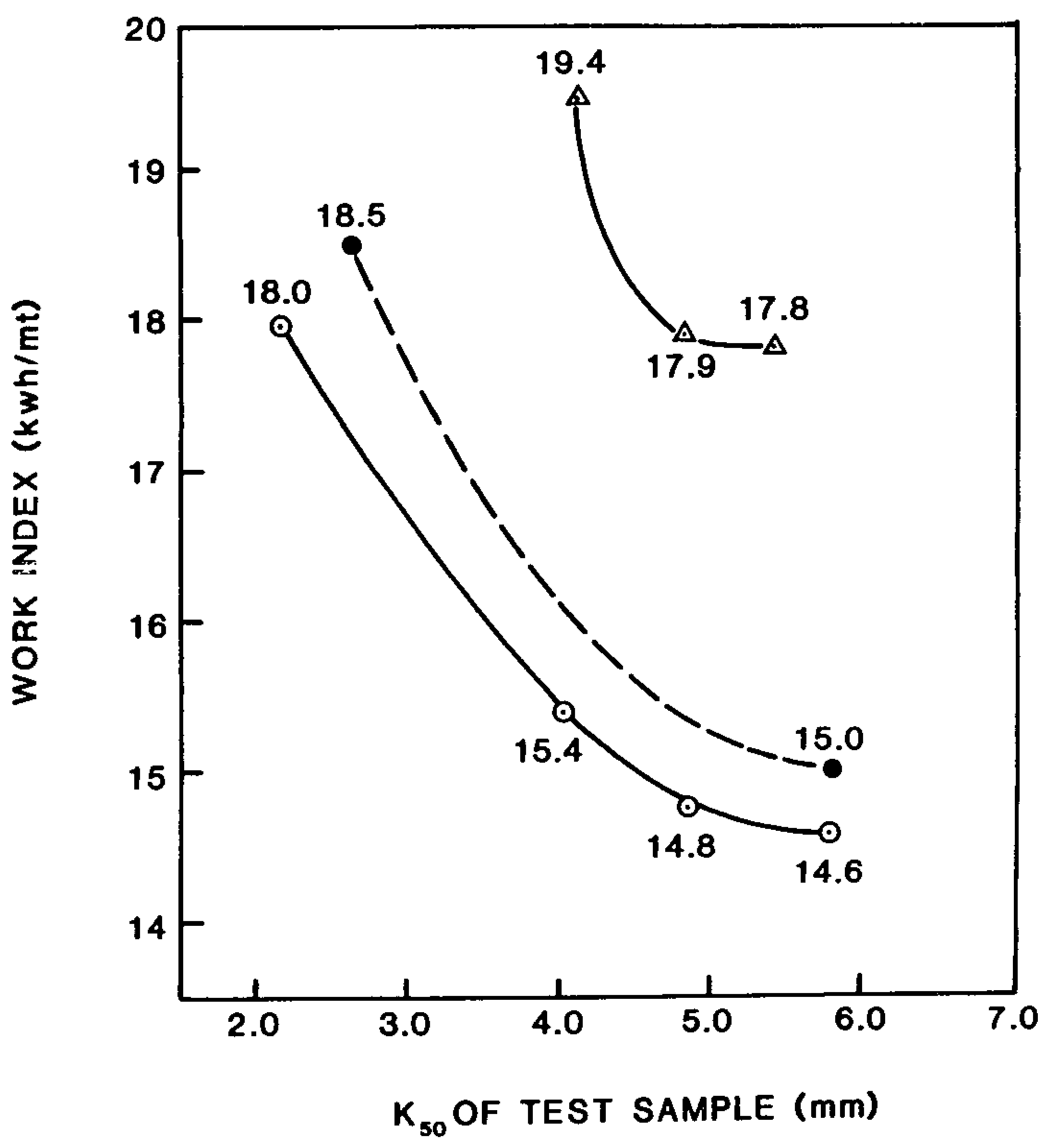


Figure 3-5. Rod Mill Test Work Index vs. K50 of Test Sample for Three Ore Samples of Different Grindabilities

Table 3-7B. Rod Mill Work Index Test Results with Different Test Feed Sizes

Sample	Feed Size (μm) F80	Feed Size (μm) K50	Product P80 (μm)	Net g/rev	W.I. (kwh/tonne)
Selbaie #4A	10,015	5,842	1,326	11.9	15.0
(same)	5,465	2,625	1,319	12.2	18.5
Selbaie #1	10,513	5,760	1,307	12.0	14.6
(same)	9,705	4,842	1,315	12.3	14.8
(same)	8,105	4,030	1,315	12.6	15.4
(same)	5,338	2,146	1,319	13.0	18.0
Kidd Creek #2	9,987	5,378	1,240	8.34	17.8
(same)	8,923	4,842	1,230	8.67	17.9
(same)	7,945	4,086	1,302	8.68	19.4

correction is necessary down to about 5.25 - 5.50 mm. At 5 mm, however, about 0.1 (or 0.7%) must be subtracted from the obtained W.I. value to correct to the "standard" test feed, with a rapidly increasing adjustment needed as it becomes finer.

It is concluded that comparative work index test values should be based on similar test feed fineness, or a suitable adjustment made accordingly, as established by grindability tests with different feed sizes. This is probably not critical for the accuracy needed for tests carried out for the purpose of sizing new grinding mills, but should definitely be considered in plant performance studies. The Bond rod mill test results determined throughout this study were therefore adjusted to a "standardized work index" value at a normal test feed size of approximately 50% minus 5.5 mm for relative circuit efficiency comparisons.

A preliminary evaluation of the same two factors (i.e. mill loading and test feed size) in Bond ball mill grindability testing was carried out on the rod mill discharge sample from survey no. 1 (see Appendix F). Although much further testwork is needed before these effects can be properly quantified, it appears that (to some degree) the mill loading and (to a significant degree) the material test feed size have an influence on the value of the Bond ball mill laboratory work index.

It should be noted that the statistical validity of work index determinations described here applies only for comparisons done at the same laboratory, with the same equipment, and ideally with the same personnel. Wyslouzil (1982) has shown disagreement between different laboratories of 33% on duplicate test samples. There are numerous sources of error due to variations in test procedures and equipment. All comparative tests should be performed at the same location for grinding circuit efficiency evaluations to have any meaning.

3.2.5 Accuracy of Relative Work Index Efficiencies

The overall accuracy of relative work index efficiency ratings can be estimated by once again applying the standard mathematics for accumulation of error to the formulation for work index efficiency:

$$\text{Work Index Efficiency} = \frac{\text{Test Work Index}}{\text{Operating Work Index}} \times 100$$

$$\text{Operating Work Index} = \frac{\text{kw Draw}}{\text{Hourly Tonnage}} \Bigg/ \left(\frac{10}{\sqrt{P80}} - \frac{10}{\sqrt{F80}} \right)$$

As discussed in section 3.1.2, relative power draw readings on the same equipment may be determined within $\pm$ 2%. According to the equipment manufacturers, relative weightometer readings are normally accurate to $\pm$ 1% (Board, 1985). When the same screens are used, experimental error in determination of the 80 percent passing sizes is negligible. Sampling error for P80 is also small, based on the fineness of the sample and the large sample size. The sampling error for the F80 can once again be estimated using Gy's formula, since the F80 can be taken as a fundamental characteristic of the sample. The primary sample size is approximately 18 kg. Therefore, applying the identical procedure discussed in section 3.1.3.2 to the same example:

$$\text{Primary } S^2\ (FE) = (2.8)\ (0.5)\ (0.5)\ (0.25)\ (1.7)^3 \left(\frac{1}{18{,}000}\right)$$

$$= 4.8 \times 10^{-5}$$

The secondary sample size for the coarse screen sizes is approximately 10 kg. Therefore:

$$\text{Secondary } S^2\ (FE) =$$

$$(2.8)\ (0.5)\ (0.5)\ (0.25)\ (1.7)^3 \left(\frac{1}{10{,}000} - \frac{1}{18{,}000}\right)$$

$$= 3.8 \times 10^{-5}$$

The variance of the combined sampling error is the sum of these, or 8.6×10^{-5}. Then S = 0.0093. For the 95% confidence limits on the measured F80 of 12,090 um:

$$(1.96)\ (S) = \frac{\pm x}{12{,}090}$$

Solving, x = $\pm$ 220 µm.

For the operating work index for survey no. 2, then:

$$F80 = 12{,}090 \pm 220\ \mu m$$

$$\text{kw draw} = 188 \pm 3.8\ \text{kw}\ (2\%)$$

$$\text{Hourly tonnage} = 70.3 \pm 0.7\ (1\%)$$

When these values are combined in the above equation for the operating work index, this yields a total error of ± 0.33 on the calculated work index of 13.2 kwh/t. When this is in turn combined (using the combined error for quotients) with the test work index of 13.8 ± 0.3 (or 2%, using the same seives), the rod milling work index efficiency becomes 1.05 ± 0.036, or 105% with a 95% confidence interval of between ±3 and 4%.

The error associated with relative ball milling work index efficiencies will be, at worst, similar to that for rod milling (Laplante et al, 1988). Smaller particle sizing will reduce sampling error, and electrical instruments do not fluctuate as widely as those on rod mills. The experimental laboratory test procedure is very similar.

3.3 EVALUATION OF THE ROD MILL CIRCUITS AT KIDD CREEK MINES AND LES MINES SELBAIE

3.3.1 Overview of Plant Testwork

The review of the literature on the theory and practice of rod milling, as reported in section 1, lead to the identification of the following items for possible evaluation in the case study plants:

1. Fines addition to the rod mill feed;
2. Mill speed;
3. Increased frequency of rod charging;

4. Rod size;

5. Feed water addition rate.

The investigation of mill speed up was selected first, as this could be done without extensive process operating data. Increasing the frequency of rod charging to maintain more constant mill power draw was deemed generally adviseable from the point of view of circuit operating stability, but could not be evaluated in quantitative terms without first providing clear circuit performance objectives and an evaluation methodology. Along with rod size adjustment, this was postponed as a longer term consideration. Fines addition and feed water addition rate eventually became the main topics of investigation.

A summary of the grinding performance testwork data generated for the Kidd Creek Mines B circuit 3.20 m by 4.88 m (10.5 by 16 ft.) rod mill during the course of the study is given in Table 3-8. Similar data for the Selbaie 2.44 m by 3.66 m (8 by 12 ft.) rod mill is given in Table 3-9.

3.3.2 Mill Power Draw vrs. Charge Level Relationships

3.3.2.1 Introduction

In order to quantify the economic effects of changes in operating parameters and grinding efficiency, in particular in terms of power draw and steel consumption, it is necessary to establish the relationship between the steel load level and mill power draw. Mill power draw at the pinion shaft was determined using the procedure outlined in Section 3.2. The

Table 3-8. Rod Mill Testwork Summary, Kidd Creek Mines Ltd.

Test Survey No.	1	2	3A	3B	3C	4D	4E	4F
Date	Dec. 18/85	Dec. 19/85	Aug. 13/86	Aug. 13/86	Aug. 13/86	Oct. 23/86	Oct. 23/86	Oct.23/86
Time			9:40-10:58 AM	12:40-1:50 PM	2:30-3:30 PM	8:45-9:53 AM	10:57-11:55 AM	1:17-2:20 PM
Conditions, Comments	Normal.	Normal.	Normal.	Fines added to feed.	Increased feed water.	Normal.	Increased feed water.	Increased feed water.
Rod Mill Feed								
Rate (dry t/h)	126.6	131.8	126.8	126.8	126.8	132.0	132.0	131.8
% Solids by Weight	97.8%	98.1%	97.8%	97.9%	97.8%	97.9%	97.9%	97.8%
Size, F80 (µm)	14,870	15,020	15,280	15,140	16,910	15,180	13,980	13,550
% - 20 m (850 µm)	20.5%					19.2%	20.6%	22.3%
Bond Test W.I.,10 m (kwh/t)	17.4	17.8	18.7	17.7	-	17.1	17.2	- **
Standardized W.I. (kwh/t)	17.4	17.7	18.6	17.6	-	17.0	17.0	-
Rod Mill Discharge (New Product)								
% Solids by Weight	81.1%	80.7%	81.7%	81.4%	76.1%	81.4%	77.0%	72.2%
Size, P80 (µm)	1,460	1,650	1,397	1,389	1,304*	1,413	1,251	1,138
% - 20m (850 µm)	62.1%					63.9%	67.3%	70.2%
Rod Mill Performance								
Power Draw (kw)	427	441	522	508	519	488	485	486
Reduction Ratio, F80/P80	10.2	9.1	10.9	10.9	13.0*	10.7	11.2	11.9
Work Applied (kwh/t)	3.37	3.35	4.12	4.01	4.09	3.70	3.67	3.69
Operating W.I. (kwh/t)	18.7	20.4	22.1	21.4	20.5*	20.0	18.5	17.5
W.I.Eff.,Test/Operating (%)	93%	87%	84%	82%	-	85%	92%	-

* Approximate values. Circuit unstable.
** Test failed to reach equilibrium.

Table 3-9. Rod Mill Testwork Summary, Les Mines Selbaie

Test Survey No.	1	2	4A	4B-1	4B-2	4C	8A	8B	9
Date	Nov.12/85	Nov.13/85	July 8/86	July 8/86	July 8/86	July 8/86	Nov. 8/86	Nov. 8/86	Nov.11/86
Time	1 - 3 PM	9 - 12 AM	9:46-10:14 AM	10:26-10:53 AM	1:34-1:55 PM	2:05-2:23 PM	11:19 AM-12:34 PM	3:03-4:05 PM	3:29-4:31 PM
Conditions, Comments	Normal, 89 mm rods.	Normal.	Normal.	Increased feed water.	Fines added to feed.	Normal.	Normal 76 and 89 mm rods.	Increased feed water.	Normal, worn liners.
Rod Mill Feed									
Rate (dry t/h)	67.1	70.3	58.1	58.2	54.4	56.3	68.4	67.1	62.1
% Solids by Weight	94.2%	94.3%	93.9%	93.4%	94.8%	94.9%	96.0%	95.2%	94.9%
Size, F80 (µm)	11,750	12,090	12,450	12,280	12,380	12,680	12,330	12,310	11,645
% - 20 m (850 µm)	18.0%	17.0%	14.7%	14.7%	13.5%	14.4%	16.4%	16.9%	19.4%
Bond Test W.I.,10 m (kwh/t)	14.6	13.8	15.0	15.4	15.5	14.9	14.9	15.9	16.4
Standardized W.I. (kwh/t)	14.6	13.8	15.0	15.4	15.4	14.8	14.8	15.8	16.3
Rod Mill Discharge (New Product)									
% Solids by Weight	80.9%	81.3%	80.8%	71.2%	82.1%	80.4%	81.4%	77.1%	81.5%
Size, P80 (µm)	1,150	1,160	1,135	938*	1,165*	954*	1,178	1,031	1,090
% - 20m (850 µm)	69.8%	69.8%	70.0%	77.0%*	67.8%*	76.5%*	70.3%	74.4%	72.6%
Rod Mill Performance									
Power Draw (kw)	188	188	198	203	190	203	209	214	197
Reduction Ratio, F80/P80	10.2	10.4	11.0	131.1*	10.6*	13.3*	10.5	11.9	10.7
Work Applied (kwh/t)	2.80	2.67	3.41	3.49	3.49	3.61	3.06	3.19	3.17
Operating W.I. (kwh/t)	13.8	13.2	16.4	14.8*	17.2*	15.3*	15.2	14.4	15.1
W.I.Eff.,Test/Operating (%)	106%	105%	91%	104%*	90%*	97%*	97%	110%	108%

* Approximate values. Circuit unstable.

average charge level and total weight of the steel charge were calculated using the following approximate relationships developed by Bond (1961), and the weight of a single rod added at rod charging.

$$Vp = 1.13 - 1.26 \frac{Q}{D}$$

$$Tr = \frac{Vp \quad D^2 \quad L}{0.2123}$$

Vp, fraction of the mill interior occupied by the grinding charge

Q, average vertical distance from the inside top of the mill to the charge (m)

D, inside diameter (m)

L, inside length of the mill (m)

Tr, tonnes of new rods in the mill

Charge level and power draw measurements were then compared with those estimated by the Allis-Chalmers power draw formula (Rowland, 1982).

$$kWr = 1.752 \ D^{0.34} \ (6.3-5.4 \ Vp) \ Cs$$

kWr = kilowatts per metric ton of rod load

Cs = fraction of critical speed

3.3.2.2 Measurements at Kidd Creek Mines

Rod mill charge level and power draw measurements were initiated on a regular basis for the long-term test on operating the B circuit rod mill with increased feed water rate. Initial readings taken on February 3, 1987, are described here.

At 7:10 a.m., the following electrical readings (average) were taken in the motor control centre.

Amps: 102

Volts: 4090

Power factor: 0.77

That morning, the mill was stopped for rodding, after a grind out period of 5 minutes. The mill was entered, and measurements were taken from the inside top of the mill (at the narrowest liner thickness) to the charge level, as follows.

Feed end : 1905 mm (75 in.)

Centre : 1816 mm (71.5 in.)

Discharge end: 1747 mm (68.5 in.)

It was also noted that the top of the charge rested approximately 254 mm (10 in.) above the inside diameter of the discharge and trunnion liner, which itself was 914 mm (36 in.) in diameter, and even with the feed and trunnion liner, 813 mm (32 in.) in diameter. An average liner thickness of approximately 133 mm (5.25 in.) was also noted. The liners had been changed 7 months previously.

Fifteen 102 mm (4 in.) rods were charged to the mill, and the mill was restarted at 10:45 a.m. Fifteen minutes later, the following electrical readings were taken.

Amps: 108

Volts: 4145

Power factor: 0.77

These and subsequent rod mill power draw versus charge level readings are summarized in Table 3-10 (Rumfeldt, 1987). The method of charge level estimation was found to be only approximate, as the scatter in the reported data indicate. However, a good general indication of the mill power draw capability over a range of operating levels can still be seen from the data.

With the mill liners in their existing condition (7 months since installation) mill power draw varies from 500 to 550 kw as the charge varies over the normal operating range of about 36 to 40 percent of the mill volume. The normal operating charge level in the mill is about 38% of mill volume, which occurs when the charge level is approximately 250 mm (10 inches) above the discharge trunnion liner opening.

The power draw of this mill based on the Allis-Chalmers formula (Rowland, 1982) is 549 kw at 40 percent charge level, which is slightly above that obtained. The power draw curve shown in Figure 3-6 is based on the same Allis-Chalmers formula with the constant adjusted so it passes through the mean of the charge level and power draw readings. The motor is capable of providing an output of 597 kw without impinging on its service factor, and total available power output is 686 kw.

Note that with new liners, the inside diameter and total volume of the mill is reduced. Despite operating with a high charge level, relative to the mill discharge opening, the total weight of rods in the mill is reduced, and the power draw will be lower than with the worn liner condition. This is demonstrated by the data from Les Mines Selbaie.

Table 3-10. Rod Mill Power Draw and Charge Level Readings at Kidd Creek

Reading No.	Date - Time	Approximate Charge Level %	Approximate Charge Level Tonnes	kw Draw
1	Feb.3/87 - 7:10	37.7	74.6	521
	15 rods added (102 mm)			
2	Feb.3/87 - 10:45	40.0	79.1	559
3	Feb.6/87 - 7:30	39.9	78.9	526
	12 rods added			
4	Feb.6/87 - 12:30	41.7	82.5	549
5	Feb.10/87 - 7:30	35.6	70.3	505
	15 rods added			
6	Feb.10/87 - 15:30	37.8	74.8	541
7	Feb.13/87 - 7:30	36.6	72.4	497
	16 rods added			
8	Feb.13/87 - 12:30	39.1	77.2	540
9	Feb.17/87 - 7:30	35.6	70.3	499
	15 rods added			
10	Feb.17/87 - 15:00	37.8	74.8	535
11	Feb.27/87 - 7:40	36.6	72.4	504
	17 rods added			
12	Feb 27/87 - 15:00	39.2	77.5	549
13	Mar.6/87 - 7:40	33.4	66.0	474
14	Mar.6/87 - 13:00	34.4	68.1	486
15	Mar.10/87 - 7:40	31.2	61.7	437
16	Mar.13/87 - 7:40	32.3	63.8	454

(Table 3-10. Continued)

Reading No.	Date - Time	Approximate Charge Level %	Tonnes	kw Draw
17	Mar.17/87 - 7:40	32.3	63.8	443
18	Mar.20/87 - 8:30	32.3	63.8	456
19	Mar.31/87 - 7:50	31.2	61.7	420
20	Apr.3/87 - 7:50	32.3	63.8	438
21	Apr.10/87 - 7:40	32.3	63.8	444
22	Apr.14/87 - 7:40	32.3	63.8	440
23	Apr.16/97 - 7:45	32.3	63.8	452
24	Apr.28/87 - 7:10	32.0	63.2	442

3.3.2.3 Measurements at Les Mines Selbaie

Power draw and charge level readings were taken on the rod mill just before it was shut down for relining on November 11, 1986, and subsequently, during the month of January, 1987. Several level measurements were taken inside the mill, while most were estimated by eye, relative to the trunnion opening, from outside the mill. A standard grind out period of 4 minutes was used with the water on and the feed off. The charge level was also taken as the average at the centre of the mill, as the rods sloped about 100 mm towards the feed end.

Due to meter fluctuations, rod mill ammeter readings were taken approximately 10 times each, noting the high and low

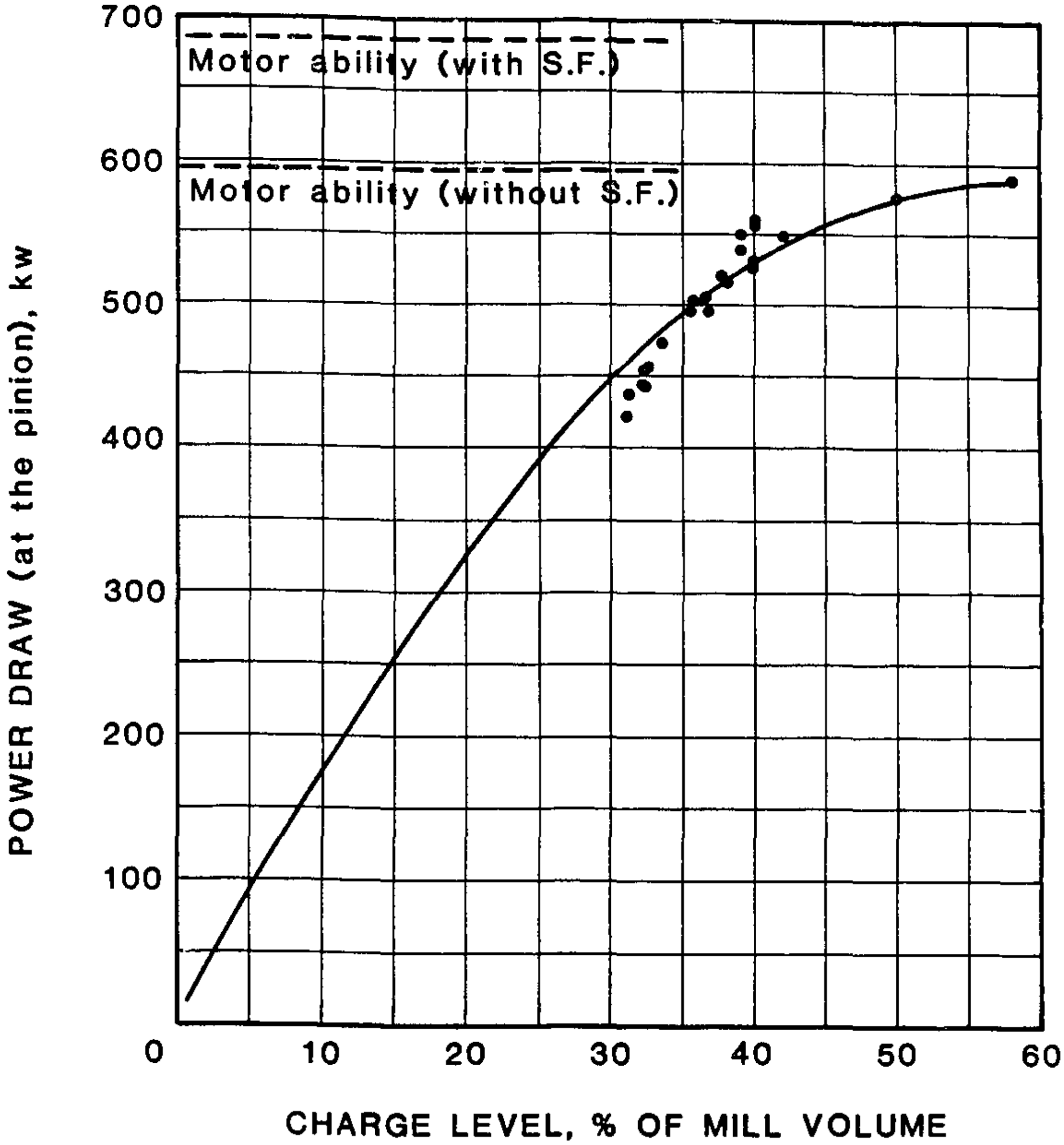


Figure 3-6. Power Draw vs. Charge Level, Kidd Creek 3.20 x 4.88 m Rod Mill

reading over a period of a few seconds each time, and then averaged for all the readings. Voltage readings were averaged for each of the three power supply phases. The reported charge level is the average distance above the inside bottom of the discharge end trunnion liner. A summary of the data is given in Table 3-11.

The mill inside diameter was measured in the fully worn condition to be approximately 2362 mm (93 inches). This is in close agreement with the calculated inside diameter, based on the average thickness of the discarded worn liners, 36.5 mm (1.44 inches), and the mill inside shell diameter, accounting for the 6 mm (0.25 inch) rubber backing. The newly relined mill has an average calculated inside diameter of 2235 mm (88 inches).

The rod mill power draw versus charge level readings for the worn and new liner conditions are summarized in Table 3-12. With the mill liners in their fully worn condition mill power draw varies from 212 to 197 kw from the highest to lowest recorded charge levels in the mill. This variation is slightly more than experienced during normal mill operation, as rods were not added during the entire 5 day period in order to prepare for the mill relining. The normal operating charge level in the mill is about 42% of mill volume, which occurs when the charge level is approximately 150 mm (6 inches) below the centreline (also 150 mm above the trunnion opening) at the discharge end, and at an ammeter reading of approximately 38 amps, with worn liners.

Table 3-11. Rod Mill Motor and Charge Level Readings at Les Mines Selbaie

Reading No.	Date - Time	Hrs. of Operation Since Rods Added	Amperage (amps)	Voltage (volts)	Approx. Charge Level, mm (in.)
1	Nov. 6/86 - 14:00	-	35.9	3980	50 (2)
	5 rods added, 76 mm (3 in.)				
2	Nov. 6/86 - 16:00	-	36.4	4030	- -
	10 rods added, 76 mm (3 in.)				
3	Nov. 6/86 - 20:30	-	38.9	3930	- -
4	Nov. 7/86 - 10:00	0	38.1	4000	150 (6)
5	Nov. 8/86 - 13:00	27	37.9	3980	125 (5)
6	Nov. 9/86 - 10:00	48	37.5	3970	100 (4)
7	Nov. 10/86 - 10:00	72	37.1	3925	75 (3)
8	Nov. 11/86 - 04:00	90	35.9	3975	75 (3)
	Mill shut down for relining				
9	Jan. 26/87 - 14:00	-	34.9	3990	175 (7)
	6 rods added, 76 mm (3 in.)				
10	Jan. 26/87 - 15:00	-	35.2	4000	- -
11	Jan. 27/87 - 11:00	20	35.7	3850	- -

Table 3-12. Rod Mill Power Draw and Charge Level at Les Mines Selbaie

Reading No.	Date - Time	Hrs. Operation	Approximate Charge Level %	Approximate Charge Level Metric Tons	kw Draw	Mill I.D.
1	Nov. 6/86-14:00	-	40	38.5	199	2362 mm (93") (Worn Liners)
	5 rods added (76 mm)					
2	Nov. 6/86-16:00	-	41	39.1	203	2362 mm (93")
	10 rods added (76 mm)					
3	Nov. 6/86-20:30	-	42	40.4	212	2362 mm (93")
4	Nov. 7/86-10:00	0	42	40.4	210	2362 mm (93")
5	Nov. 8/86-13:00	27	41	39.5	208	2362 mm (93")
6	Nov. 9/86-10:00	48	39	37.5	206	2362 mm (93")
7	Nov.10/86-10:00	72	37	35.6	202	2362 mm (93")
8	Nov.11/86-04:00	90	37	35.6	197	2362 mm (93")
	Mill shut down for relining					
9	Jan.26/87-14:00	-	43	37.0	190	2235 mm (88") (New Liners)
	6 rods added (76 mm)					
10	Jan.26/87-15:00	-	44	37.7	194	2235 mm (88")
11	Jan.27/87-11:00	-	43	37.0	192	2235 mm (88")

The power draw of this mill based on the Allis-Chalmers formula is 213 kw at 40 percent charge level, which is slightly above that obtained with fully worn liners. The theoretical power draw curves for new and worn liners, fitted to the mean of the data points obtained from the power draw readings are shown in Figure 3-7. The motor is capable of providing an output of 298 kw without impinging on its service factor, which is approximately 50 percent above the present loading.

3.3.3 Rod Mill Speed

3.3.3.1 Selection of an Alternative Mill Speed at Kidd Creek

The example presented here is the consideration of speed up of the rod mill at Kidd Creek. The present speed of the rod mill is 17.25 r.p.m. or 70.6 percent of critical speed, based on a mill diameter inside new liners of 3.00 metres (9.83 feet). In order to evaluate the different options for mill speed up, a target speed of between 75 and 82 percent of critical was selected. A number of possible options for modifying mill speed were then considered.

Inquiries lead to the conclusion that the cost of motor speed adjustment, by variable frequency control, would be prohibitive. Rumble Controls Ltd. quoted a budget price of $180,000 (excluding transformers and field installation and start-up) for a frequency control system for a motor of this size. A pinion change, at a total estimated cost of approximately $40,000, was therefore considered the only viable approach at this time.

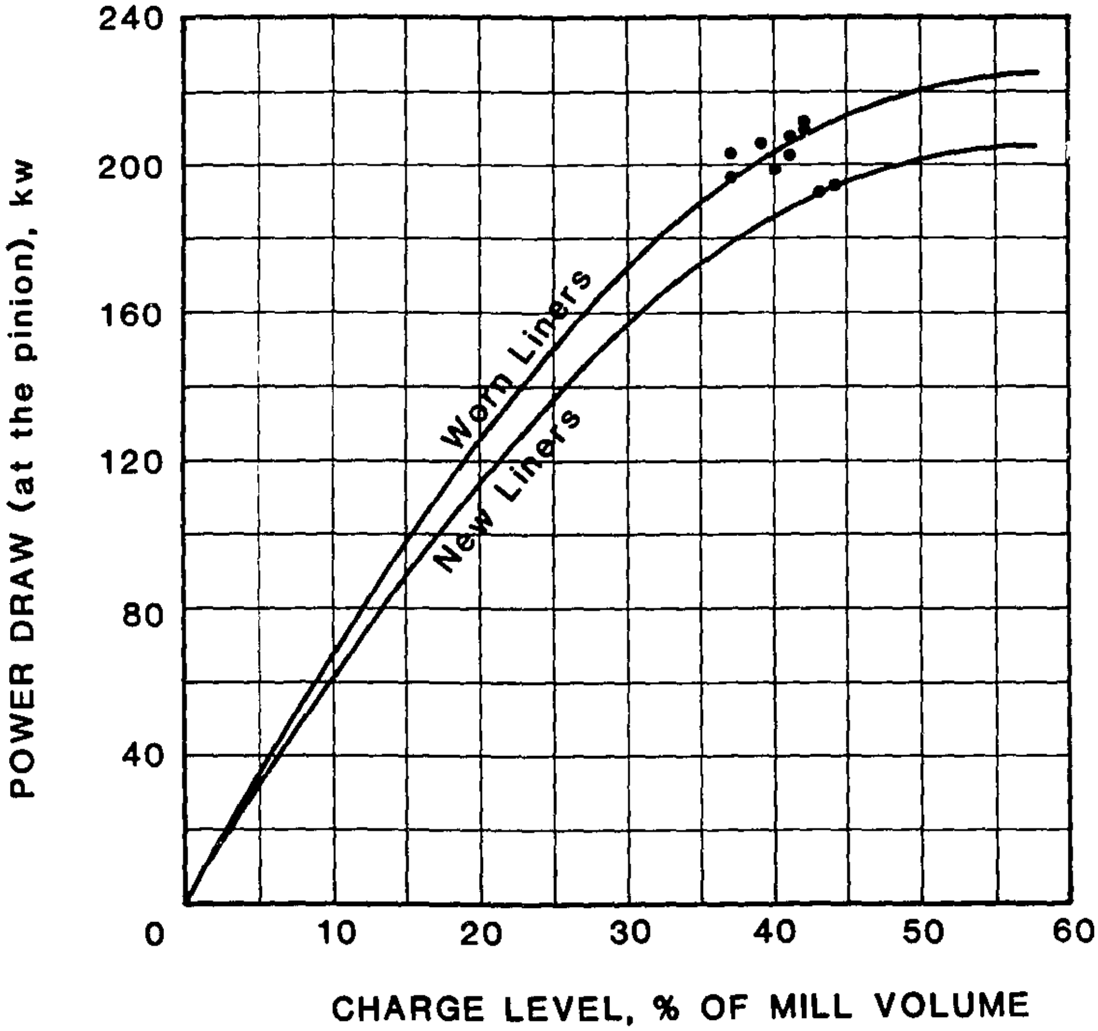


Figure 3-7. Power Draw vs. Charge Level, Selbaie 2.44 x 3.66 m Rod Mill.

A change from the presently operating 21 tooth pinion to one with either 23 or 24 teeth (assuming either to be feasible in design and installation) would yield the following mill speeds:

Table 3-13. Number of Pinion Teeth vs. Rod Mill Speed at Kidd Creek

No. of Pinion Teeth	Mill Speed, R.P.M.	% of Critical Speed
21	17.25	70.6%
23	18.89	77.3%
24	19.71	80.6%

Since the pricing of either option is likely to be very similar, whether the 23 or the 24 tooth pinion should be selected depends on:

a) Whether both are feasible in design (i.e., suitable tooth profile and strength design, suitable non-integer ratio between number of gear and pinion teeth, etc.) and installation (i.e., available adjustment in position of mill bearings to accomodate new pinion pitch circle diameter, location of gear guards, etc.);

b) The estimated relative benefits to be gained by the two higher speeds;

c) Whether there is any perceivable increased risk at the higher of the two speeds.

The manufacturer confirmed that both of the new pinions were feasible in design, and that the increased pitch circle diameter would mean the mill would have to be shifted approximately 38 mm (1.5 in.) for the 23 tooth or 45 mm (1.75 in.) for the 24 tooth design. The position of the gear guards may have to be shifted, depending on the existing clearance, and the slots in the bearing bases also may have to be ground wider to accomodate the shift. All such adjustments are normal procedure for a mill speed increase. The mill bearing bases are designed to accomodate such adjustments so that the mill drive motor need not be moved. The ability of the existing clutch to start the mill for either of the higher speeds, even at mill charge loadings as high as 45%, was also verified by the equipment manufacturer (Kerber, 1986).

Since media consumption per unit power will be expected to decrease with higher critical speed, the objective is to select the highest possible speed without exceeding the practical limit. Just what this limit is can be judged from a number of factors:

1. Previous operation of the Selbaie rod mill at 82.5% of critical resulted in: (a) oversize discharge surging from the mill, and largely passing over the trommel screen; (b) intermittant drive slippage (at the fluid coupling); and (c) reduced power draw. An important qualifier is that the mill was operated at very high charge level at the time. Both high speed and high charge level would contribute to intermittant centrifuging of the charge (and consequential

loss of grind and power draw), coinciding with build up, and then discharge, of mill solids. As indicated from the information that follows, there is a very good chance that these problems would be alleviated at reduced charge level.

2. Operation of the 3.51 by 3.66 m (11.5 by 12 ft.) long rod mill at Sullivan at 81.3% of critical speed (19 r.p.m., based on inside diameter of 10.75 ft.) and 37 - 39% charge level gave no operating problems, excellent grind performance, and remarkably low steel consumption on a highly abrasive ore, as summarized below (Banks, 1952).

Feed rate:	302 mtph (333 stph)
Feed size:	80% minus 25 mm (1 in.)
Product size:	80% minus 2360 um (8 mesh)
Rod consumption:	0.0753 kg/kwh (0.166 lbs/kwh)
Power draw:	708 kw (950 HP)

3. From a discussion with Mike Redfearn (1986) of Gibraltar Mines, operation of their 4.11 m by 6.10 m (13.5 by 20 ft.) rod mills at 78% of critical results in absolutely no operating problems whatsoever. (Note that they have also run at 67 and 74% of critical.)

4. Table 3-1 provides a number of successful "high speed" rod mill installations.

From the above, we can conclude that running the rod mill at either 77.3 or 80.6% of critical speed presents little risk.

3.3.3.2 Economic Analysis

To establish the possible benefits of operating at higher speed, the power draw/speed/charge level interelationships for the mill must be considered.

The power draw formula for rod mills indicates that power draw is directly related to mill speed in the normal operating range. Although we know that above a certain point power draw will eventually start to level off with increased speed (and charge level) as centrifuging is induced, there is no mention of this experience at Sullivan or at Kloof, so we can safely extend this to 82.5% of critical speed. The charge level versus power draw curves at 77.3 and 80.6% of critical speed are shown in Figure 3-8, as estimated from the present mill power draw curve.

It can be seen from Figure 3-8 that at increased mill speeds of 77.3 or 80.6% of critical, the charge level could be reduced from 38% to approximately 33 or 31 percent of mill volume, respectively, to maintain the same average mill power draw of 520 kw. The lower speed option of the two will be used as the basis for estimation of the minimum economic benefit to be gained by mill speed up.

The statistical analysis of rod consumption versus mill speed (Figure 3-4) indicates approximately a 10 percent average reduction in steel consumption for an increase in critical speed of 7 points. On this basis, the yearly savings would be as follows.

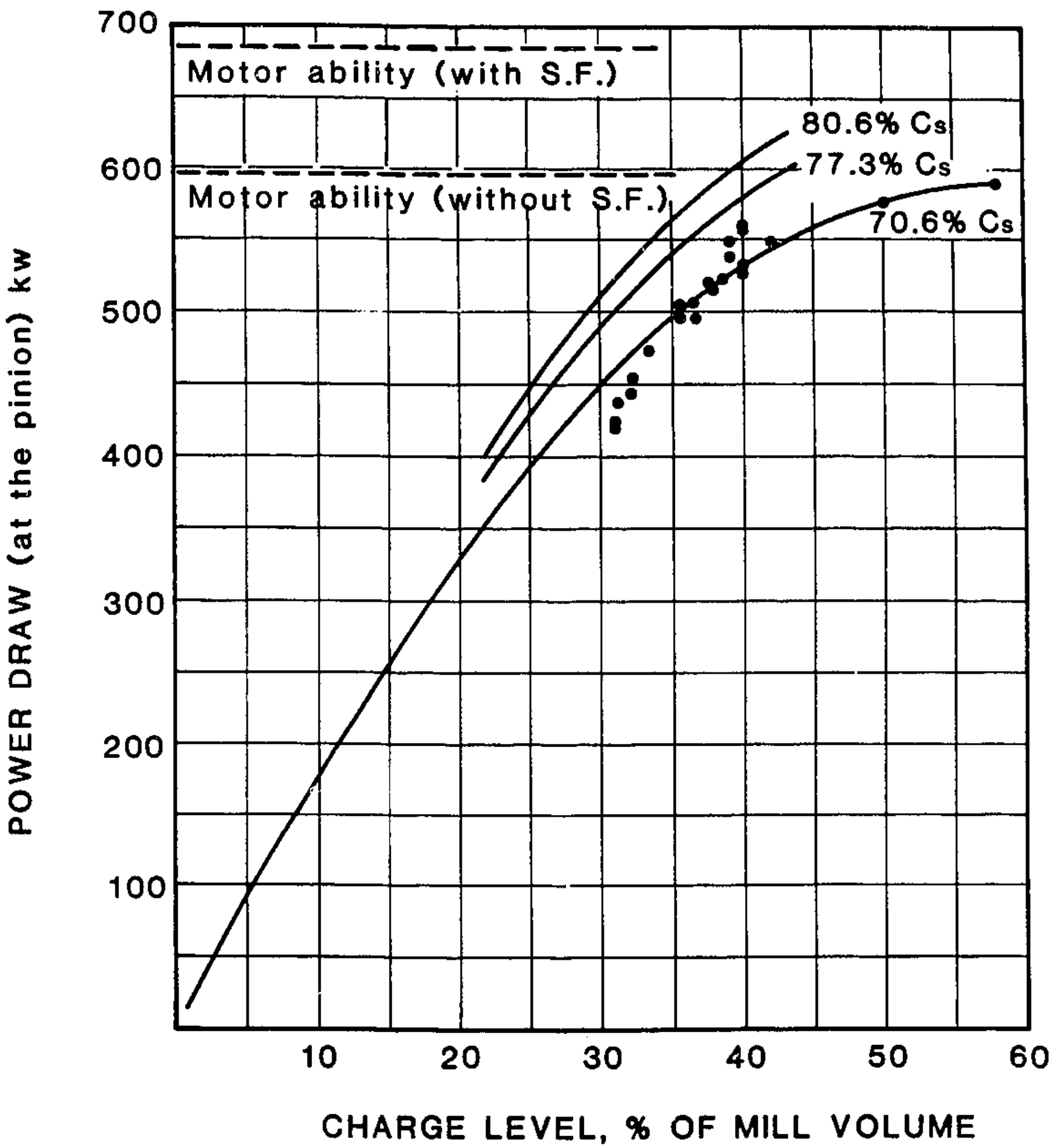


Figure 3-8. Power Draw vs. Charge Level, Kidd Creek Rod Mill at Different Mill Speeds.

Current cost: 29.8 ¢/t
Expected saving: 10% of 29.8 ¢/t = 2.98 ¢/t

2.98 ¢/t x 3240 t/day x 360 days/yr. = $35,000/yr.

However, this represents the minimum saving that would be expected as all the data represented in Figure 3-4 are generally for mills operated at high (approximately 40%) charge level. The units of steel load per unit of power draw are much more reduced by reducing charge level in conjunction with mill speed increase, as shown below.

Table 3-14. Steel Load to Power Draw Ratios at Different Rod Mill Speeds and Load Levels

Case	Mill Speed (% Cs)	Steel Load (% Vp)	Power Draw (kw)	Ratio of Steel Load to Power Draw
1	70.6	38	520	0.0731
2	77.3	38	569	0.0668
3	77.3	33	520	0.0635

Note that the relative decrease in steel load per unit power draw from case 1 to case 2 is 9%, which closely coincides with the regression line of Figure 3-4. However, the decrease from case 1 to case 3 is 13 percent, which exactly coincides with the percentage reduction of steel load in the mill. The latter factor would apply if corrosion was the sole media wear mechanism, due to its direct relationship to surface area under attack. The role of impact and abrasion means that actual savings would likely fall between the previous estimate of 10 percent and the reduction of steel load of 13 percent. At a cost of $40,000 for the replacement pinion, including installation, this would mean a payback period of approximately one year.

There does not appear to be any reason to be concerned about loss of liner life at increased mill speed, as evidenced by experience from Snow Lake (MacDermid, 1953) and by previous rod mill speed increases at Kidd Creek. Slippage between the charge and the shell appears to be a major factor contributing to poor liner life. Increasing mill speed (similar to renewing the internal lifter profile) would tend to decrease slippage. It has also been shown that maximum lift (and impact conditions) occur near 75 percent of mill critical speed, and decreases at either higher or lower speeds (McIvor, 1983).

A disadvantage encountered at reduced charge level is that mill power draw is more sensitive to variations in charge level (i.e., the steepness of the power draw versus charge level curve is higher). However, this effect is small.

Since the lost production during the down time needed to change the pinion is extremely costly, this should be coordinated with a scheduled shutdown, if possible. Otherwise, the production can be made up by a 1% increase in average tonnage rate over the period of a month, assuming 8 hours for the change.

In conclusion, this analysis shows a one year payback on an investment of approximately $40,000, or a saving of approximately $160,000 over 5 years for a rod mill speed-up to 77.3%

of critical speed. There are no physical or mechanical design limitations which cannot be overcome. Available mill power will also increase, should higher tonnage or a finer grind be desired.

3.3.4 Fines Addition to Rod Mill Feed

3.3.4.1 Introduction

The possibility of improving rod milling efficiency through fines addition to the rod mill feed was indicated by previous work at the Frood-Stobie concentrator of INCO Limited (Zickar et al, 1981). Like the Selbaie operation, fines were pre-washed from the feed to the crushing plant, resulting in a rod mill feed with substantially less fine material than normally expected. Furthermore, work at Tennessee Copper (Myers and Lewis, 1946) with closed circuit classification on their rod mill suggested that additional fines could help the operation of any rod mill (refer to section 3.1). A general program of fines addition testing was therefore undertaken for both the Selbaie and Kidd Creek operations.

3.3.4.2 Experience at the Frood-Stobie Concentrator

A visit was paid to the Frood-Stobie concentrator to observe the practice of fines addition to the rod mills. Fines addition was initiated in 1972 to eliminate the problem of

frequent rod tangles, but also was reported to improve rod mill grinding efficiency by approximately 15%, based on (uncorrected) operating work indexes with and without fines addition (Zickar et al, 1981).

In the crushing plant, coarse ore is washed to remove fines before two-stage, open-circuit crushing. The product size is typically 90% minus 19 mm (0.75 in.), but may at times become substantially coarser and slabby (50 to 75 mm in one dimension) with harder to crush ore. This material would segregate in the fine ore bins, which when drawn down, would result in rod tangles.

The fines from the crushing plant are classified in a hydrocyclone, the fine product being sent to the concentrating circuit after thickening, and the coarse product to the ball milling head tank, which also receives the rod mill and ball mill discharges, and feeds the grinding circuit hydrocyclones. The material from this head tank, which is the ball mill circuit cyclone feed, is the source for fines that are added to the feed end of the rod mills. Three lines from the head tank discharge directly into each of the rod mill feed chutes with the fine ore. There is no flow control except for a straight on or off air actuated pinch valve. This is the only source of water added to the rod mill feed, and a sensing device, interlocked with the rod mill feed conveyors, is used to ensure a positive flow.

Crushing plant fines can make up to about 20% of the plant feed tonnage when the crushing plant is running, which is about 75% of the time.

They have not noticed that any ball chips make their way back into the rod mills, although no precautions are taken to avoid this.

Some of the basic operating data on the rod mills and fines addition practice, taken from the published report, are summarized below (Zickar et al, 1981).

Mill size:	4.11 x 5.49 m (13.5 x 18 ft.)
Critical speed:	66%
Media:	102 mm (4 in.) rods
Power draw:	1250 kw
Feed rate:	317 mtph (350 stph)
Mill % solids:	80%
Fines addition:	63-73 mtph solids at 60% solids by weight, typically 80% minus 20 mesh, 30% minus 200 mesh
Rod Mill Feed:	Fine ore only: 4 to 5% minus 200 mesh Fine ore plus fines: 10% minus 200 mesh

3.3.4.3 Testwork at Selbaie and Kidd Creek

Taking the percentage of minus 75 µm (200 mesh) material as the definition of fines, the work at Frood-Stobie indicated the need to raise the fines content of rod mill feed to a minimum of approximately 10 to 12 percent to gain any advantage. Rod mill feed at Selbaie, where the crusher feed is pre-washed, contains approximately 5% minus 200 mesh. The natural feed at Kidd Creek, although not pre-washed, is coarser overall, and contains approximately 8 to 9% minus 200 mesh.

Ball mill circuit feed was chosen as the most convenient source of fines at both plants, and suitable piping and valve connections installed. Water connections were also provided for flushing to avoid sanding after the tests. The required flow rate of cyclone feed material needed to raise the total amount of fines to the desired level was calculated from earlier size distribution data. The valve settings to achieve desired flows were calculated by water and solids mass balance of the rod mill feed, cyclone feed fines, rod mill feed water, and rod mill discharge. Actual flowrates were calculated during each of the tests in the same way. Due to the minimum amount of additional effort required, it was decided to test for the effect of feed water addition rate during the same tests.

The first full test was carried out at Selbaie on May 29, 1986 (Selbaie survey no. 3). Unfortunately, problems with the samples were encountered, and upon checking, size distribution data could not be reproduced. As well, the desired level of fines in the rod mill feed was not achieved. Although other results had to be rejected, a reduction in power draw of approximately 6% (from 203 to 190 kw) was evident.

A repeat test (Selbaie survey no. 4) was carried out on July 9, 1986. Details are provided in Appendix G. Also refer to Table 3-9 for a summary of the test results. Note that due to concerns with the rod mill foundation, the rod size was changed from 89 to 76 mm (3.5 to 3 in.) during the previous month. A check of the mill contents the following November showed that both 89 to 76 mm rods had been added during the second half of 1986.

The rod mill was operating in 4 modes during this test for approximately 20 to 30 minutes in each mode, as follows.

Mode A. Normal
Mode B-1. Increased rod mill feed water addition rate.
Mode B-2. Cyclone feed fines added to rod mill feed.
Mode C. Normal.

Note that a five minute stabilization period was allowed after feed adjustments before sampling for modes B-1, B-2 and C.

A similar test was carried out at Kidd Creek (August 13, Kidd Creek survey no. 3), as described in Appendix H. Please refer to the summary in Table 3-8. This time, three operating modes were used.

Mode A. Normal.
Mode B. Cyclone feed fines added to rod mill feed.
Mode C. Increased rod mill feed water addition rate.

This time, a one hour sampling period was used. Following sampling of the normal operating condition the cyclone feed fines were turned on, and the feed water adjusted. The mill then had approximately 40 minutes to stabilize before the sampling period for mode B. However, only 10 minutes were allowed between the feed water addition rate adjustment and sampling for mode C.

Bond laboratory work index tests were first carried out on the rod mill feed samples from Les Mines Selbaie. This revealed an apparent discrepancy in the results from the two normal operating conditions (modes A and C), which should have yielded similar operating efficiencies. This, in fact, lead to investigation of sources of error related to the Bond work index laboratory tests. However, the probable source of the problem was found to be the lack of a sufficient period of time for mill stabilization. Although actual residence time is on average, very short (1 to 2 minutes), there is still a great deal of back mixing in a rod mill, meaning that a stabilization period of 30 to 60 minutes is required (Laplante, 1986). Results for modes B-1, B-2, and C could therefore be taken as only approximate.

For the test at Kidd Creek, this also meant that the results for mode C could also only be taken as approximate. Laboratory work index tests were therefore performed on feed samples from the previous two operating modes only.

3.3.4.4 General Conclusions

The results from both tests are summarized in Tables 3-8 and 3-9, and may be considered together to arrive at the following general conclusions:

1. A period of approximately one hour should be allowed for the rod mill to stabilize after making an adjustment to the feed conditions. The minimum time period for stabilization is open to some speculation, subject to further testing.

2. A reduction in mill power draw is apparent during fines addition in the range of 3 to 6%.

3. There is no indication that the addition of cyclone feed fines improves rod mill grinding efficiency. Although the test results during mode B-2 at Selbaie are only approximate, no efficiency gain is apparent. Test results from Kidd Creek, which can be taken as valid within defined statistical limits, also show no efficiency improvement.

4. There is a strong indication that addition of extra feed water has a positive effect on rod mill grinding efficiency. Although results from the above tests at both locations must be taken as preliminary efficiency improved approximately 10 percent at Selbaie, and the operating work index improved (i.e., was lowered) by over 5 percent at Kidd Creek.

Increased water addition appeared to be very promising, and further tests carried out on this variable follow. It may also be possible that addition of a more dilute stream containing finer solids than has been tested may offer the advantage of improved grinding efficiency, and therefore may be a suitable subject for future testwork. However, if possible, these fines should originate prior to the cyclone in the first ball milling stage of the process where their return will not be a detriment to ball mill circuit classification performance.

3.3.5 Feed Water Addition Rate

3.3.5.1 Testwork at Kidd Creek and Selbaie

More extensive rod mill feed water addition rate testing was prompted by the preliminary results from tests which were auxillary elements of the fines addition tests. The details of the tests performed at Les Mines Selbaie (survey no. 8) and Kidd Creek Mines Limited (survey no. 4) are described in Appendices I and J, respectively.

The first tests were performed at Kidd Creek in October, 1986. The rod mill was operated for approximately one hour after feed water adjustments to allow the mill to stabilize, and a sampling period of one hour was used. Rod mill discharge samples during each half of each of the 3 operating modes (D, E, and F), were collected separately so that operating stability could be verified. Bond rod mill laboratory work index tests were performed on each of the feed samples.

The rod mill was operated at two densities during the test at Selbaie performed in November, 1986. Sampling times and analysis procedures were identical to those applied during the tests at Kidd Creek. The results from both sets of testwork are also summarized in Tables 3-8 and 3-9.

3.3.5.2 General Conclusions Concerning Grinding Efficiency

Increased feed water addition rate has the effect of increasing rod mill grinding efficiency in both of these operations. At Kidd Creek, grinding efficiency improved by 8 percent when feed density was decreased from 81.4 to 77 percent solids by weight. At 72.2 percent solids, the operating work index improved (i.e., was lower) by 12.5 percent compared to the normal operating density. An eleven week test confirmed that the rod mill could maintain the same product sizing at approximately 12% less power draw by operating at lower density (Rumfeldt, 1987). At Selbaie, grinding efficiency improved by 13 percent by operating at 77.1 rather than 81.4 percent solids by weight in the mill feed (and discharge).

As discussed in section 1, the effect of feed water addition rate on rod mill efficiency has been reported by Klimpel (1982-83), who correlated the results to the mill discharge rheological characteristics. The probable importance of both the breakage environment and particle transport mechanisms in the rod mill is supported by both this study and Klimpel's findings.

3.3.5.3 Economic Analyses

Although the effect of lower operating density on grinding efficiency is quite clear from the above data, the effect it may have on steel consumption requires a much longer term test. Steel consumption in the rod mill tested at Kidd Creek was approximately 3 percent lower than the rod mill in a parallel circuit over a six week period, but this was definitely less than enough time for the rod charge to stabilize. Although a reduced charge level producing the same grind at lower density would tend to reduce steel consumption, this may be partially offset due to the impact conditions in the mill.

For the economic evaluation, let us assume that the increased efficiency at lower density is used to lower the power draw and charge level in the mill, while maintaining the same product size. At Kidd Creek, power consumption may then be reduced by approximately 8%, from 555 kw to 510 kw at the motor input (approximately 520 to 480 kw at the pinion). From Figure 3-6, the charge level could be reduced from approximately 38 to 33%, or a net reduction of approximately 13 percent in the total steel load. Conservatively, the reduction in steel consumption may be estimated at about 4%. This is about one-third of the reduction in the mill load, and one half of the reduction in energy savings. On this basis, the minimum yearly savings would be as follows:

a)	Power:	8% of 12.3 ¢/t	=	0.98 ¢/t
b)	Media:	4% of 29.8 ¢/t	=	1.19 ¢/t
		Total		2.17 ¢/t

2.17 ¢/t x 3240 t/day x 360 days/yr = $25,000/yr

At Selbaie, power consumption may then be reduced by approximately 13%, from 220 kw to 191 kw at the motor input (approximately 200 to 174 kw at the pinion). From Figure 3-7, the charge level could be reduced from approximately 42 to 33%, or a net reduction of over 20 percent in the total steel load. Conservatively, the reduction in steel consumption may be estimated at about 7%. This is about one-third of the reduction in the mill load, and one half of the reduction in energy savings. On this basis, the yearly savings would be as follows:

a) Power: 13% of 10.0 ¢/t = 1.30 ¢/t
b) Media: 7% of 17.7 ¢/t = 1.24 ¢/t
Total 2.54 ¢/t

2.54 ¢/t x 1650 t/day x 360 days/yr = $15,000/yr

3.3.6 Other Considerations and Continuing Testwork

3.3.6.1 Media Size

The ideal rod size, R, according to Allis-Chalmers experience in rod milling can be estimated by the following equation (Rowland, 1982).

$$R = 0.16\ F80^{0.75} \left(\frac{W.I. \times S.G.}{100\ Cs \times (3.281 \times D)^{0.5}} \right)^{0.5}$$

R = rod diameter in mm
F80 = feed 80% passing size in mm
W.I. = Bond rod mill laboratory work index
S.G. = specific gravity
D = mill inside diameter in meters

At Kidd Creek, the average feed size (F80) is 15,000 um and the average test work index is 17.6 kwh/t. The ore specific gravity averages 3.1, the present mill speed is 70.6% of critical, and mill inside diameter is approximately 3.0 m (9.8 ft.). Solving, R = 108 mm (4.24 in.). This is very close to the present rod sizing of 102 mm (4.0 in.). At Selbaie, the average feed size (F80) is 12,200 µm and the average test work index is 15.1 kwh/t. The ore specific gravity averages 2.76, the present mill speed is 67% of critical, and mill inside diameter is approximately 2.29 m (7.5 ft.). Solving, R = 84 mm (3.3 in.). Based on availability, either 89 mm (3.5 in.) or 76 mm (3.0 in.) rods may be selected, and both of these sizes have been used in this mill. From the amount of data generated to date, there is no indication that either of these rod sizes has any advantage in terms of grinding efficiency. However, larger rods would likely contribute to reduced steel consumption due to the reduced surface area of grinding steel (see Appendix A2).

3.3.6.2 Liner Condition

As discussed in section 3.3.2, the installation of new liners decreased the power draw capability of the Selbaie rod mill by approximately 9 percent. This can be attributed to the loss in mill volume, and would therefore be expected to be relatively less for the larger diameter mills at Kidd Creek. There is no indication that liner wear has any appreciable effect on grinding efficiency, or that slip between the shell and the charge occurs, even when the liners are nearly worn out.

3.3.6.3 Feed Size

According to Allis-Chalmers (Rowland, 1982), the optimum feed size, Fo in µm, for a rod mill can be estimated from the following equation (metric basis).

$$Fo = 16{,}000 \left(\frac{14.3}{W.I.}\right)^{0.5}$$

At Selbaie, the average work index is 15.1, and Fo = 15,570 um. Since the average feed size (12,200 μm) is less than this value, there is no oversize feed correction factor or associated inefficiency by this method of analysis.

At Kidd Creek, the average work index is 17.6, and Fo = 14,400 um. Since the average feed size (15,000 μm) is more than this value, there is an oversize feed correction factor (or associated inefficiency), as follows (Rowland, 1982).

$$EF4 = \frac{Rr + (.907\ W.I. - 7)\ (F80 - Fo)\ /\ Fo}{Rr}$$

Rr = reduction ratio

For the average observed reduction ratio for normal operating conditions of 10.2, EF4 equals 1.04. This indicates a possible inefficiency of approximately 4% due to oversized feed to the rod mill. Combined with slightly undersized rods, this could be a significant factor in overall rod milling efficiency.

3.3.6.4 Reduction Ratio

As described in the review in section 1, when the feed size is fixed, product size and reduction ratio reduce to the same parameter, and are closely related, as well, to the feed rate. The four may therefore be considered together as a general "mill work load" parameter.

According to Allis-Chalmers (Rowland, 1982), the ideal reduction ratio, Rro, for a rod mill of this configuration is given by the following equation.

$$Rro = 8 + \frac{5\ Lr}{D}$$

For the Kidd Creek mill, Lr = 4.7 m, and solving the above equation, Rro = 15.8.

The inefficiency factor for rod mill reduction ratio, EF6, is given by:

$$EF6 = 1 + \frac{(Rr - Rro)^2}{150}$$

The average reduction ratio in the Kidd Creek plant is actually 10.2, yielding a value of EF6 = 1.21, or an estimated inefficiency of 21% by this method.

However, this inefficiency is not born out by the operating data. Although the data are insufficient to be conclusive, the higher mill work loadings, as measured by the product of tonnage rate times test work index divided by mill power for all the normal operating conditions actually indicate higher operating efficiency, as summarized below. Note that a similar inefficiency factor for low reduction ratio (17%) was calculated for the Selbaie rod mill. See Appendix K for a description of the final rod mill survey (no. 9) carried out at Selbaie.

Note that although the comparative efficiencies between operations are subject to much more error than for relative efficiency measurements for each, the Selbaie rod mill does appear to be operating at higher efficiency, on average. Although overall reduction ratios and loadings are similar, both feed and product sizes are approximately 25 percent

Table 3-15. Rod Mill Work Loading and Work Index Efficiencies

Run No.	Work Loading $\frac{\text{(Feed Rate x WI)}}{\text{Mill Power}}$		Efficiency (%)
KCM 2	5.29		87%
KCM 1	5.17		93%
KCM 4D	4.60		85%
KCM 3A	4.52		84%
		KCM Ave:	87%
Selbaie 1	5.21		106%
Selbaie 2	5.16		105%
Selbaie 9	5.13		108%
Selbaie 8A	4.83		97%
Selbaie 4A	4.39		91%
		Selbaie Ave:	101%

greater for the Kidd Creek rod mill. This also suggests that there may be inefficiency associated with oversize feed to the Kidd Creek rod mill.

3.3.6.5 Continuing Testwork

Following a speed-up of one or more of the rod mills, rod consumption should be monitored over a period of 6 months to one year to define more precisely the economic benefits of operating at higher mill speed. A similar test period is needed to assess the effect of operating at reduced mill density.

A short term test to assess the effect of the addition of fines from primary cyclone overflow to the rod mill feed should be carried out. If this has a positive effect on rod mill grinding efficiency, the benefits in rod milling will have to be weighed against the possible negative effect of returning these fines to the head of the primary ball milling circuit.

The possibility that the coarseness of the feed size leads to inefficiency in the Kidd Creek rod mills should be investigated further. The program there may include tests at increased or decreased feed size and/or tonnage. The possibility of using slightly larger rods may also be considered.

3.3.7 Summary and Conclusions

1. Estimated savings in grinding media consumption of at least $35,000.00 per year can be achieved through speeding up the rod mill at Kidd Creek. The purchase and installation of a new pinion to achieve the speed-up would cost approximately $40,000.00.

2. There is no indication that the addition of cyclone feed fines to the rod mill feed has any effect on grinding efficiency or economy.

3. The operation of the Kidd Creek rod mill at a reduced feed density of 77% (versus the current 81%) solids by weight improves grinding efficiency by approximately 8%. There is no cost associated with this change. If implemented so that the current mill product size is maintained, savings in grinding media and power consumption will conservatively total approximately $25,000.00 per year. At Selbaie, grinding efficiency improves by approximately 13% with the same change in densities, and savings here would total

approximately $15,000.00 per year. Further testwork is needed to determine optimum mill density.

4. Oversize feed appears to be a likely cause for some inefficiency in the Kidd Creek rod mill performance, and should be the subject of further investigation.

3.4 DISCUSSION

This investigation has attempted to present a more concise quantitative analysis of the error associated with operating and test work index determinations than has previously been undertaken. In doing so, it was found that experimental error from a number of sources may be significantly reduced, for example, through careful power draw measurements, the use of the same sieves for all size analyses, and standardization of grindability test procedures. Several failed attempts at plant sampling were also very instructive, particularly with respect to the need to establish a means of verifying stable circuit operation during plant testwork.

In general, the testing and analysis procedures can be considered successful for the specifically defined task. It could be reasonably concluded from one set of two or three work index efficiency comparisons at each plant that cyclone feed fines addition was unsuccessful in improving rod mill grinding efficiency. It was also shown from one set of tests that feed water addition rate could have a significant effect on rod milling efficiency in these mills. A subsequent month long test confirmed the results at the Kidd Creek concentrator (Rumfeldt, 1987 and Roberts, 1988).

However, the limitations of this type of testwork for circuit performance characterization, and the specific results achieved here, must also be recognized. The range of feed characteristics and operating conditions encountered was limited in these tests. Over a broader range of feed and product size distributions, the Bond relationship could no longer be reasonably assumed to remain valid for overall grinding efficiency comparisons. Furthermore, while comparison of relative circuit efficiencies from a pair of surveys just before and after a change (following the return of circuit stability) appear valid within the mentioned statistical limits, individual surveys performed at substantially different times (weeks or months apart) would not necessarily yield equally comparable results. Variations may occur due to an important variable that shifted over the longer period of time but was neglected during the testwork (for example the effect of slurry temperature noticed by Klimpel, 1982-83). Thus, even though accurate measurements may reduce the amount of testwork required, they do not eliminate the need for longer term or repeated tests when knowledge about the process is incomplete.

Finally, while the empirical approach can provide useful quantitative data to help improve grinding equipment performance, it reveals little about fundamental mechanisms of the process, the understanding of which may be the most important means for process improvements in the long term.

CHAPTER 4

CLOSED CIRCUIT BALL MILLING

4.1 INTRODUCTION: IMPORTANT ASPECTS OF BALL MILL CIRCUIT PERFORMANCE

Closed circuit ball milling performance evaluation involves several more dimensions than must be considered for rod milling. Bond work index analysis is also used here as the overall measure of circuit grinding efficiency. As discussed by Laplante et al (1988), the relative error in ball mill work index efficiency estimations will be no greater than that for rod milling. However, the complexity of the closed ball mill circuit is such that the effects of individual design and operating variables on overall circuit efficiency cannot be analysed by either the Bond approach alone, or by available circuit simulation techniques in many cases. An intermediate level of ball mill circuit performance characterization that may provide a useful step for relating these variables to overall circuit efficiency is first presented in this chapter.

Another aspect of ball mill circuit performance is the important role that it plays as the final feed preparation step for the mineral separation process that follows. Interfacing of grinding and flotation operations is therefore also examined here in some detail.

Finally, process control of the ball mill-classifier circuit is a far more complex issue than for the open circuit rod mill. The objectives for ball mill circuit control stem from both the need to achieve the desired feed characteristics for the subsequent process, as well as to maintain the design and operating conditions which have been selected to provide efficient grinding circuit size reduction performance. As well, the introduction of a process control system is an important circuit design charge which itself may be evaluated in terms of how circuit efficiency is affected. Process control practices in conventional ball milling circuits are therefore also reviewed before evaluation of the case study ball milling circuits.

4.2 SIZE REDUCTION PERFORMANCE CHARACTERIZATION OF CLOSED CIRCUIT BALL MILLING

4.2.1 Relating Design and Operating Variables to Overall Circuit Performance

Table 4-1 is a summary of major design and operating variables in closed circuit ball milling for an operation with a specified feed size, product size, and tonnage handling requirement (McIvor, 1983). All except the ball mill dimensions are normally considered to be subject to modification, and continual testing of alternatives is a common practice in many plants.

Steady state modelling provides the plant engineer with performance parameters which can be calculated and compared from sets of plant operating test data. When a design or operating change is made to a circuit (or in a parallel one), Bond work index analysis provides a relative measure of the overall circuit efficiency and economy. Population balance

Table 4-1. Major Design and Operating Variables in Closed Circuit Ball Milling

Ball Mill Dimensions	° Inside effective working length ° Inside average diameter
Circuit Configuration	° Single mill ° Series grinding mills ° Parallel grinding mills
Mill Speed	° R.P.M. or percent of critical
Mill Liner Design	° Average thickness ° Lifter height and shape ° Spacing or number of rows ° Wear profiles ° Material
Media Utilization	° Make-up size(s) ° Shape ° Charge level ° Material
Feed Preparation	° Feed size distribution
Classification System Design	° Number and dimensions of classifiers ° Pumping equipment ° Classifier feed conditions (pressure, flowrate, feed water addition) ° Configuration (normal or reverse closed circuit, number of stages)
Mill Operating Density	° Feed water addition rate
Process Control	° Automatic control system ° Manual control practices

modelling parameters provide more detailed technical characterization of ball mill and classifier performance, and are more directly related to specific circuit design and operating variables. Thus, it can be seen that a combination of the two systems could potentially be useful to evaluate the technical and economic effects of a change to the circuit.

The above describes the use of steady state modelling for performance parameter estimation. To be used in a predictive manner all the significant circuit equipment dimensional and performance characteristics, ore and slurry characteristics, and the interactions among them, along with the boundary conditions created by operating constraints, must be identified and characterized with sufficient accuracy. This is the task of steady state circuit simulation.

It can be said that the purpose of any process modelling system is to establish cause and effect relationships between specific design and operating variables and the performance objectives of the circuit, whether used for parameter estimation or in a predictive manner. In terms of maximizing grinding efficiency, the objectives of either of the two grinding circuit modelling systems being discussed here are essentially the same. Specific objectives of a finer grind, increased tonnage, or reduced grinding power consumption on the same feed can all be reduced to improved work index effeciency. The fundamental nature of the problem of ball mill circuit steady state modelling in this context is thus depicted in Figure 4-1.

As discussed in chapter 1, both Bond work index analysis and population balance modelling often fail to provide a satisfactory link between the two levels of process performance characterization. As part of an experiment in the application of value engineering techniques (Miles, 1972; Wales and Pfeiffer, 1986) to grinding process modelling, two

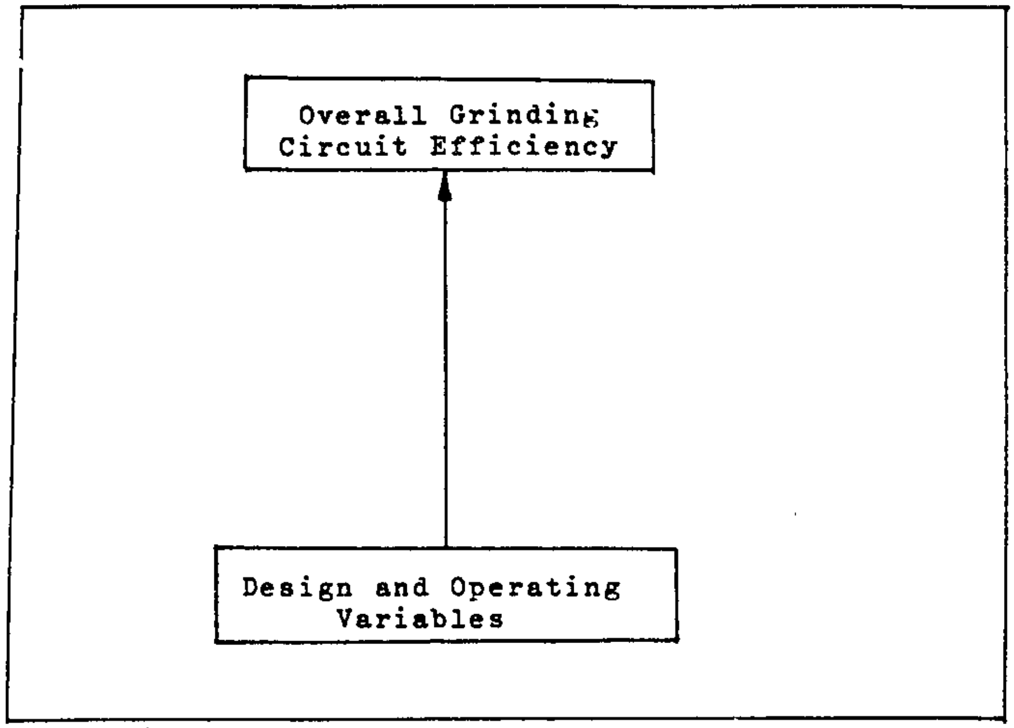


Figure 4-1. Relating Ball Mill Circuit Design and Operating Variables to Overall Grinding Circuit Efficiency.

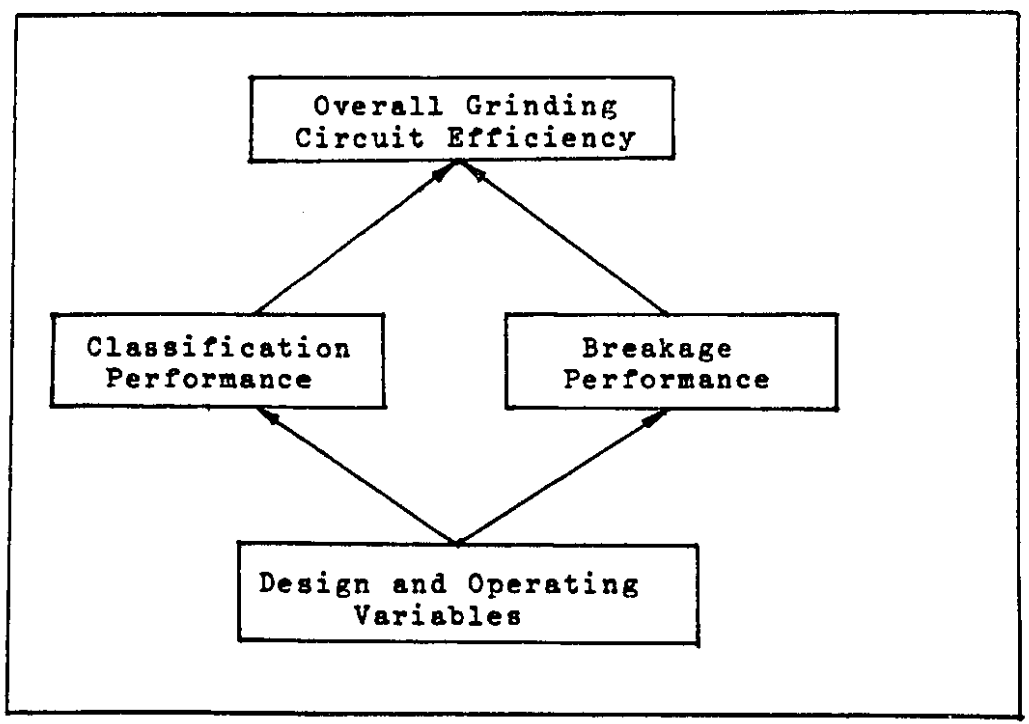


Figure 4-2. Relating Ball Mill Circuit Design and Operating Variables to Overall Grinding Circuit Efficiency through Classification and Breakage Performance.

primary functions of the ball mill-classifier circuit were identified. In terms of a specified circuit product size which defines "coarse" or oversize material and "fines" or undersize material, these primary functions are (a) breakage of the coarse material and (b) removal of the fines. This suggests the intermediate step of ball mill circuit performance characterization depicted in Figure 4-2. It is proposed that it may be useful to relate detailed design and operating variables to the primary circuit functions, which although related, are separate and distinguishable. If each can be quantified with suitable performance parameters, then either or the two together may be correlated to overall circuit efficiency, and hence used to link individual design and operating variables to overall circuit performance. Ball mill circuit classification system performance is considered here first because it provides the basis for subsequent characterization of ball mill breakage efficiency.

4.2.2 Review of Classification Effects in Closed Circuit Ball Milling

4.2.2.1 Introduction

It can be said that the function of the grinding circuit is to carry out the breakage necessary to suitably prepare material for the upgrading process (Coleman, 1980). This definition of purpose implies that the removal of finished size material is an integral part of proper circuit operation. The capability of the grinding circuit to perform its task effectively can then be seen to depend on two processes that are taking place:

a) size reduction of the coarse material; and,

b) removal of fines from the circuit.

Lack of removal of fines from the circuit creates two apparent problems. First, fines are further ground (over-ground) into extreme fines (or slimes) which may have a number of deleterious effects on grinding itself, as well as on down-stream recovery processes. Secondly, fines occupy valuable mill volume, which is then unavailable for grinding of coarse material. Ultimately, without fines removal, the circuit will be unable to grind, and comminution energy will be wasted.

A basic constraint on the size distribution of product material from a grinding circuit is imposed by the range of sizes produced by each single comminution event. Crabtree et al (1964) quantified Gaudin's (1939) observation that the impact or crushing type of mechanism, interpreted as nipping of particles between the grinding media, is dominant in normal ball milling. A certain amount of abrasive action also takes place due to the exposure of rock surfaces in the mill environment. These mechanisms generate fines over the whole size spectrum. The sizes of particles produced from single breakage events, together with the fines present in the new feed to the circuit, define the product size distribution with the minimum of extreme fines from a circuit with an idealized classification system where any particle smaller than the cut-off size is immediately removed (Austin and Perez, 1977).

4.2.2.2 The Fines Removal System

Analysis of classification effects in grinding circuits requires consideration of how the equipment in the circuit performs in terms of both size reduction and fines removal. While it is only inside the mill that breakage takes place, fines removal relies on transport of material through the mill and to the classifying device, as well as the classifier's suitable sizing performance. Any circuit design or operating variable which affects material time of exposure to grinding forces before presentation to the classifying device is therefore also relevant to fines removal. Factors relevant to the performance of the fines removal system will therefore include both:

a) the mill residence time characteristics, as determined by slurry throughput rate and effective mill volume, as well as variations associated with forward and backward mixing; and,

b) the performance of the classification equipment.

<u>Mill Residence Time</u>

The shorter the average retention time in the mill before presentation to the classification device, the closer to ideal is the fines removal system. The mean residence time of a slurry passing through a mill is the volume of slurry hold-up in the mill divided by the volumetric slurry throughput rate. Any grinding circuit variable which affects mill volume or throughput rate will therefore have an effect on the classification system. From the approach of conventional circuit design, the most important of these will include:

a) the grinding unit size (or sizes) and relative dimensions used to achieve the total power draw required for the circuit tonnage and grind;

b) the slurry density; and

c) the circulating load ratio.

Circuit capacity is approximately proportional to mill power draw. Ball mill power draw is directly proportional to mill length, and, theoretically, to the diameter raised to the 2.5 power, or empirically, to the 2.3 power, according to Bond (1961). Because mill volume is proportional to the diameter raised only to the power 2, this means that, at fixed circulating load ratio, the mean residence time decreases with increasing mill diameter by the factor of the diameter to the power 0.3 to 0.5.

Decreasing the concentration of solids in the slurry mixture will increase the slurry volume for a fixed dry solids tonnage rate through the mill. Thus, for otherwise similar operating conditions, mean residence time in a given mill decreases as mill feed slurry dilution increases. This observation must be qualified by assumptions of:

a) unappreciable changes in total mill holdup volume with variations in the slurry consistency and feed rate; and,

b) unappreciable difference between mill holdup slurry density versus that of the feed and discharge.

The circulating load ratio is defined as the amount of solid material that is directed back into the circuit in the classifier oversize stream, divided by the circuit new feed (or final product) solids tonnage rate. The advantage of operating ball mills in closed circuit with high circulating loads has been reported in innumerable instances (Dorr and Anable, 1934; Taggart, 1945). A recent example is given in Table 4-2 (McIvor, 1984a). In 1925, Davis reported the relationship between circulating load and circuit productivity in a three foot (0.9 meter) diameter, 4 kilowatt test mill, as reproduced in Figure 4-3. He demonstrated that the circuit capacity increased rapidly to over double the open circuit production rate with a circulating load of approximately 200%, with a gradual levelling off of capacity at higher figures.

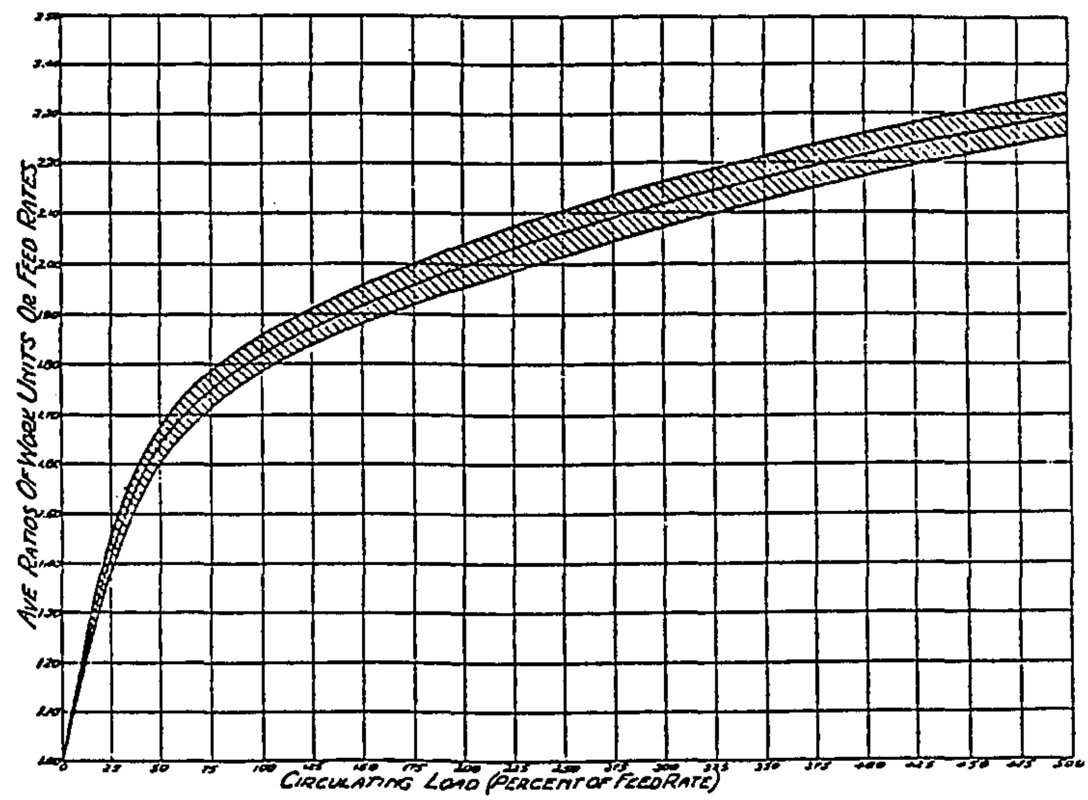
Coghill and DeVaney (1938) provide an excellent explanation for the nature of this relationship, citing the initial rapid increase in "useful grind" (as opposed to overgrinding of fines) as a result of the relatively large increase in mill loading rate (corresponding with a decrease in mean retention time) with the first 100% of circulating load, then increasing proportionately less with each additional 100% of recirculation. A better match of a narrowed mill hold-up size distribution with the size of the grinding media is also noted. Mill and classifier overloading as well as material handling considerations eventually impose practical limitations, causing the curve to level off.

Table 4-2. Circuit Productivity at Different Circulating Loads (Les Mines Belmoral Ltee.)

Survey Date	Tonnage (M.T.P.H.)	Circuit Product %-200M	80% Passing	Circulating Load	Power kwh/t	Operating Work Index	Relative Efficiency
3/15/83	41.0	65%	112 µm	250%	11.21	17.37	(Base)
6/03/83	42.8	66.4%	110 µm	298%	10.74	16.42	+ 5.5%
6/21/83	41.8	77.1%	85 µm	411%	10.99	14.00	+19.4%

NOTES:

1. Grinding circuit feed size: F80=1115 µm.
2. Total ball mill grinding power available: 459 kw.
3. Operating work index calculated using Bond formula with ore grindability and feed size assumed constant.



CURVE SHOWING RELATION BETWEEN CIRCULATING LOAD AND WORK DONE IN BALL MILL.

Figure 4-3. The Davis Relationship (Davis, 1925)

Basically, as average mill retention time decreases, fines are more quickly removed from the circuit. Overgrinding (generation of more extreme fines) is reduced, and more space is made available in the mill for coarse material to be ground. In practice, approaching ideal classification by minimizing average retention time is limited by a number of factors, to be discussed later.

Studies utilizing tracers in continuously operating mills have demonstrated that material does not pass steadily through the mill (analagous to plug flow), but rather is subject to considerable forward and backward mixing, producing a typical "residence time distribution" similar to that shown in Figure 1-3 (Austin et al, 1984). Effective fines removal is obviously disrupted as a result of the greatly extended retention time for some of the material.

The complex system of mass transport mechanisms in overflow ball mills has been described by Hogg (1984). Of interest are several observations relevant to internal mill classification:

a) Size dependent particle classification does take place along the axis of the mill.

b) Size dependent particle classification also takes place at the mill outlet.

c) Water mean residence time is somewhat less (10-15%) than that of the solids.

All the above tend to favor the faster movement of finer particles through the mill compared to the coarser ones. Note the last point also leads to the interesting conclusion that slurry density inside the mills is higher than that of the feed and discharge (Austin et al, 1984). Studies by Austin et al (1984) have also shown that residence time distributions for different mill sizes do not vary considerably when tracer discharge concentration and time are expressed in a dimensionless form, although the degree of mixing was seen as somewhat sensitive to slurry density.

Classifier Performance

For the purpose of this discussion, "fines" may be taken as any material smaller than a specified product size, and "coarse" as any material that is larger. The target 80 percent passing size is convenient for this purpose, as this figure is often used to express the grind size objective of a milling circuit. The basic function of the classification equipment then is to remove the fines from the circuit and to return the coarse material to the mill. Simultaneously, it serves as the means by which large flows may be recirculated so that mill throughput rate can be increased, and mean retention time decreased. Larger capacities may be handled by increasing equipment size when cut size is basically equipment size independent (eg., vibrating screens with fixed screen opening size), or by increasing the number of operating units when cut size is equipment size related (eg., hydrocyclones). As well, in the context of wet grinding, these tasks must be performed while the desired densities in the classifier product streams are maintained.

The classifier selectivity or classification characteristic curve is used to graphically represent a recovery to the coarse product stream of each mesh particle in the feed. The corrected recovery curve is generated by mathematically eliminating the fines in the oversize product stream that report there solely because of the water bypass fraction. Typical actual and corrected selectivity curves for a hydrocyclone are shown in Figure 1-4, (Austin et al, 1984) and demonstrate that considerable misplacement of fine material to the oversize takes place in this type of a classification device.

Design and operating parameters which affect the cut size and shape of the separation characteristic curve of hydrocyclones are reflected in hydrocyclone models such as that of Plitt (1976) and Lynch (1977). Use of such models for defining hydrocyclone performance criteria and dimensional requirements for given circuit performance specifications will be discussed later.

Circuit Classification System Efficiency

It can be seen that the overall effectiveness of the fines removal system is dependent on the length of time to which material is exposed to grinding forces before presentation to the classifier, and the capability of the classifier to selectively remove the fine particles and return the coarse ones to the mill. In relation to the specific task of size reduction, it also can be seen that an ideal, immediate and perfect sorting action would result in a circuit product with a minimum of fines (dependent only on the material breakage function), and avail the maximum possible mill volume for the breakage of coarse particles. The fraction of fines in the mill hold-up may therefore be taken as a direct indication of the lack of the system's fines removal effectiveness.

It is proposed that the fraction of the total mill solids inventory which is coarser than some defined cut-off product size is a single parameter which describes in quantitative terms the overall classification system efficiency of the ball milling circuit. The inventory which is finer than the specified product size of the circuit represents wasted potential production volume. Since it is the steel charge which delivers grinding energy to the solids, it also represents the fraction of the charge volume and mill power draw which is wasted. The remainder is fraction of the effective grinding charge and energy (i.e., being delivered to material which should be the target of grinding forces), and thus also represents the overall classification efficiency of the grinding circuit. As they are complimentary, one may consider either the fines or coarse mill inventory as desired.

Although an estimate of the average mill contents size distribution could be derived from a more detailed ball mill model, a simple relative fines inventory term has been selected. It is the average of the amount of fines in the ball mill feed and discharge size distributions. Although a proven relationship between mill contents and the measured feed and product size distributions has not been established, the general relationship between mill production capacity (to a given product size) and these distributions are supported by evidence from a variety of sources, including pilot plant testwork, grinding circuit simulation, and operating plant performance, as shown by the following examples.

Coghill and deVaney (1938) conducted experiments in a pilot plant ball mill closed circuited with a rake classifier, operating the circuit with a wide range of circulating load ratios. They noted that, "It held true that the percentage of decrease in mill capacity was almost identical with the percentage of finished material in the composite feed. For example, when forty percent of the finished material was returned it amounted to 19 percent of the composite feed and the reduction in capacity was 21.5 percent."

Kelsall et al (1969) were among the earliest to conduct experiments with a computerized grinding circuit simulator, testing the general effect of various circuit design and operating variables, including circulating load ratio. "All product size distributions had the same 80 percent passing size, and fresh feed rate was altered to give a range of circulating loads .... It was shown that the ratio of the closed to open circuit feed rate, which may be taken as a measure of the increased efficiency of closed circuit operation, was proportional to the fraction of the mill contents which was coarser than the classifier separation size."

Laplante et al (1986) have demonstrated that the cumulative specific breakage rate (i.e., the specific disappearance rate of material of the complete size distribution down to a defined particle size) remains virtually constant over a reasonable range of operating conditions, including variations in the mill feed size distribution itself. This means that the production rate of material finer than a specified size is directly proportional to the percentage of material in the mill feed greater than this specified size, which reflects the coarse solids inventory.

Figure 4-4 shows computer simulation generated mill feed and product size distributions in circuits producing a similar product (80% passing) size, but at different estimated circulating load ratios. These size distributions are very similar to those reported from plant circuits operating at nearly the equivalent circulating load ratios (Picket, D.E., 1978 (b); Gould, 1976). It is significant that the increase in the relative mill coarse solids inventory (plus 150 mesh), of approximately 20 percent is nearly equivalent to the increase in grinding circuit capacity (or efficiency) associated with the same circulating load ratios as shown by the Davis relationship (see also Figure 4-3).

4.2.2.3 Implications for Circuit Design and Operation

<u>The Effect of Circulating Load</u>

The importance of a sufficiently high circulating load ratio is clear from the evidence presented above. Note,

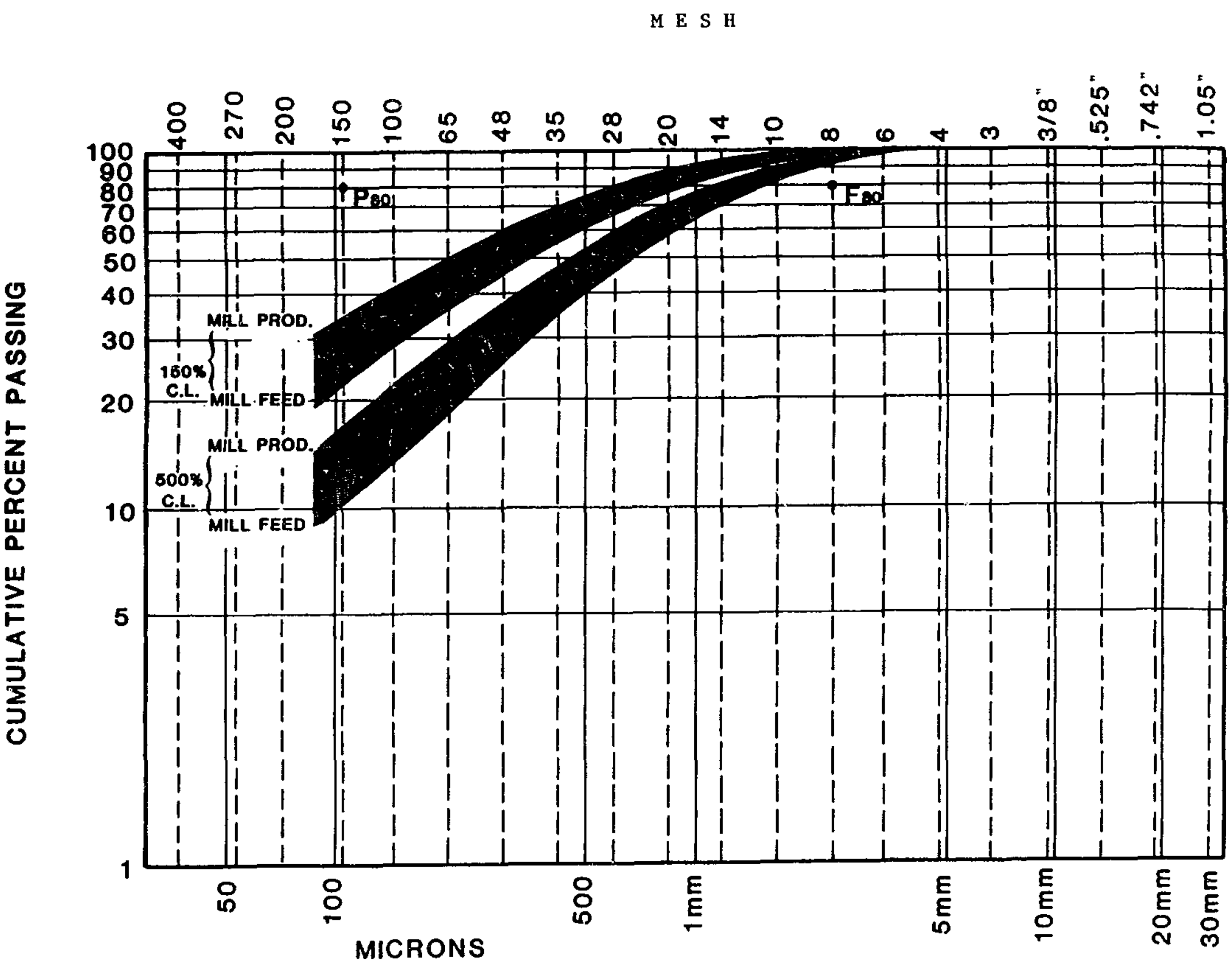


Figure 4-4. Ball Mill Feed and Product Size Distributions at Different Circulating Load Ratios.

however, that the Davis (1925) relationship most assuredly peaks and turns downward with higher and higher circulating loads. This is due to a number of factors.

At the classifier:

a) For the same overflow and underflow densities, classifier feed density must increase to satisfy the water and solids material balance. For a cyclone, this means loss of sharpness of separation, and a higher fraction of water recovery to underflow, and hence higher bypass of fines to the coarse product stream.

b) As Coghill and Devaney (1938) point out, the classifier is faced with handling a leaner and leaner feed (that is, a feed with less and less product size present in it), making selective removal at the fines more difficult.

In the mill:

c) An overfilling effect may occur, as suggested by Austin et al (1984).

d) As flow rates increase, the media will tend to be washed through the mill and discharged with the slurry flow.

The Effect of Mill Dimensions

For a fixed total power draw and corresponding fixed circuit tonnage (at a fixed circulating load), selection of the circuit configuration or dimensions of the mill (or mills) has a direct effect on the fines removal system. Power draw and throughput increase proportionally with mill length, as does the mill volume, and therefore mean residence time and fines removal effectiveness are not affected. However, slurry flow rate does increase with mill length, and contributes to a limiting factor equivalent to d) mentioned above with respect to circulating load. As mill diameter increases, mill power draw (and throughput rate) increase faster than total mill volume. The result is both decreased mill mean residence time and increased slurry flow rate.

Decreased retention time and increased flow rates for larger ball mills (of roughly the same proportions) therefore produce similar effects as increasing circulating load for a fixed mill size. Fines removal effectiveness increases, in this case without the detrimental effects at the classifier. However, overfilling and, as operating plant observations and media wear rates studies confirm (Lorenzetti, 1984), loss of small grinding media are more prevalent. It can be concluded that the curve for large mills will peak at a lower value of circulating load ratios than that for smaller ones performing the same grinding task.

The Effect of Classifier Performance

The effect of the sorting efficiency of the classification equipment was investigated at length by Hukki (1967, 1968), who presented theoretical analyses and numerous batch grinding test results that indicated the potential for 40 to 50 percent improvements in circuit productivity with moderate improvements in classification system sharpness of separation. He later proposed the use of a special hydraulic cone classifier for second stage treatment of cyclone underflow for closed circuit ball milling applications (Hukki, 1967, 1968, 1977 and 1979).

Unfortunately, the magnitude of the early estimates of improvements in circuit efficiency could not be achieved in actual plant operations (Hukki and Heiskanen, 1981). The assumption of plug flow through the mill, analagous to batch grinding in terms of the mill residence time characteristics, was one shortcoming of the earlier studies. As well, although the reports were not explicit with respect to size distribution information, the material breakage function may also have been extrapolated to yield less production of fines than material breakage characteristics actually impose. Despite these limitations, Hukki's studies did draw attention to the importance of efficient classifier performance in milling circuits.

Recent operating plant case studies (Tarvainen, 1980; Rao et al, 1982; Kanna, 1982) have reported improvements in the range of 5 to 15 percent with multi-stage classification, figures in line with results from simulation studies (Herbst and Fuerstenau, 1980; Rogers et al, 1981). Kelsall et al (1969) reported an overall increase in circuit efficiency of 7 percent when the assumption of perfect classification was substituted for actual cyclone performance, and concluded that "deviation from perfect classification did not impose much limitation on the grinding efficiency of the particular circuit."

The Hydrocyclone in Closed Circuit Grinding

In its relatively short history, the hydrocyclone has become the prominent classification tool for a broad range of mineral processing applications. Its mechanical simplicity, adaptability, and high capacity at minimal energy requirements have all contributed to its widespread acceptance.

As described previously, the nature of the ball milling process is such that a high mill throughput rate, provided by a high circulating load, is essential for efficient circuit performance. At the same time, the overall circuit performance is relatively much less sensitive to the classification equipment sorting efficiency. The hydrocyclone is thus an ideal tool for most grinding circuit applications.

Minimization of the bypass fines fraction to the hydrocyclone underflow by operating at the maximum underflow density (within the limitation of maintaining a stable "umbrella" type, non-roping, apex discharge) appears compatible with optimum mill operating density conditions. As described by Katzer et al (1981), mill operating density should be a maximum, to a point where negative viscosity effects just start to come into play, for maximum grinding efficiency. In fact, for optimum rheological conditions at the feed end of the mill, it has been observed that the mill feed may actually require the presence of a certain amount of very fine material (Klimpel, 1982-83; Laplante, 1983).

Limiting the feed density for efficient hydrocyclone operation, in conjunction with the desirability of high mill operating density, normally calls for the addition of dilution water to the cyclone feed. This may preempt the use of standard hydrocyclones in favor of screening devices where high circuit product density must be provided, and a dewatering or thickening stage is not feasible before the subsequent treatment process (Kirk, 1984).

Hydraulic classifiers will preferentially send higher specific gravity particles to the oversize product stream (for example, see Laplante and Finch, 1984). This will lead to build up and preferential grinding of the heavier minerals, which may have serious implications on the circuit performance, as well as the recovery process. Once again, a fixed-opening screening device may be called for instead of or in conjunction with the hydrocyclone classifier.

Because the hydrocyclone cut size is dependent on its dimensions, simultaneous capacity and classification requirements may exclude the use of hydrocyclones in applications where the total process flow is small, such as in a pilot plant or very low tonnage operations. Hydrocyclone dimensions must be selected on the basis of the cut size requirement, with capacity as a secondary, although possibly limiting, consideration (McIvor, 1984 b).

4.2.2.4 Summary and Conclusions

The fines removal effectiveness of ball milling circuits is closely related to the residence time characteristics of the mill, as well as the performance of the classification equipment. Material breakage characteristics and lengthy retention time for at least some of the solids limit the extent to which the classification characteristic of the equipment influences the overall fines removal process and the grinding efficiency of the circuit.

Any parameter which influences the material residence time in the circuit has an effect on the fines removal effectiveness. These include the selection of mill dimensions and circuit configuration to achieve the required power draw, as well as the design circulating load, slurry density, and the classification equipment performance.

It is proposed that relative overall ball mill circuit classification system efficiency can be quantitatively characterized by the fraction of coarse material (i.e., larger than some defined circuit product size) in the mill solids inventory, and that its value can be estimated from the mill feed and product size distributions.

Maximization of the coarse inventory is a clear objective for classification system improvements. Changes to the circuit size distributions through classification equipment adjustments then provides a direct means for classification system and overall grinding circuit efficiency improvement. Example calculations of ball mill circuit classification efficiency, along with an example of its use in economic optimization of classification system performance, are given in section 4.5.

4.2.3 Separate Overall Classification and Breakage Parameters for Closed Circuit Ball Milling

4.2.3.1 Ball Mill Circuit Size Reduction Performance

For a given ball mill circuit, the difference between the amount of product size material in the circuit product less the amount in the feed gives the production rate of new "product size" material. The "specific production rate" can then be defined as the production rate of new material per unit of work applied, and can be seen to reflect both the overall efficiency of the size reduction process, and the grindability characteristic of the ore over the size reduction range.

It can be stated that:

$$\text{Mill power draw} \times \text{Power specific production rate} = \text{Production rate of new product size material.}$$

Note that this relationship applies equally to the ball mill circuit and the ball mill itself.

The effective portion of the work being applied is the total times the fraction of coarse mill inventory, or the so defined circuit classification efficiency. We can also define the "specific grinding rate" as the specific production rate divided by the effective mill power, as follows:

$$\text{Specific Grinding Rate (SGR)} = \frac{\text{Production rate of new product size material}}{\text{Effective mill power draw}}$$

$$= \frac{\text{Production rate of new product size material}}{\text{Total mill power draw x Mill coarse solids inventory (the classification system efficiency)}}$$

The specific grinding rate is also the rate at which "plus" product size material in the mill becomes "minus" product size material. It reflects both the suitability of the mill environment to cause breakage and the grindability characteristic of the ore. To arrive at a term which reflects only the efficiency imposed by the mill environment, we must factor out the grindability of the ore, such as the net grams per revolution measured in a Bond work index test. This will yield the specific grinding rate in the plant ball mill relative to the specific grinding rate in a standard test mill, and may be termed the "grinding rate ratio".

$$\text{Grinding Rate Ratio (GRR)} = \frac{\text{Plant Specific Grinding Rate}}{\text{Test Specific Grinding Rate}}$$

The grinding rate ratio is a measure of the relative size reduction performance of the plant mill compared to the size reduction performance of the test mill. It may be considered dimensionless because each revolution of a test mill requires a fixed amount of power. It is not directly related to the classification system efficiency of either the plant or test circuit as (a) the plant specific grinding rate is based on breakage of only the coarse inventory, and (b) the classification system efficiency in the test mill circuit is, or can be held, constant. It is therefore proposed that the grinding rate ratio is a direct indication of size reduction efficiency.

4.2.3.2 The Ball Mill Circuit Functional Performance Equation

It is proposed that the above described parameters for system size reduction efficiency and for classification system efficiency factor the overall task of the ball mill circuit into its two distinct functional objectives, namely, fines generation and fines removal. The effect of design and operating variables can be studied on each separately, and when the product of the two is maximized, maximum overall circuit efficiency will be achieved.

Note that the power draw of the mill times the system classification efficiency times the specific grinding rate yield the circuit fines production rate.

Mill Power Draw	x	Classification System Efficiency	x	Power Specific Grinding Rate	=	Production rate of new product size material.

This may be termed the ball mill circuit functional performance equation. Example calculations of these parameters are given in section 4.5.

4.3 REVIEW OF INTERFACING OF GRINDING AND FLOTATION OPERATIONS

4.3.1 Introduction

Interfacing of size reduction and mineral separation operations is an important aspect of plant operation. Unfortunately, however, operators must often consider the two areas with less than concisely defined parameters which relate the operations, so that interfacing of the two is largely a matter of the experience and judgement of the plant personnel.

The mineral separation process is the primary revenue generating operation in the concentrator, and grinding is the primary feed preparation process for mineral separation (Coleman, 1980). As such, consideration of these two is paramount to the first level of plant optimization.

This review attempts to establish a basic framework for evaluation and optimization of the grinding and separation interface for mineral processing operations. Although froth flotation is specifically considered, some of the principles would apply to other mineral separation processes.

4.3.2 Characterization of Grinding Circuit Product/Separation Circuit Feed

4.3.2.1 General Characteristics

The basic characteristics of grinding circuit product/separation circuit feed may be summarized as show in Table 4-3A. Other characteristics commonly referred to which can be derived from the given basic characteristics are listed in Table 4-3B. Twelve basic and seven derived characteristics are identified. All but two are affected by size reduction processes, and several are significant factors relative to flotation circuit performance.

4.3.2.2 Mineral Liberation

Mineral liberation is obviously a fundamental requirement for separation to take place, and therefore places a limit on the ultimate performance capability of any separation process.

Table 4-3A. Characteristics of Grinding Circuit Product/Separation Circuit Feed

BASIC characteristics:

1. solids flowrate (tonnage) *

2. solids/water ratio *

3. product size (K80) and size distribution *

4. particle composition (degree of liberation, grade distribution of locked particles) in each size class *

5. S.G., and other physical/chemical properties of each mineral phase

6. texture (degree and type of mineral dissemination; shape, size, and orientation of mineral grains) of locked particles

7. particle shapes *

8. state of agglomeration/dispersion *

9. temperature *

10. chemical/physical additives or residues (wood plastic and metal fibres, grinding media, grinding aid, flotation reagents, etc.) *

11. water (in slurry) qualities (pH, Eh, solutes, etc.) *

12. solids toughness (grindability, brittleness, abradability) *

* Directly controlled or influenced by comminution processes.

Table 4-3B. <u>Derived Characteristics</u>

	(derived from)
- total mass flowrate **	(1, 2)
- chemical/physical properties ** of particles	(3, 4, 5, 6, 7, 8)
- overall mineralogical composition *	(3, 4)
- particle density distribution, and overall solids specific gravity	(3, 4, 5)
- mineral distributions by sizes (mineral size distributions when 100% liberated) **	(3, 4)
- slurry rheological properties	(2, 3, 4, 5, 7, 8, 9, 10)
- slurry abrasiveness, corrosiveness	(2, 3, 5, 7, 9, 10, 11, 12)

** Significant factor in flotation circuit performance.

This is illustrated in Figure 4-5. The feed to the separation process is composed of two minerals, A and B, distributed among particles containing from 100%A to 0%A (Figure 4-5a). Even a perfect separator, one which took only 100%A particles first, followed by 95%A/5%B particles and so on, could do no better than the grade/recovery relationship shown in Figure 4-5b. This is the limit to separation imposed by the state of liberation of the feed.

In order to be of practical merit in terms of the interfacing problem, it is necessary to relate the size of solid particles to the degree of liberation of the mineral grains. Three of numerous proposed liberation models from different eras are summarized in Table 4-4 (Gaudin, 1939; Weigel and Li, 1967; Meloy and Gotoh, 1985). These offer some useful, though largely qualitative observations, such as significant liberation of the abundant (usually gangue) phase at coarse particle size, and diminishing returns in terms of liberation of the scarce phase with finer grinding (Gaudin, 1939). Unfortunately, generation of meaningful quantitative data for prediction of grind size requirements to achieve desired mineral separability from liberation studies is a much more difficult matter.

The random fracture pattern assumption, which implies that mineral assay in all size classes after breakage would be identical (Barbery and Leroux, 1985), is a clear weakness of these models (for example, see Table 4-5). On the contrary, preferential detachment along grain boundaries, to a greater

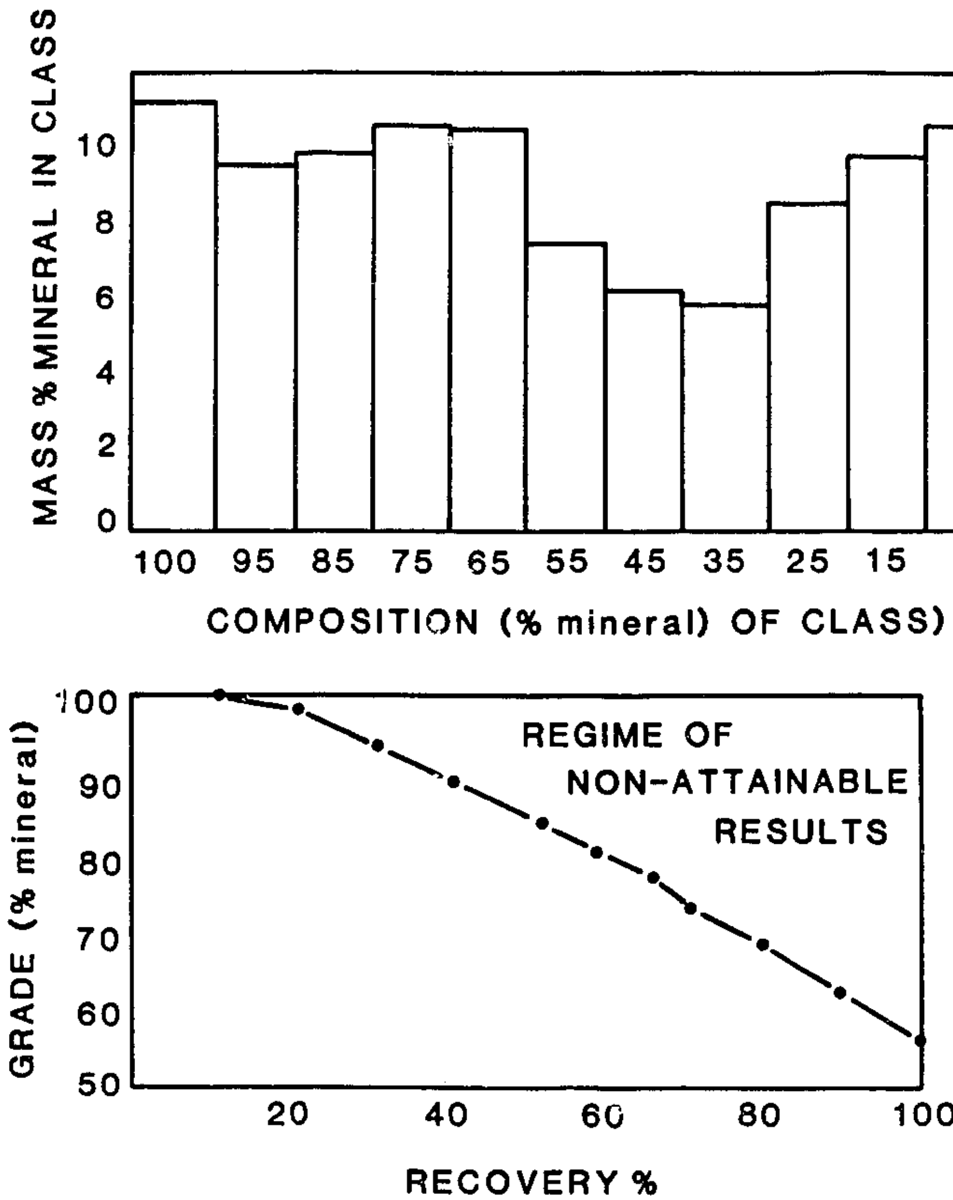


Figure 4-5. a) Example of Possible Distribution of True Particle Compositions.

b) Resulting Limiting Grade/Recovery Relationship from Feed with Composition Given in a).

Table 4-4. Assumptions of Single Parameter Liberation Models

	Gaudin, 1939	Weigel and Li, 1967	Meloy and Gotoh, 1985
Grain shape	Cubical	Cubical	Spherical
Grain sizes	Equal	Equal	Equal
Grain arrangement	Aligned in perpendicular planes.	Aligned in perpenicular planes.	Random
Spatial distribution of valuable grains	Maximum spacing between grains	Random	Random
Broken particle shape	Cubical, from aligned fracture planes parallel to grain surfaces	Cubical, from aligned fracture planes parallel to grain surfaces	Spherical
Position of fractures with respect to grain locations	Random	Random	Random

or lesser extent, appears to be a certainty which means the grain size distribution in the unbroken rock may be a better estimator of desired grind size (Petruk and Mainwaring,1987). The nature of comminution processes themselves are undoubtably a factor in this regard, and present levels of modelling accuracy and sophistication fall far short of handling such complexities.

Direct measurement of mineral liberation in ground materials is also a complex issue. Polished sections used for microscopical analysis always introduce stereological errors, since a locked particle may be observed as free if the section happens to pass through only one of the minerals present. The difficulty of the mathematical transformation from one or two dimensional observations into three dimensions (Baba et al, 1985; Gateau and Broussard, 1986) means that uncorrected comparative results, rather than absolute approximations, may be the best ones available at this time. While developments continue, the problem of accurate definition of the limiting grade-recovery curve (Figure 4-5) from liberation studies remains unresolved.

4.3.2.3 Mineral Distribution by Size

Grinding circuit product size is usually described in terms of the individual or cumulative weight percentage passing (or retained on) a standard series of sizing sieves. The 80% passing size, originally noted by Gaudin (1939) as near the modal value in many observed size distributions, is most often quoted when a single size is used to describe a distribution of ground materials.

The mass size distribution is adequate to describe fully the product size characteristics in grinding sytems used for homogenous materials, such as cement clinker. However, the mineral content in ores, as calculated from assays of each size class, is not generally evenly distributed through the size classes. This is particularly evident with grinding circuit products, but is also observable in material as coarse as the crushing plant product (Table 4-5).

Table 4-5. Typical Screen Analysis and Assays of Mill Feed (Crusher Plant Product) Armstrong-Smith, 1974

Size	Wt. %	Copper %
+ 20 mm	4	2.1
+ 13 - 20 mm	22	2.1
+ 6 - 13 mm	28	2.2
+ 3 - 6 mm	12	2.4
+ 10 mesh - 3 mm	9	2.8
+ 48 - 10 mesh	12	4.0
+ 200 - 48 mesh	5	4.8
- 200 mesh	8	2.7
Total	100	2.6

Consider Figures 4-6 and 4-7 (Myers and Bond, 1957; Hartley et al, 1983) in which reference is to the "copper distribution by size" to differentiate from "copper size distribution", which would incorrectly imply pure minerals and hence complete liberation. These examples show the copper mineral in the grinding circuit product is finer than the gangue particles at the coarse end, (as typified by a smaller K80 size) but that the copper mineral distribution is narrower, to an extent that a crossover point is observed, below which the copper mineral distribution is now coarser than the gangue. Figure 4-7 also shows that pyrite follows a similar behaviour, and that this characteristic was present in the feed to the circuit. Also note that the relative K80 of the mineral and that of the overall material size distribution may not bear a very consistent relationship, as shown by the data in Table 4-6.

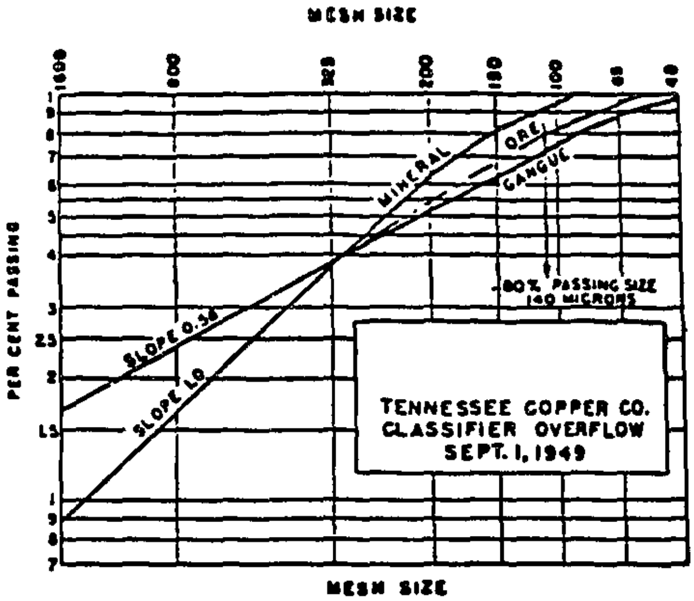


Figure 4-6. Ore, Gangue and Copper Distribution by Size in Classification Overflow (Myers and Bond, 1957)

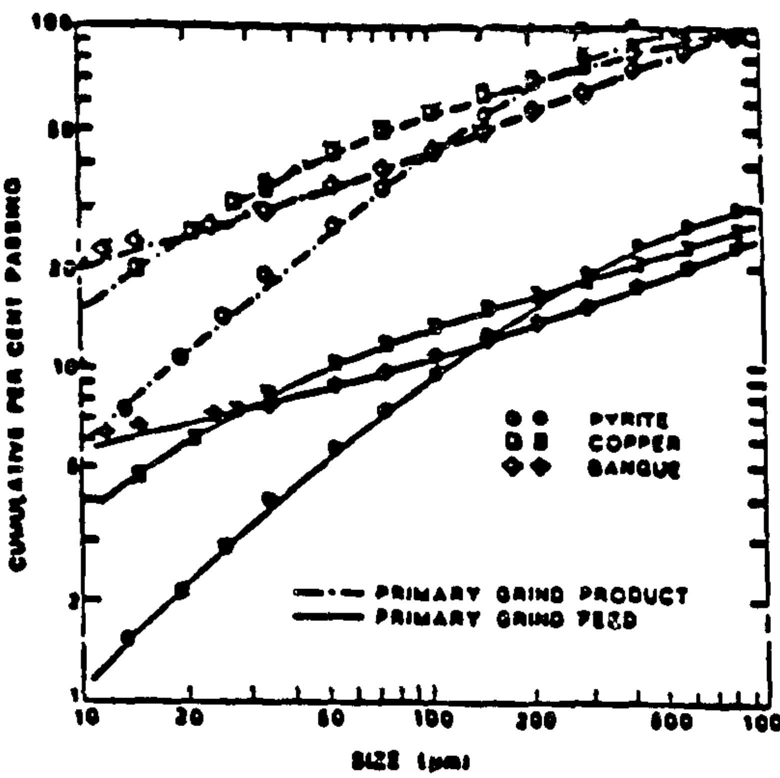


Figure 4-7. Gangue, Copper and Iron Mineral Distribution by Size, Circuit Feed and Product, Mt. Lyell (Hartley et al, 1983)

Table 4-6. K80 of Cyclone Overflow Solids vs. Copper in Monthly Composite Samples (Les Mines Selbaie)

K80 (μm)

Month (1985)	Solids	Copper	Ratio
September	108	56	1.93
October	110	64	1.72
November	115	66	1.74
December	100	64	1.56
		Average:	1.74

The fact that the copper minerals are typically denser than the gangue, and that this results in preferential recirculation to the ball mill by the classifier has been previously discussed (for example, see Finch and Ramirez-Castro, 1981; Laplante and Finch, 1984). However, while higher mineral density explains the finer K80 it is not apparent that it will also cause the copper mineral to be coarser at the fine end. This may reflect different breakage characteristics in different size classes for each mineral, including grain boundary effects.

4.3.3 Flotation Performance and Particle Size

Changes in the average flotation time, such as caused by variations in the feed rate or pulp density, may have effects ranging from minimal to significant on flotation performance, depending on prevailing operating conditions. Figure 4-8 (Klimpel, 1984) shows some typical recovery versus cell number (or residence time) curves. If flotation capacity in the plant is already strained, the trade off between recovery and feed rate will be a serious one. However, it is evident that for a plant currently operating in the high recovery range, small changes in flotation time will have little effect on total recovery.

Figure 4-9 portrays examples of recovery versus flotation time, first for a batch laboratory cell and secondly in a plant (Trahar, 1981), this time revealing the importance of particle size. These are typical curves for hydrophobic (flotable) minerals, and characterize three primary regions of flotability (Trahar, 1981; Trahar and Warren, 1976; Dobby and Finch, 1987).

In the fine region, somewhere below 5 to 10 um, decreased recovery is generally attributed to decreased probability of particle-bubble collision. The low flotation rates in this region will largely dictate total required retention time, that is, the design capacity of the circuit. Water entrainment is also a factor in the fine size range. In the intermediate size range, normally about 10 to 70 μm, the particles are most readily floated. As will be disscussed, the relative position of the highly flotable size range for two minerals with different degrees of hydrophobicity is a key factor in selective flotation. Finally, decreased recovery in the coarse region, above about 70 um, may be related to flotation

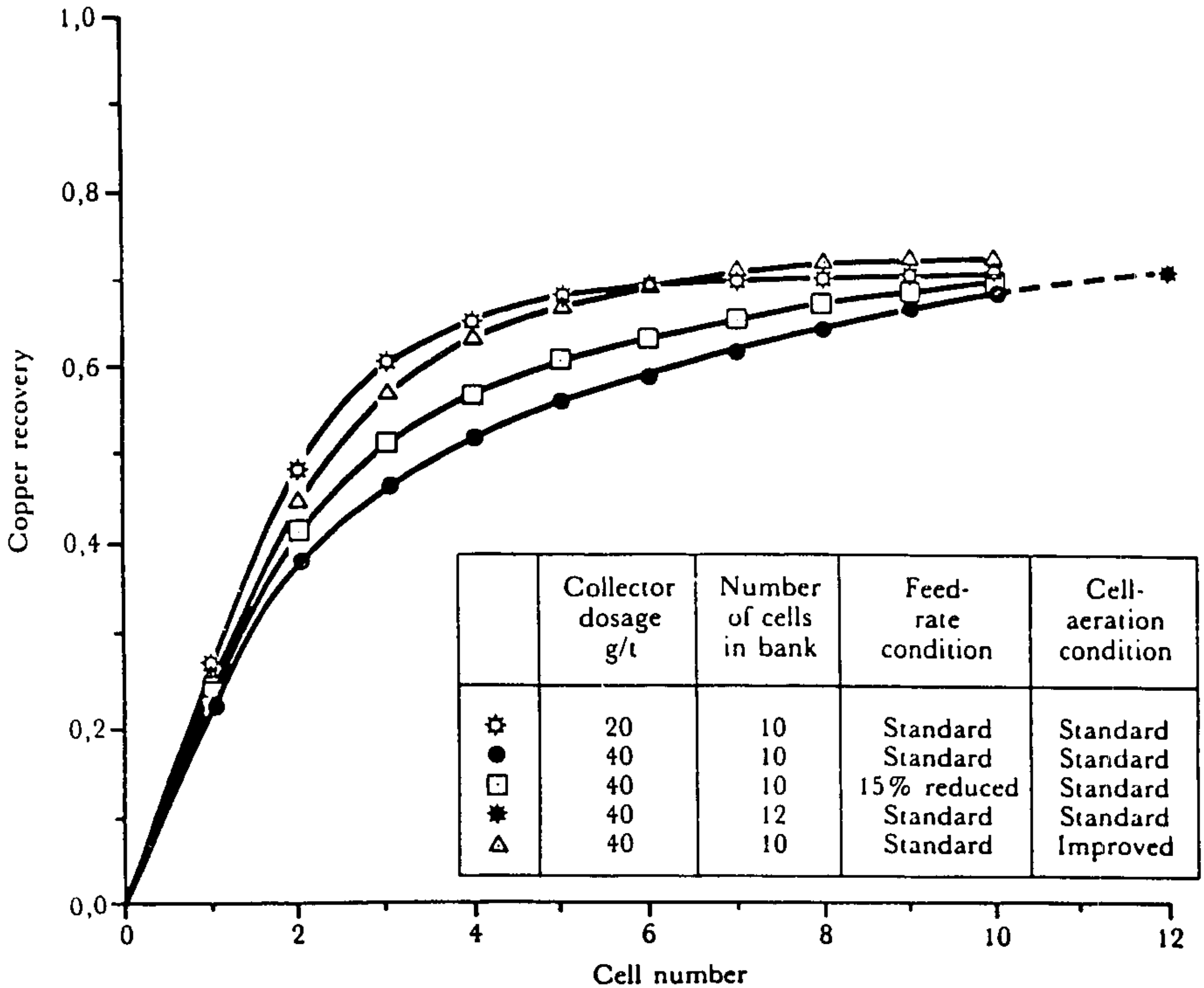
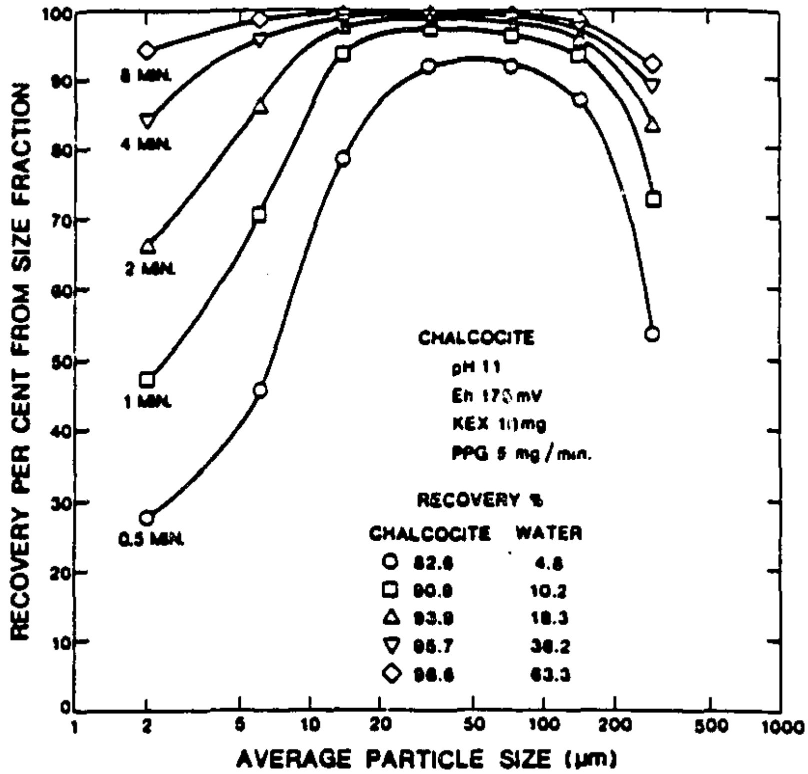


Figure 4-8. The Relative Influences of Changes in Collector Dosage, Feed Rate, Extra Cell Capacity, and an Improved Aeration Mechanism Leading to More Uniform Bubble Distribution Over the Cell (Klimpel, 1984).



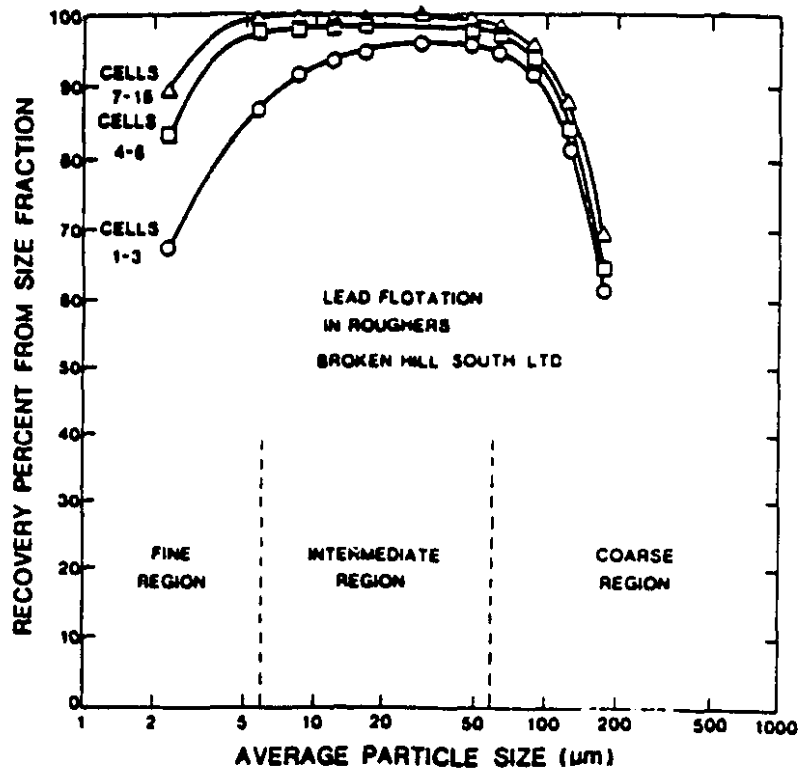


Figure 4-9. a) Recovery vs. Size with Time as a Parameter, Batch Flotation
b) Cumulative Recovery Down Flotation Bank (Trahar, 1981)

kinetics as a function of particle size itself (Dobby and Finch, 1987; Crawford and Ralston, 1988), but it is also in this region that locking of mineral and gangue is most probable and therefore may be a contributing factor (Trahar and Warren, 1976).

Note that while liberation may or may not be the underlying factor for poor flotation response, this is not the issue when characteristic particle size behavior persists despite various flotation practices. The relationship between flotation performance and particle size itself can form the basis for both technical and economic analyses of plant performance.

The usefulness of size by size mineral recovery as an engineering tool depends on its relative consistency with changes in design and operating conditions, other than those specifically intended to shift it. Figures 4-9a and b show that intermediate to even extremely short flotation times reflect ultimate recovery patterns based on mineral sizes. Laboratory batch and plant scale data also compare very favorably, as shown in Figure 4-10. Finally, an outstanding example from a plant is shown in Figure 4-11. While a different frother was successful in providing a shift in the size-recovery relationship, the corresponding plant size by size recovery curves remained basically unchanged with wide variations in the feed size and feed rate, as summarized below (Luckie and Klimpel, 1986).

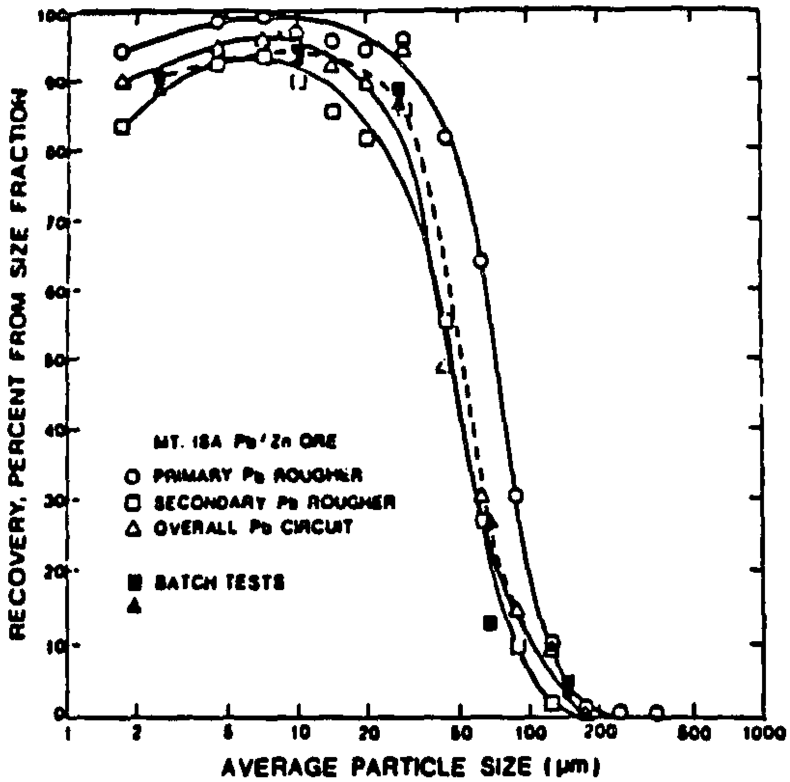


Figure 4-10. Similarity of Laboratory and Plant-scale Size-by-size Recovery, Mt. Isa Mines (Trahar and Warren, 1976)

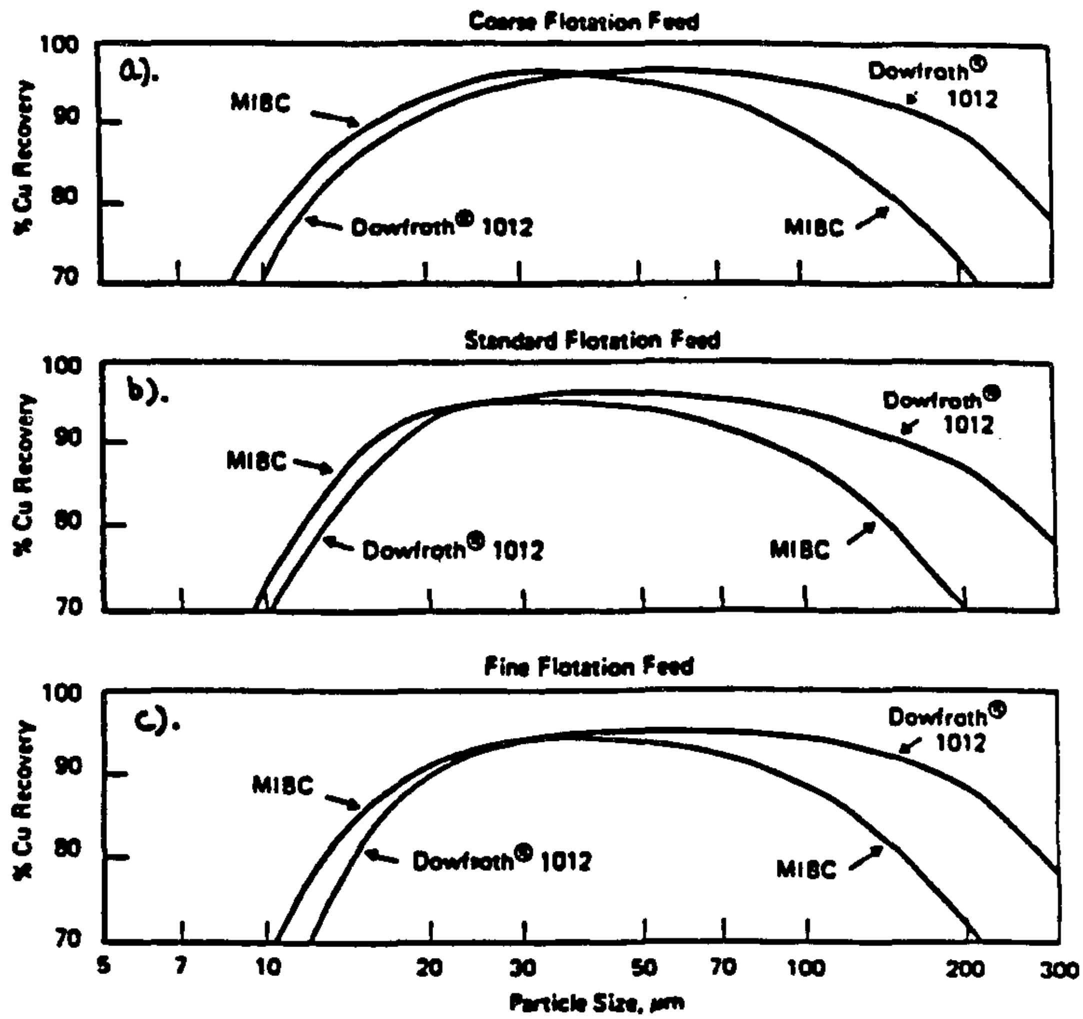


Figure 4-11. Size-by-size Recovey of Cu Sulphide at Three Different Grinds a) 80% - 130 μm, b) 80% - 100 μm, c) 80% -74 μm with Two Types of Frother (Luckie and Klimpel, 1986)

Table 4-7. Variations in Tonnage and Grind Size for Size-Recovery Evaluations (Luckie and Klimpel, 1986)

Case	Feed	Tonnage (MTPH)	Grind Size K80 (μm)
a	Coarse	95	130
b	Standard	75	100
c	Fine	53	74

Another important attribute of this means of expression of flotation peformance is the direct relevance of the technical parameter, recovery, to the plant economics. Significantly, the size region of maximum recovery of the valuable mineral also normally corresponds to the region of minimum recovery (maximum rejection) of the gangue (for example, see Figure 4-12). In consideration of these two phases, the sizes of maximum recovery of valuable mineral also yield the maximum concentrate grade.

The more complex case of a three phase system is exemplified in Figure 4-13, which has been constructed from broad laboratory and plant experience (Finch, 1986). It shows the general relationship between recovery and particle size for three mineral types, one strongly flotable (or strongly hydrophobic) (A), one weakly flotable (B) and one non-flotable (C). As an example, in zinc flotation, mineral A is sphalerite, B could be pyrite and C dolomite. This behavior is exemplified in the operating plant data shown in Figure 4-14.

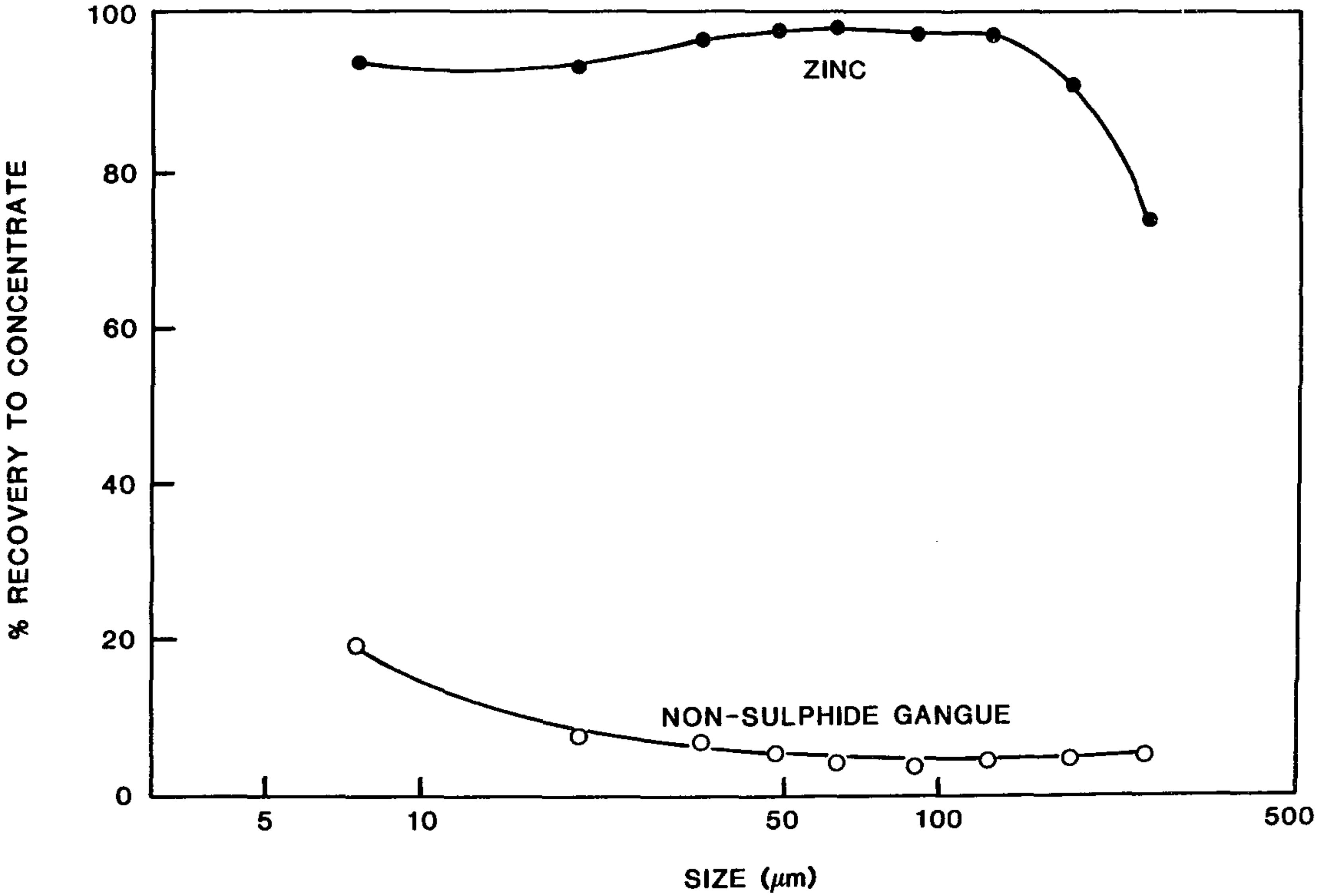


Figure 4-12. Size Recovery Curves for Zinc and Non-Sulphide Gangue In Zinc Rougher Flotation, Pine Point Mines.

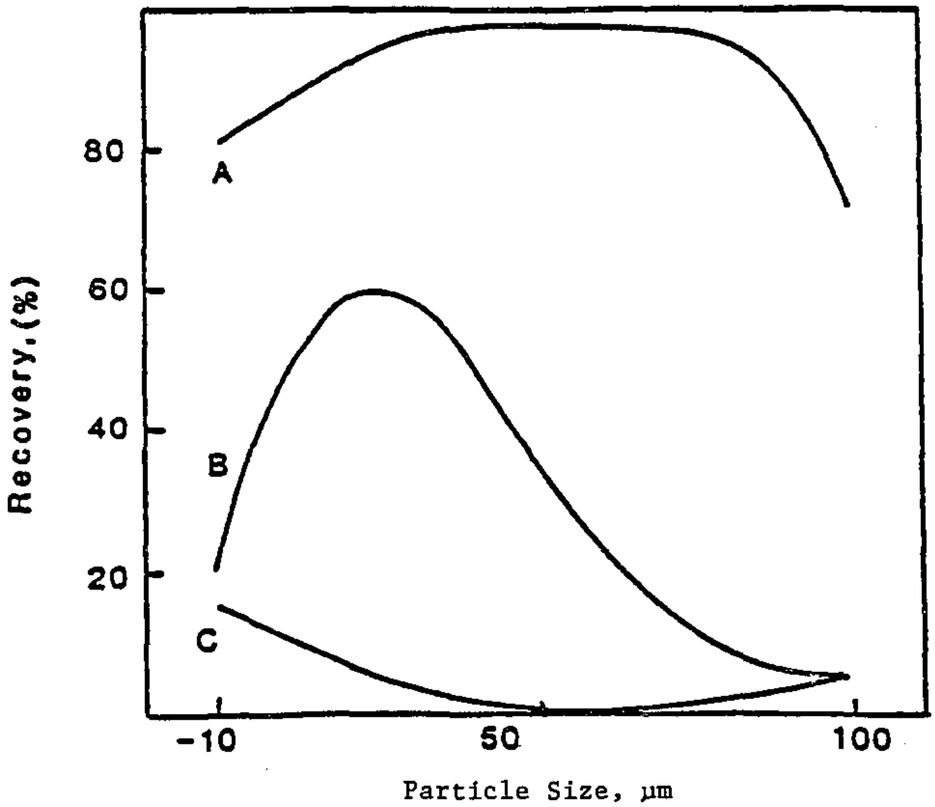


Figure 4-13. General Size-by-size Recovery for Highly Floatable (A) Moderately Floatable (B) and Non-floatable (C) Minerals.

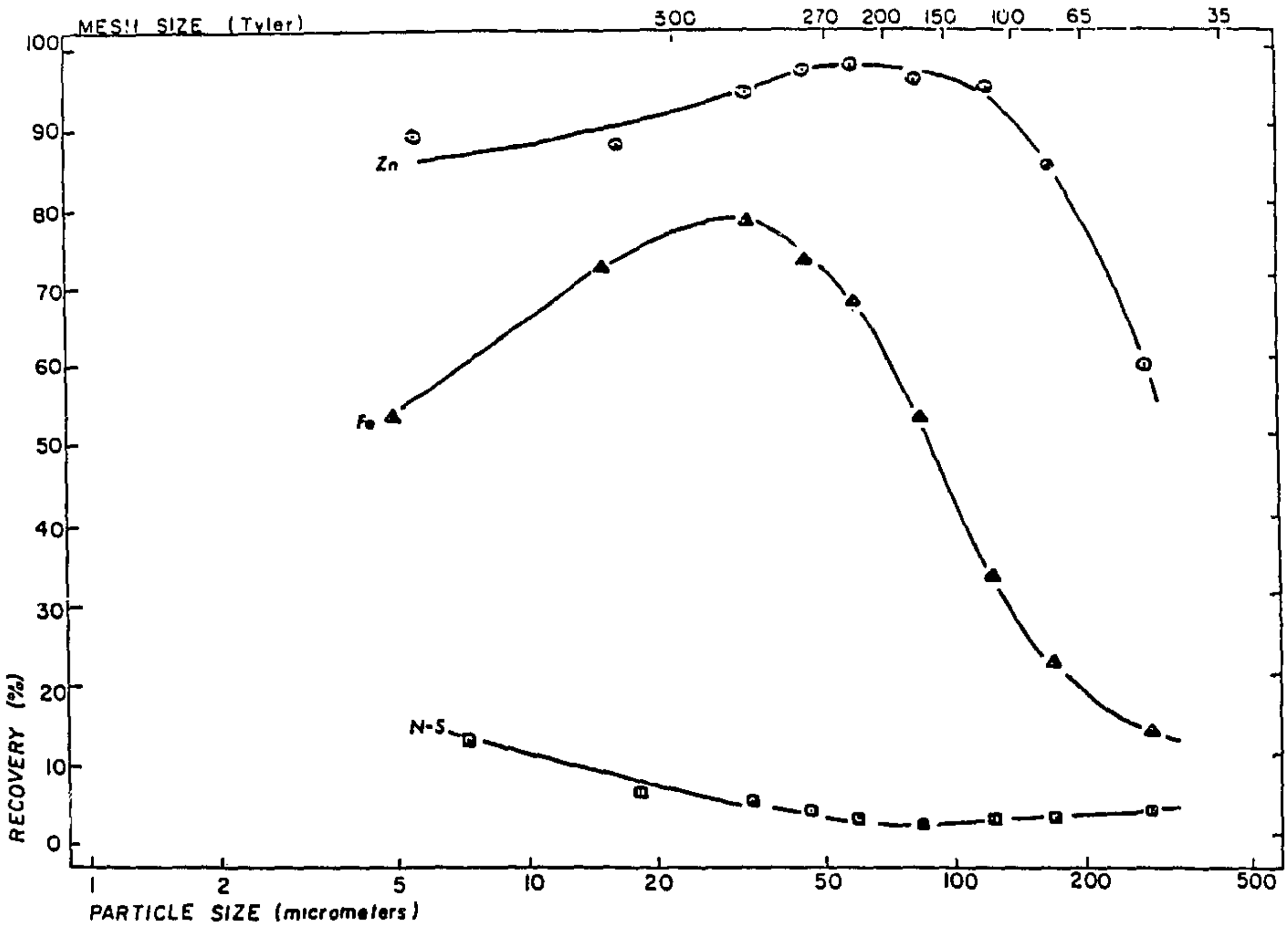


Figure 4-14. Size-by-size Recovery of Zn-sulphide, Fe-sulphide and Non-sulphide Gangue (NSG) across Zn rougher.

The relative positions of the size range of maximum recovery for the strongly versus weakly flotable minerals can be traced to flotation kinetics (Trahar, 1981; Trahar and Warren, 1976; Dobby and Finch, 1987). From mathematical modelling, relative flotation rates versus particle size for varying degrees of hydrophobicity can be respresented as shown in Figure 4-15. With decreased hydrophobicity, both the flotation rates of coarse particles and the size range of the maximum rate decrease. At the coarse end, hydrophobicity dominates because of the probability of attachment. However, at the fine end the rate is dominated by particle-bubble collision frequency, which is independent of hydrophobicity. The practical consequence is that selective flotation is relatively easy in the intermediate to coarse particle size regions, but increasingly difficult in the fine region. This difficulty is enhanced by non-selective water entrainment of all fine particle species.

4.3.4 Grinding Circuit Design and Operation

4.3.4.1 An Objective for Grinding Defined

Analysis of size-by-size recovery has been called "diagnostic metallurgy" (Kelsall, 1974). It can be used not only to indicate the desired size range for minerals in flotation feed (and hence grinding circuit product), but can be used to infer the state of locking of the minerals. Inferences regarding locking have been largely verified by liberation analysis, on the few occasions when this has been done

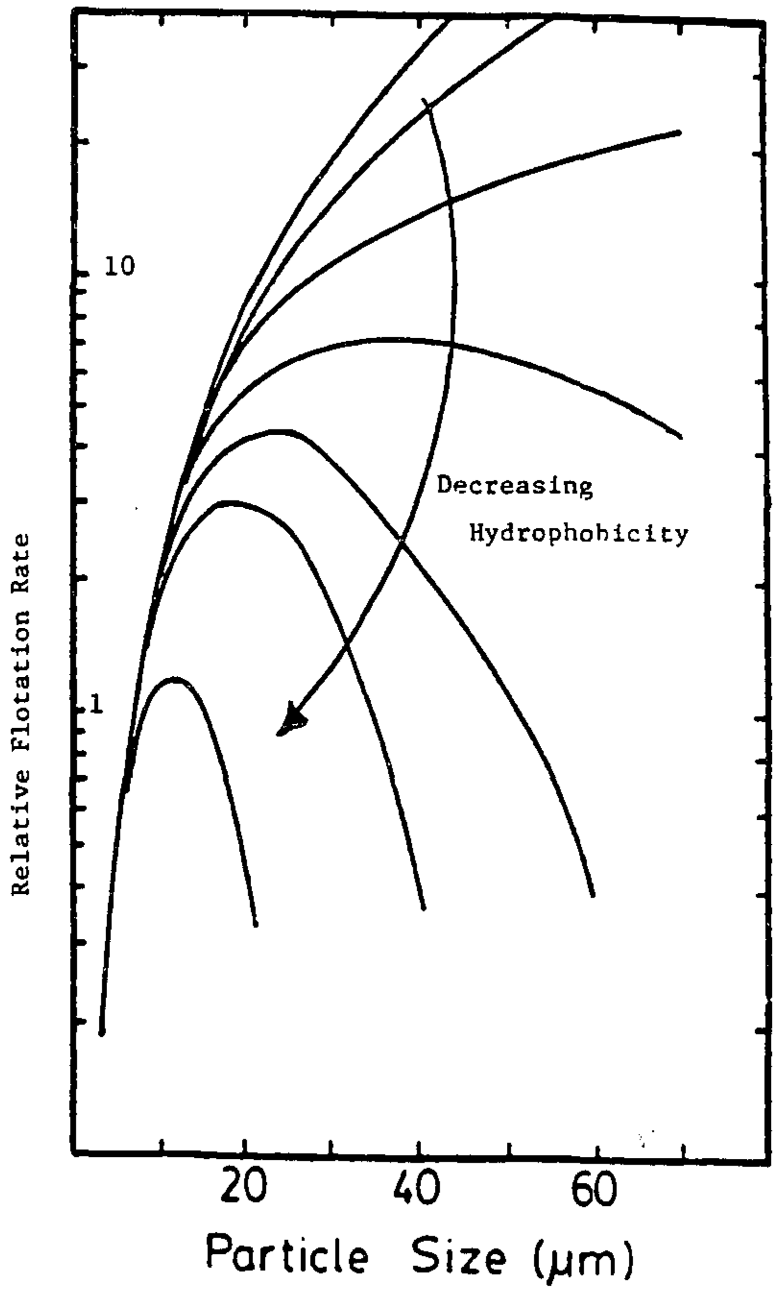


Figure 4-15. Relative Flotation Rate vs. Particle Size with Hydrophobicity as a Parameter.

(Johnson, 1987). This is an important point for it means, once again, that the plant engineer does not need to measure or know the degree of liberation in order to make use of size-by-size recovery data for interfacing studies.

By reference to the size-by-size flotation recovery characteristic, the grinding circuit objective for the simple, single flotable mineral case can be defined as the production of the most recoverable distribution by size of the valuable mineral within the constraints of total related operational costs. This objective is in agreement with the conclusion by Trahar (1981) in consideration of grinding and classification as the feed preparation process for flotation, and was adopted by Apling et al (1982) in their development of a combined communition-classification-flotation model.

For the more complex, multiple mineral case described earlier (Figure 4-13), it is clear that the grind size objective must be adjusted to account for grade losses in A concentrate because of recovery of B. Furthermore, if B is a valuable product to be recovered in a second flotation circuit, the use of staged grinding and flotation circuits to enhance selectivity would merit consideration.

4.3.4.2 Key Operating Variables

Reference to the mineral size-recovery characteristic of flotation also highlights factors of primary importance in grinding circuit operation for production of the most valuable product. Besides achieving the desired mineral K80 size, it is clearly desirable to produce as narrow a size distribution as possible to direct the maximum amount of the mineral value into the highest recovery region. This implies the need for:

a) effective classification (removal of suitable product size material from the grinding circuit before overgrinding, plus retention of oversize material) in order to produce the narrowest possible mineral size distribution at any given moment in time; and,

b) control of circuit operation to produce a consistent mineral product size distribution with variations in input or operating conditions over a given period of time.

While each of these is a major topic in its own right, two important observations should be noted at this time. First, in terms of its function as a feed preparation process, the final stage of grinding before flotation can be assigned specific desired product characteristics, the achievement of which is compatible with efficient, low-cost operation of the grinding circuit. Secondly, overgrinding, which contributes to both increased grinding and flotation costs and decreased mineral recovery, is an important economic concern. Therefore, in the sense that improved classification and process control can reduce overgrinding, they are of paramount importance to overall plant performance.

4.3.4.3 Use of Multiple Grinding/Flotation Stages

If the simple flowsheet of grinding followed by flotation was applied to a complex ore, that is one with two or more significant hydrophobic minerals, the economic analysis required for determination of the grind objective is more complicated than for the simple ore case. The relative economic gain of shifting the product size distribution towards the highest recovery range of the secondary valuable mineral (or away from it, if it is a contaminant in the concentrate) must be weighed against the resulting loss in recovery of the primary mineral. If both minerals are considered valuable, but yet have distinct, incompatible recoverable size ranges, then the need for flotation at more than one product size through stages of the comminution may arise.

The possible purposes of using multiple grinding/flotation stages can be summarized as follows (Finch, 1986). In consideration of a single mineral (whether in the simplest, single valuable mineral ore, or a more complex one) it may be;

1) to separate the valuable mineral at as coarse a size as possible in order to minimize grinding and other related processing costs, as well as to minimize overgrinding; or,
2) to prepare (either break, scrub, or re-condition) middlings streams, normally considered to consist mainly of locked particles (but more realistically of particles of similar flotation response), for subsequent re-treatment.

In consideration of multiple minerals which are to be collected in separate concentrates from a complex ore, it may also be:

3) to enhance overall selectivity between minerals in a given flotation circuit; and/or,

4) to maximize the individual recoveries of different valuable minerals which respond favorably to flotation in different particle size ranges.

4.3.5 Case Studies

4.3.5.1 Effect of Grind Size on Plant Recoveries

A simple way of showing the effect of fineness of grind on overall plant performance in an operation with limited grinding circuit capacity is given in Figure 4-16. At Mount Lyell (Hartley et al, 1983), a grinding circuit study was carried out in an effort to find ways to produce a finer grind, based on the observation of reduced recovery with coarser plant tailings. Results of the study showed that a reduction of the media size used in the secondary ball mills could provide an overall finer grind, which in turn would be expected to provide a 2% increase in total copper recovery.

Analysis of size-by-size recovery data at North Broken Hill (Slattery and Burgess, 1978), where the lead and zinc flotation circuits follow sequentially after grinding, showed the need for a finer grind for lead and silver, and a coarser grind for zinc. At the same time, the observation that a steady grinding circuit product was extremely important for

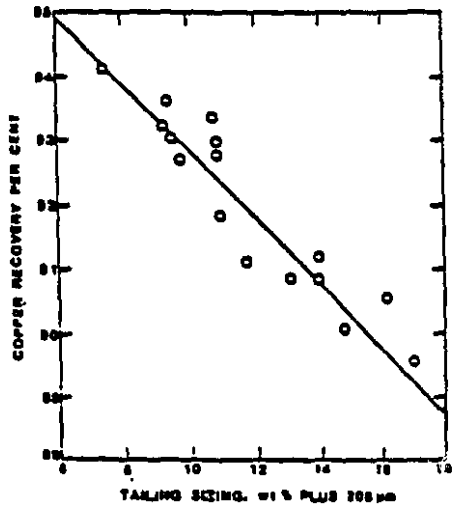


Figure 4-16. Recovery as a Function of
% + 208 µm in Tailings at
Mt. Lyell (Hartley et al, 1983)

stable flotation circuit operation led to the removal of a recirculating flotation middlings stream from the ball milling circuit.

4.3.5.2 Effect of Grinding Circuit Classification Improvements

The change from parallel to series ball milling at Broken Hill South (Kelsall et al, 1972) resulted in a reduction of the amount of coarse, poorly floating lead and zinc particles, and was entirely attributed to the improved classification performance of the new circuit arrangement. From size-by-size recovery data, it was determined that overall recoveries of both lead and zinc were increased by approximately 1% as a result of this modification.

At the Outokumpo concentrator in Hammaslahti (Tarvainen, 1980) a hydraulic cone classifier has been utilized for secondary classification of hydrocyclone underflow inside the pebble milling circuit. Besides reported gains in grinding circuit efficiency, 0.5 to 1% improvements in both concentrate grade and recovery have been attributed to reduced slimes production in the milling circuit.

4.3.5.3 Selection of Flotation Chemicals

Figure 4-11 from the work by Luckie and Klimpel (1986) summarizes the results from plant tests using MIBC, identified as providing a "fine particle froth" and DOWFROTH 1012, which provides a "coarse particle froth". They concluded that in this operation, the use of a frother which improved coarse particle flotation allowed a coarser grind while providing comparable overall recovery. It is also noteworthy that the relative ideal particle size ranges for maximum recovery in flotation with the two different frothers could be readily observed from laboratory flotation tests on the same ore.

4.3.5.4 Flotation Inside the Grinding Circuit

The practice of "unit cell" flotation inside the grinding circuit has been practiced in the past, (Taggart, 1945), but there has been a resurgence of interest in operating plants in Canada (Ramirez-Castro and Finch, 1980; Rawling and Goyman, 1984; Gowans and Simkus, 1984; Scheding, 1985) and Finland (Kallioinen and Niitti, 1985; Kallioinen and Tarvainen, 1984; Kennedy, 1985; Anon, 1986). The tendency for classifiers, particularly hydrocyclones, to act as concentrating devices for heavy minerals inside the circuit, as well as the substantial difference in size distributions inside versus after the circuit suggests its possible merit. Operating experience has shown that this method of sequencing grinding-flotation stages has the greatest potential for operating plant improvements under the following conditions:

1) when the valuable mineral has a strong tendency to slime due to its softness, or to preferentially recirculate because of its specific gravity;
2) when there is more than one significant mineral or concentrate produced, and the maximum recovery size

for one is substantially coarser than the other in the coarse flotation cell, enhancing selectivity between the two;

3) when a substantial degree of liberation is achieved for the mineral at a very coarse size; and,

4) when flotation capacity (or time) in the flotation circuit following the grinding circuit is a current constraint, related to fine particle recovery.

In the simple single mineral case, no net improvement in plant recovery is expected unless flotation performance at the coarse particle size is superior to that achieved at the finer size in the conventional circuit. In a more complex case, with two hydrophobic minerals, the unit cell concept has better potential.

4.3.5.5 Development of a Grinding Circuit Control Scheme

Perhaps the most striking example of recognition and use of individual mineral behavior in grinding in conjunction with the size dependent performance of flotation comes from the New Broken Hill Consolidated Concentrator in Australia (Lean and Baker, 1984). The operators had been sensitized to the importance of individual mineral behavior from observations of significantly different mineral distributions in grinding circuits operated with rake versus hydrocyclone classifiers (see Figure 4-17). Testwork directed at characterizing mineral behavior in the hydrocyclones led to the development of a control algorithm which predicts the lead mineral size

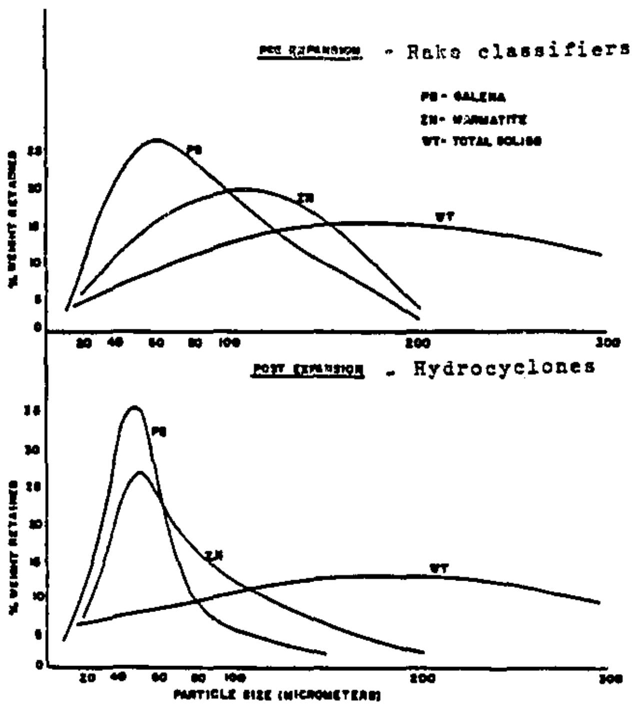


Figure 4-17. Mineral Distribution with Size in Grinding Circuit Product with Rake and Hydrocyclone Classifiers (Lean and Baker, 1984)

K80 in the flotation feed as a function of ore feed rate, grade, and cyclone overflow density. Since overall cyclone product K80 was also found in this case not to correlate consistently with the lead mineral K80, this control methodology was found to be superior to alternatives using direct measurement or model based prediction of overall product size for circuit control. It is extremely interesting in that it directly adopts a fundamental objective for the grinding circuit, which is to produce a specific mineral distribution size for the flotation circuit.

4.3.6 Discussion and Conclusions

One of the most important characteristics of flotation circuit feed relative to the recovery and grade performance of the flotation circuit is the distribution by size of the different minerals. This is because flotation rates and hence flotation recoveries vary significantly in the coarse, intermediate, and fine particle size regions. Although numerous factors influence overall flotation performance, relative size by size particle recoveries remain consistent over a reasonably wide range of operating conditions, particularly when flotation circuit capacity is not a severely limiting factor. This suggests that one approach to interfacing grinding and flotation operations may be to assess grinding circuit product value according to the distributions by size of minerals, and compare these to the size versus recovery performance of the minerals in the flotation circuit. Similar principles have been applied to analysis of industrial flotation plant performance (for example, see Luckie and Klimpel, 1986, and Johnson, 1987), development of a grinding circuit process control system (Lean and Baker, 1984) and the early development of an integrated communition-classification-flotation plant model (Apling et al, 1982). For the plant engineer, this appears to offer a practical approach to interfacing of grinding and flotation from normally available operating plant data. From this review it may be concluded that interfacing grinding and flotation, including determination of optimum grind size, can be accomplished from a knowledge of mineral distribution by size in the grinding circuit product and size-by-size recovery in flotation. It is not necessarily required to meausre directly or know the degree of liberation.

From analysis of mineral distribution by size in grinding circuit products, it has been observed that dense minerals have a finer K80 than the average. In the case of some copper minerals, the mineral distribution has often been observed to be narrower than the overall solids distribution, and the mineral is coarser at the fine end. It is not known if this is a general result or dependent on specific breakage behaviour (e.g., preferential breakage at grain boundaries) of the ore. The observation does have important practical consequences.

From analysis of size-by-size recovery in flotation, it follows that minerals can be divided into flotable (type A), weakly flotable (type B) and non-flotable (type C). Type A and B show typical fine, intermediate and coarse size range behaviour, the difference being that the intermediate range for B is narrower and shifted to a finer size than that for A.

Analysis of interfacing indicates that for a simple case of a single flotable mineral, the flotation size by size recovery relationship alone can be used to characterize the economic value of the grinding circuit product. An example will be given in section 4-5. Staging grinding and flotation will benefit the more complex case with two or more flotable minerals. A coarse primary grind will help reject more of the second mineral from the concentrate of the first. Unit cell flotation inside the grinding circuit falls into this category.

4.4 REVIEW OF PROCESS CONTROL IN CONVENTIONAL BALL MILLING

4.4.1 The Need for Grinding Circuit Control

The economic effectiveness of a process is determined by the quality or value of the product and the related processing costs. If the inputs (both the raw feed and the applied processing work) are absolutely constant, the product characteristics and the process efficiency will be determined only by the engineering design. The need for and degree of process control is therefore determined by the nature of variations or upsets in the process inputs, and the extent to which these affect product value and operating costs.

Conventional ore grinding is subject to a number of variations in the process inputs, including the following:

a) the feed rate, size, mineral content, and grindability of the ore (as well as other solids influxes);

b) water inflow from water addition points, intermittant and minor streams, and with the ore;

c) grinding media charge level and mill power draw;

d) the mechanical and dimensional condition of grinding, pumping, and classification equipment.

At the extremes, these variations may be compounded by instabilities originating from the circuit itself such as sump overflowing, pump surging, pipeline sanding, or ball mill overloading.

From the previous discussion, it can be said that a fundamental objective in closed circuit milling is to produce as narrow and as ideally sized (with reference to the subsequent separation process) a product as possible. Without suitable control, specified design conditions cannot be maintained, and efficiency is lost. Even a narrow mineral distribution that fluctuates will yield a broadened average size distribution over a period of time. Variations in the circuit operating conditions will result in the following problems:

a) an off-specification product size;

b) a varying product size resulting in a broader average size distribution; and,

c) inefficient circuit operation in terms of both classification system and size reduction performance.

Note once again that overgrinding is very costly. The fines represent wasted energy by their very production, and may create a number of negative downstream effects in terms of reagent consumption, thickening and filtering efficiencies, dust losses and handling, backfill plant recoveries, and the like (Cross, 1967).

The two fundamental objectives of a base level of grinding circuit control which must adjust for variations in the circuit inputs can therefore be generalized as follows:

a) to maintain the desired feed sizing for the concentration circuit; and,

b) to maintain efficient size reduction and classification system performance.

As discussed by Smith (1976), the benefit of circuit stability is not only to improve the level of plant performance, but also to provide the needed operating conditions before circuit optimization work is possible.

4.4.2 Requirements for Grinding Circuit Control

The first two sources of variation mentioned above, being short term in nature and requiring rapid response, can best be addressed with automatic control. Response to the second two, of a longer term nature and eventually requiring manual interaction for adjustment or correction, will follow as a result.

It is impossible for a circuit with a mill of fixed power draw capability (i.e. fixed speed and charge level) to adjust to short term changes in the fresh feed qualities (i.e., size or grindability), while achieving both of the above stated objectives, unless there is a change in the fresh feed rate. If the ore becomes softer, for example, and product sizing is maintained along with the same tonnage rate, this must be accompanied by a reduction in the overall efficiency of the circuit. When reasonable variations in the separation circuit throughput rate are acceptable, and when a fixed average (or maximum) tonnage will meet production scheduling requirements, instantaneous new ore feed rate should be allowed to vary about plus or minus 10 percent, if possible (Hulbert and Barker, 1984). This distinguishes conventional or single stage primary ball milling circuits from those in which the rate of new solids feed cannot be controlled, such as regrind milling applications. Usually the only short-term control variables in a grinding circuit with fixed speed drives on both mill and cyclone feed pump are the ore and water influx rates.

Maintaining constant cyclone feed conditions is the primary control requirement, as this will stabilize the overflow product size. At least a minor compromise in cyclone operating characteristics is necessary if new feed rate, and therefore cyclone overflow solids rate, is to be varied. However, known cyclone models do not suggest that the effect of the solids mass split per se on cyclone overflow product size is significant.

Sump overflowing, or the level running so low that pump surging can occur, must be avoided. Note that with fixed speed pumps, sump level is a direct indication of the circulating load volume (Mular and Jull, 1980; Deister, 1986; Webber and

Diaz, 1973). As well, ball mill operating density should be controlled to maintain breakage efficiency (Klimpel, 1982-83). The general requirements for a base level of ball milling circuit control can therefore be summarized as follows:

a) maintaining cyclone feed conditions;

b) maintaining cyclone feed sump level (or alternatively, circulating load); and,

c) maintaining ball mill operating density.

4.4.3 Control Strategies and Methods

Plant experiences with automatic control in either rod-ball (or pebble) or single-stage ball milling circuits covering a variety of control strategies and methods have been reviewed and a chronological listing of the data is presented in Appendix L. The dynamic response of circuit variables to changes in operating conditions is generally well understood and documented (for example, see Lynch, 1977), and information on control theory and the use of automatic controllers is readily available (Moys, 1986; Smith, 1979). Basic control philosophies have not changed from those applied strictly for manual operation, as an excellent article for plant operators written in 1945 attests (Ramsey, 1945). As well, a number of timely review articles (Gault et al, 1979; Birch, 1972; Herbst and Rajamani, 1979; Ulsoy and Sastry, 1981; Hales et al, 1982; Lynch, 1984; Mular, 1986) describe the evolution of grinding circuit control technologies over the past 20 years.

A number of general conclusions concerning selection and implementation of a practical control system can be substantiated by a consensus of the findings in these works, as given below:

1. Despite numerous variations, all strategies to control circuit product size fall into one of two categories (Herbst and Rajamani, 1979). These are type I, by varying the ore fresh feed rate, or type II, by varying the classifier feed water addition rate.
2. Concise, short-term control of cyclone overflow product size can only be achieved by cyclone feed water adjustments. This is supported by numerous test confirmations, and the historical trend towards type II strategies (see Ribiero et al, 1979 for an excellent example).
3. Some control approaches are inherently less satisfactory than others in terms of providing circuit stability. For example, control of sump level by water addition rate or pump speed will have a disruptive effect on classifier performance.
4. Longer term control of cyclone overflow product size, as influenced by normal ore hardness or coarseness variations, can be controlled by fresh feed rate to the circuit.
5. Variable speed pumps are sometimes considered to be a basic necessity for proper circuit control (Lynch, 1977). However, numerous plants operate successfully with fixed

speed pumps, which can handle variation in flow and self-regulate to a considerable degree with a well-designed sump (Mular and Jull, 1980). Since sump level is a direct indication of circuit circulating load, control of one will facilitate the other. Although control of sump level or circulating load by the new ore feed rate involves transport lags and requires careful tuning of controllers, it has been successfully implemented in a number of plants with fixed speed pumps (Cross, 1967; Deister, 1986; Webber and Diaz, 1973; Gault et al, 1979).

6. As discussed earlier, use of fresh feed rate as a manipulated control variable is a requirement to achieve both constant product size and efficient grinding objectives. This appears to be the motivating force behind a basic change in operating philosophy at Mt. Isa (Fewings, 1981). Mill power draw should be matched with average long term production requirements to ensure adequate circuit loading (high circulating load) for efficient grinding.

7. Control of either cyclone feed or overflow density alone is not sufficient for control of circuit product sizing. This is supported by plant results (Atkins et al, 1974), but can also be readily deduced from principles of hydrocyclone performance.

8. Particle size measurement of circuit product size can provide a direct means of circuit product size control (Hulbert and Barker, 1984; Deister, 1986; Webber and Diaz,

1973; Mansonti and Maio, 1984; Bradburn et al, 1976; Ribiero et al, 1979; Huls et al, 1986; Draper et al, 1969). Cyclone model based control can also be extremely effective and has much simpler equipment needs (Lean and Baker, 1984; Huls et al, 1986; Lees and Lynch, 1972; Manning, 1978; Balles et al, 1977). Direct particle size measurement does not provide mineral distribution information, which is an important characteristic ofthe separation circuit feed. However, mineral distribution size in the classifier product can be modelled (Lean and Baker, 1984). At Climax Molybdenum Company, use of the particle size monitor was discontinued after utilizing it to develop the cyclone control algorithm (Manning and Chang, 1977).

4.4.4 Proposed Method for Base Level Control of Primary Ball Milling

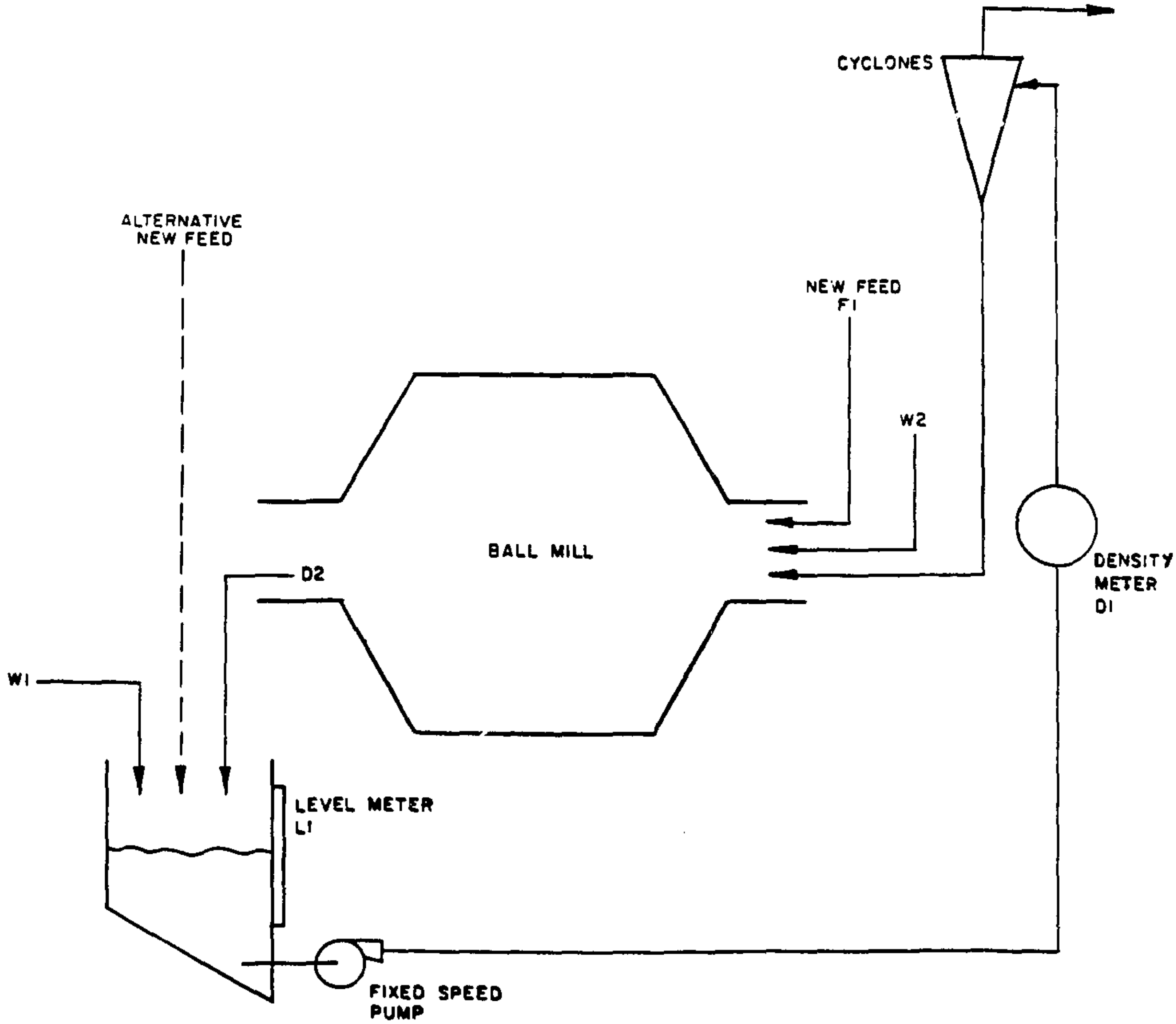
The above findings suggest the following control method can be applied to any primary ball milling operation (Figure 4-18), that is, whether following a rod mill or not, as long as feed rate may be adjusted.

Objectives:

1. Constant product sizing.
2. Maximum tonnage for prevailing mill power draw (charge level) conditions.

Control Loops:

1. Cyclone feed density controlled by cyclone feed sump water addition.



CONTROLLED VARIABLE	CONTROL VARIABLE
D1	W1
L1	F1
D2	W2

Figure 4-18. Base Level Control of Primary Ball Milling

2. Sump level (and circulating load) controlled by the fresh feed rate of ore.
3. Mill density controlled by mill head water (manual discharge density measurement).

This system requires a minimum of sensing and control equipment, is simple in concept and use, and lends itself to analog or computer control. It is also flexible in that the future addition of other sensors (e.g. cyclone feed flowrate, mineral assays) in conjunction with computer control can provide for development of the system to eventually model and control the separation circuit feed (grinding circuit product) mineral distribution by size, as practiced at New Broken Hill (Lean and Baker, 1984). The two primary control loops (no. 1 and 2) have been successfully applied in industry in the past, and dynamic simulation has shown this simple strategy yields very good circuit stability after suitable tuning (Dube et al, 1987).

4.4.5 Historical Benefits of Automatic Grinding Circuit Control

Well applied automatic control of grinding now has an established history of economic success. Average efficiency or capacity improvements over manual control have exceeded 5 percent (Horst and Bender, 1979), a minimum expectation, the value of which is often multiplied several times by achieved results. Some of these are summarized in Table 4-8.

4.4.6 Summary and Conclusions

This review was carried out to determine the need for grinding circuit control, to consolidate operating plant

Table 4-8. Reported Benefits of Automatic Grinding Circuit Control

Plant (references)	Reported Benefits
Mamoraton (Steffensen and Aubrey, 1957)	Control of -45 µm fraction.
East Malartic (Daniel, 1967; Kelly and Gow, 1966)	Reduced variability of grind.
Moose Mountain (Bilon, 1967)	Stabilized grind, increased recovery.
West Driefontain (Cross, 1967)	More stable operation, $70,000 per year in improved recovery.
Inspiration Consolidated (Diaz and Musgrove, 1973)	4-10% increased tonnage.
Craigmont (Webber and Diaz, 1973)	5% increased tonnage, stable operation, consistent grind.
Mount Isa (Fewings, 1981; Gault et al, 1979; Lees and Lynch, 1972; Lynch, 1977)	20% increased circulating load and 5% increased capacity, narrower product size distribution. Cost was 1/4 of 1% of plant capital cost.
Brenda (Bradburn et al, 1976; Gault et al, 1979)	4% increased tonnage with a 1 year payback.
Morenci (Balles et al, 1977)	Increased tonnage, reduced reagent consumption, stability, flexibility and process knowledge.
Climax (Manning and Chang, 1977; Manning, 1978)	Increased tonnage 9%, greatly reduced variance in produce size.
Arafertil (Ribiero et al, 1979)	Paid for itself in the 3 month familiarization phase.
Ozark (Mansonti and Maio, 1984)	Increased tonnage 15%, increased recoveries.
Buick (Hulbert and Barker, 1984; Perkins and Marnewecke, 1978)	3-5% increased in throughput, stability, and other cost reductions valued at $2.3 million per year (plant wide).

experiences, and to seek specific conclusions and recommendations relative to automatic control of ball milling circuits. The following is a summary of the most important conclusions.

1. A base level of control should provide stable classifier feed conditions through normal variations in the size and grindability of the circuit feed and the work input through the grinding charge.
2. Effective short-term control over the classifier product (cyclone overflow) sizing can be effectively achieved by classifier feed water adjustments.
3. Constant product sizing and efficient circuit operation, taken together, require the use of feed rate as an adjustable variable.
4. A simple, effective system for base level automatic control of ball milling has been proposed. This can be a high return investment in its own right. Just as importantly, operating stability is a basic requirement for circuit design optimization work to be effective.

4.5 EVALUATION OF THE BALL MILL CIRCUITS AT KIDD CREEK MINES AND LES MINES SELBAIE

4.5.1 Mill Power Draw and Charge Level Relationships

4.5.1.1 Ball Mill at Les Mines Selbaie

A set of mill power draw measurements were taken on July 16, 1986. Following a four minute grind out period during which the feed to the circuit was stopped, the mill was entered, and the inside diameter and average charge level measured. The data obtained are summarized below.

Motor amperage: 93 amps
Line voltage: 4050 volts
Mill inside diameter: 3150 mm (124 in.)
Average distance from top inside of mill to
to charge level: 2020 mm (79.5 in.)

The mill power draw at the pinion was calculated to be 545 kw using the method described in chapter 3. The mill volumetric charge loading, Vp, can be calculated from the following equation (Bond, 1961).

$$Vp = 1.13 - 1.26 \frac{Q}{D}$$

Q = distance from inside top of mill to charge level
D = mill inside diameter

Solving, Vp = .322, or 32.2% of mill volume.

The charge level versus power draw curve in Figure 4-19 is based on this measurement, and the empirical power draw formula used by Allis-Chalmers (Rowland, 1982). Note that although the power draw measurement is quite accurate, charge level measurements can easily result in variations of 2 to 3 percent, so this curve should be taken as approximate pending collection of more extensive data. For relative charge volume estimates for economic analysis, however, it should be adequate.

Also note that this curve applies to this mill when operating with a charge of grinding slugs rather than balls. Slugs may cause the mill to draw higher than expected power owing to their higher packing density (McKim, 1986).

4.5.1.2 Ball Mills at Kidd Creek Mines

The mill power draw (at the pinion) versus volumetric charge loading curves for the primary and secondary, and

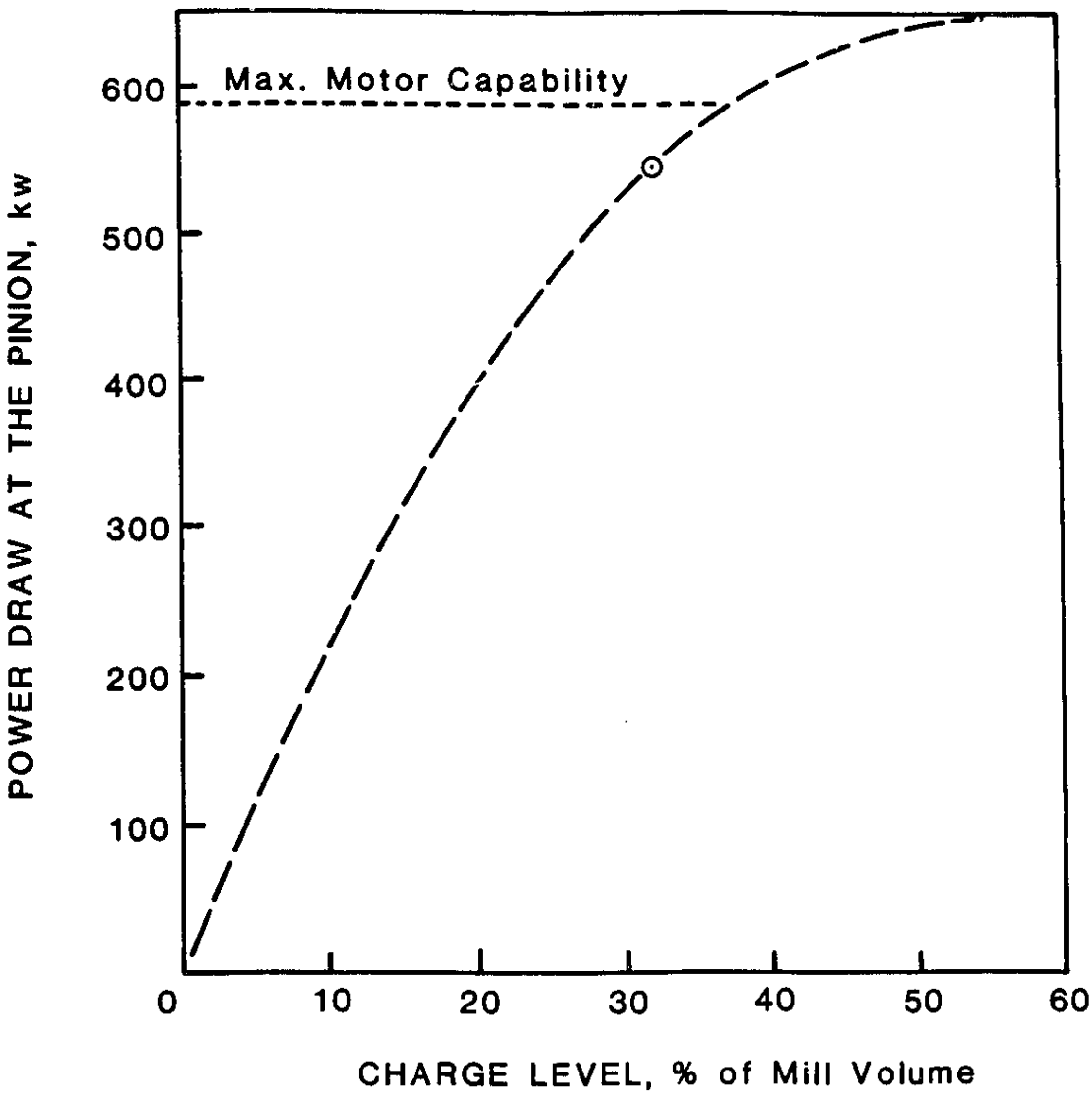


Figure 4-19. Power Draw vs. Charge Level, Selbaie 3.20 x 3.96 m Ball Mill

regrind ball mills, based on the Allis-Chalmers empirical power draw formula (Rowland, 1982), are shown in Figures 4-20 and 4-21. The minor effect of media sizing on mill power draw has been ignored in these calculations. Note that these curves should be taken as approximate until verified by actual power draw and charge level measurements.

The motor output capability, both with and without impinging on its service factor, is also shown for each mill.

4.5.2 Ball Mill Circuit Performance Characterization

4.5.2.1 Ball Milling at Les Mines Selbaie

Size and Mineral Distribution Data

The raw and mass-balance adjusted size distribution data down to 37 µm (400 mesh) for general surveys no. 1 and 2 are shown in Figures 4-22 and 4-23. The overall solids and copper mass distributions by size in the cyclone overflow for survey no. 2 down to 5 µm are shown in Figure 4-24. The latter clearly displays the phenomenon of a narrower copper mineral distribution.

Copper Values in Grinding Circuit Product, Survey No. 2

When the size by size recovery data for the flotation circuit given in Figure 2-2 are applied to the observed copper mineral distribution in the cyclone overflow in Figure 4-24, the total calculated recovery to concentrate is 95.9%. Figure 2-2 also shows recoveries in the range of 98% in the high plateau region for particles of near ideal size for the

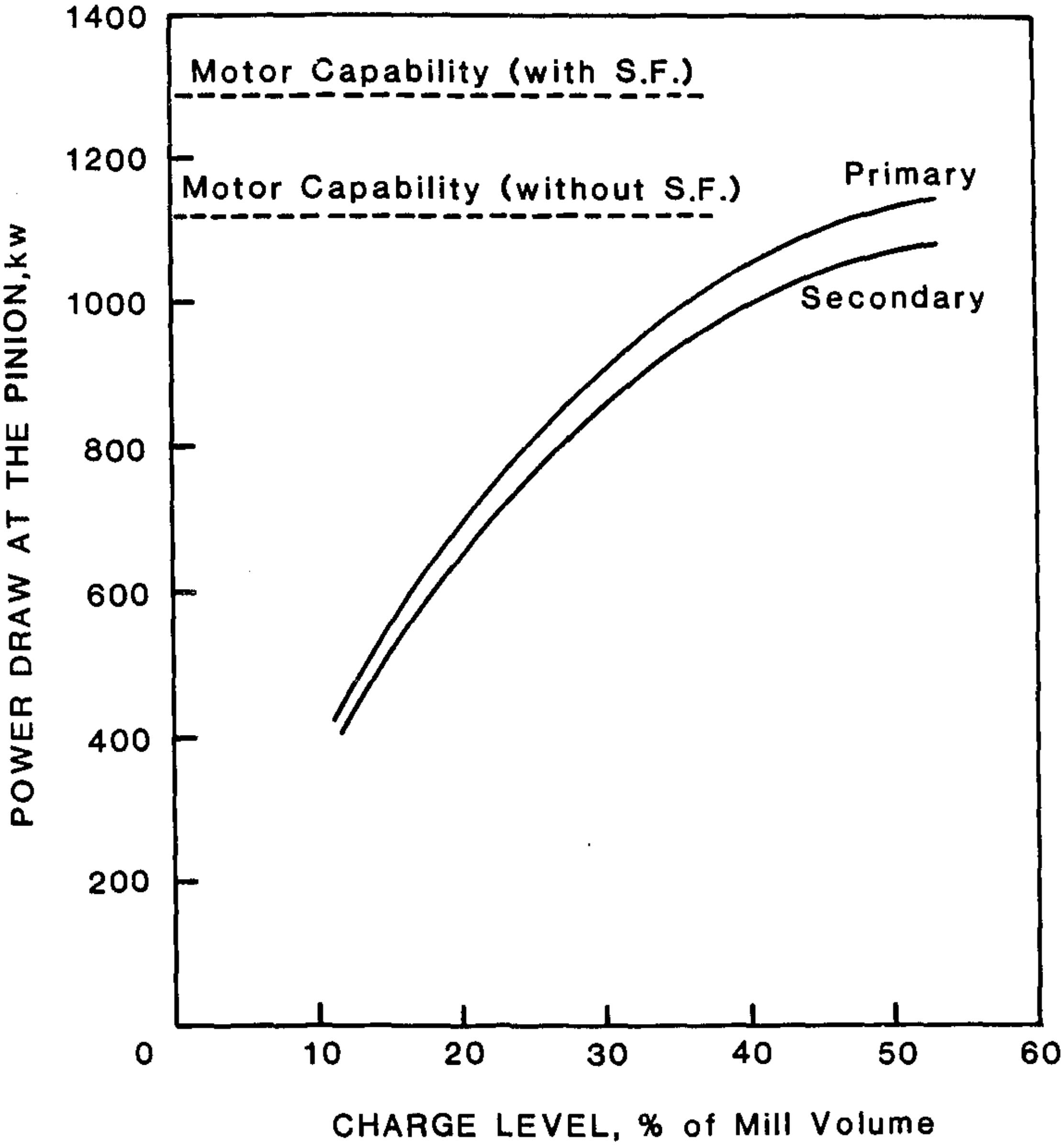


Figure 4-20. Estimated Power Draw vs. Charge Level, Kidd Creek 3.66 x 5.49 m Ball Mill

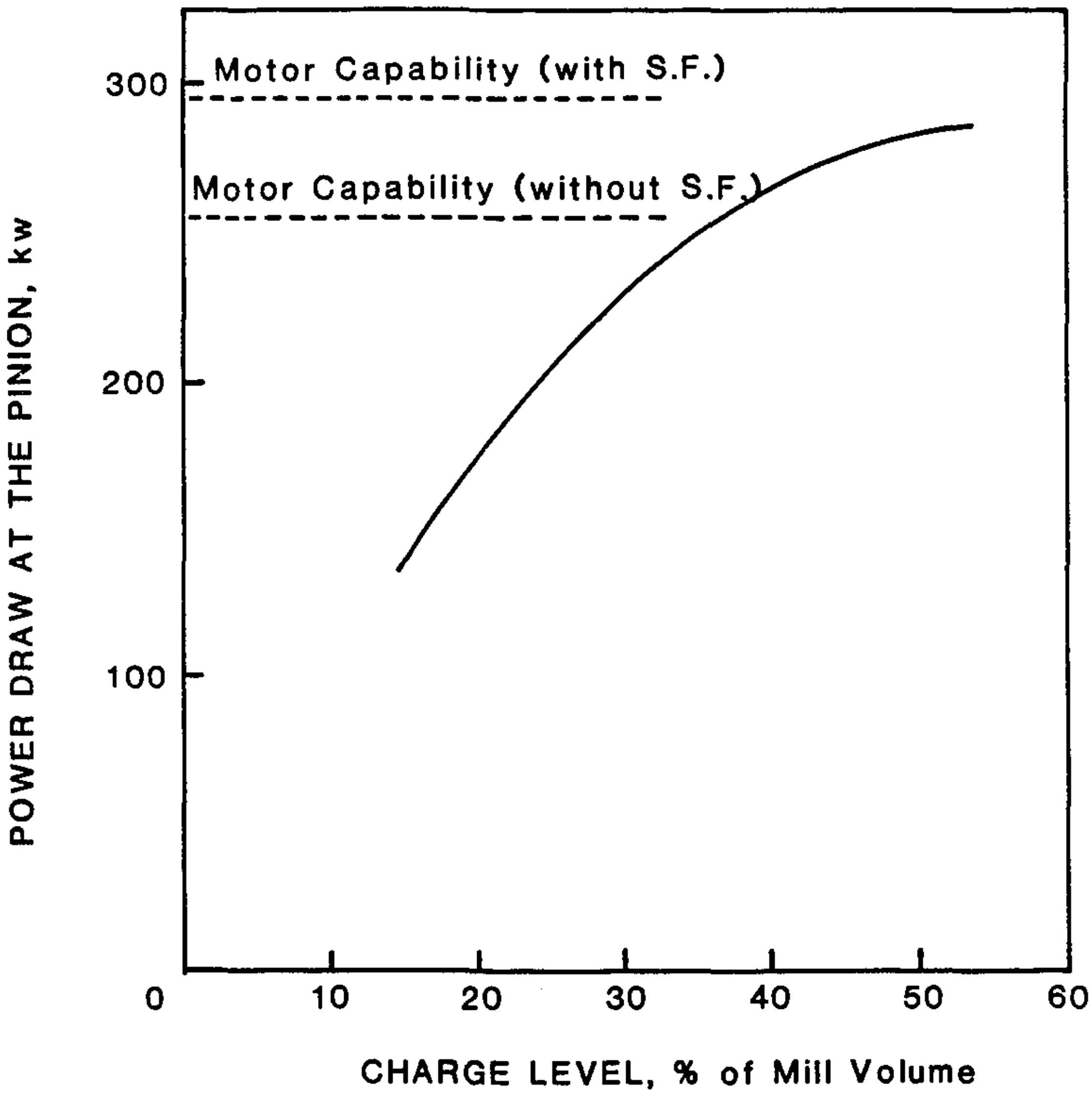


Figure 4-21. Estimated Power Draw vs. Charge Level, Kidd Creek 2.44 x 3.66 m Regrind Mills

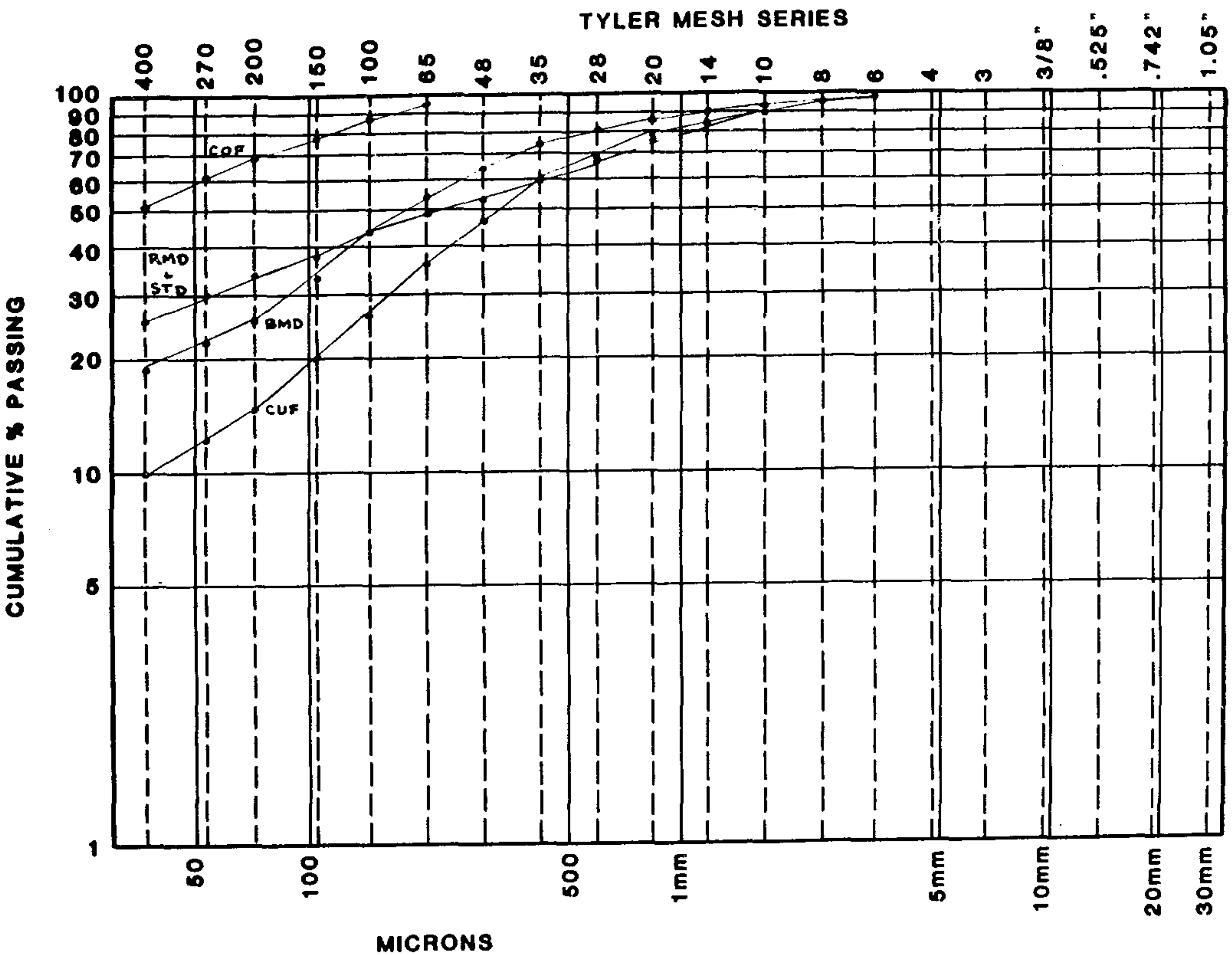


Figure 4-22. Raw (Points) and Adjusted (Lines) Size Distribution Data, Selbaie Survey No. 1

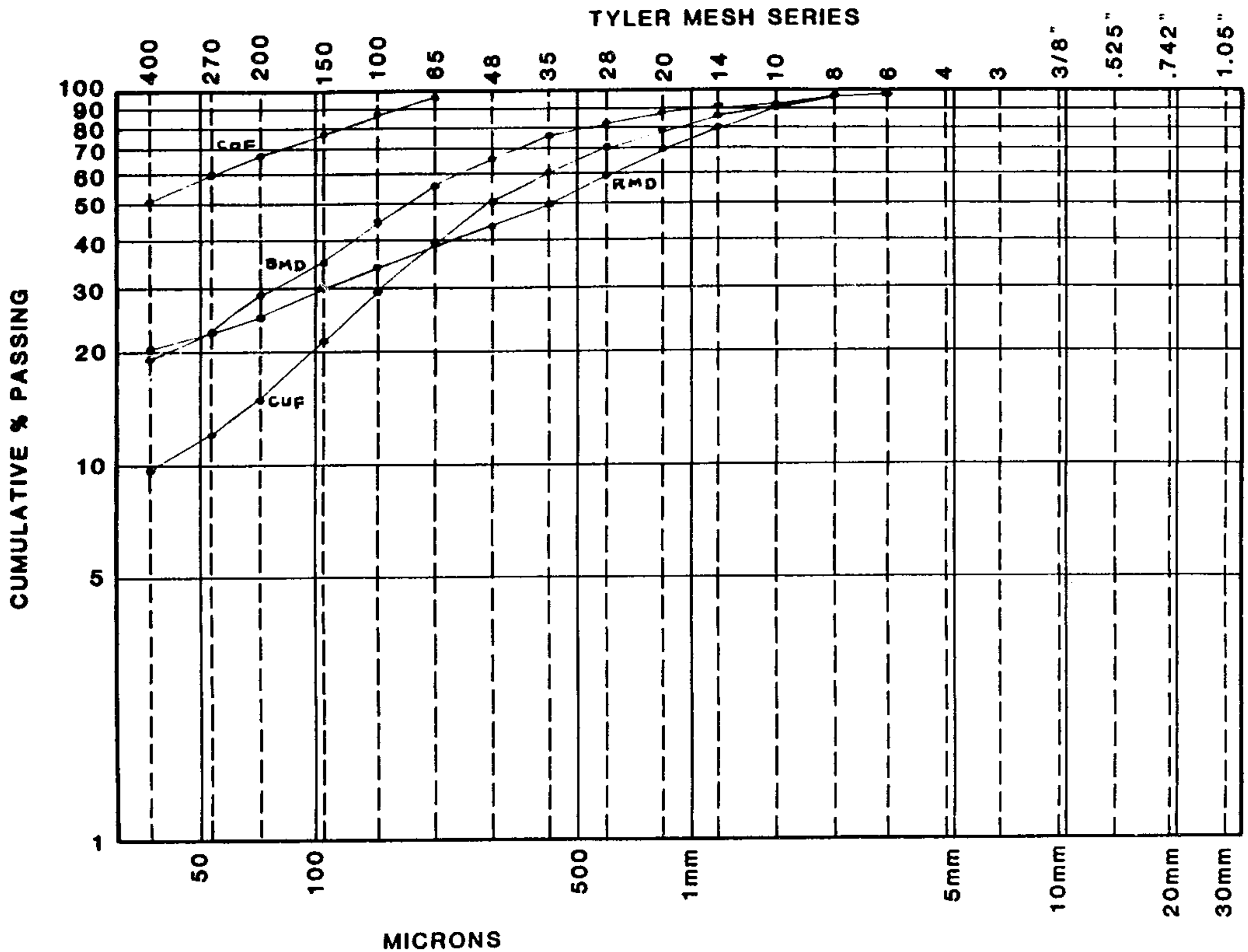


Figure 4-23. Raw (Points) and Adjusted (Lines) Size Distribution Data, Selbaie Survey No. 2

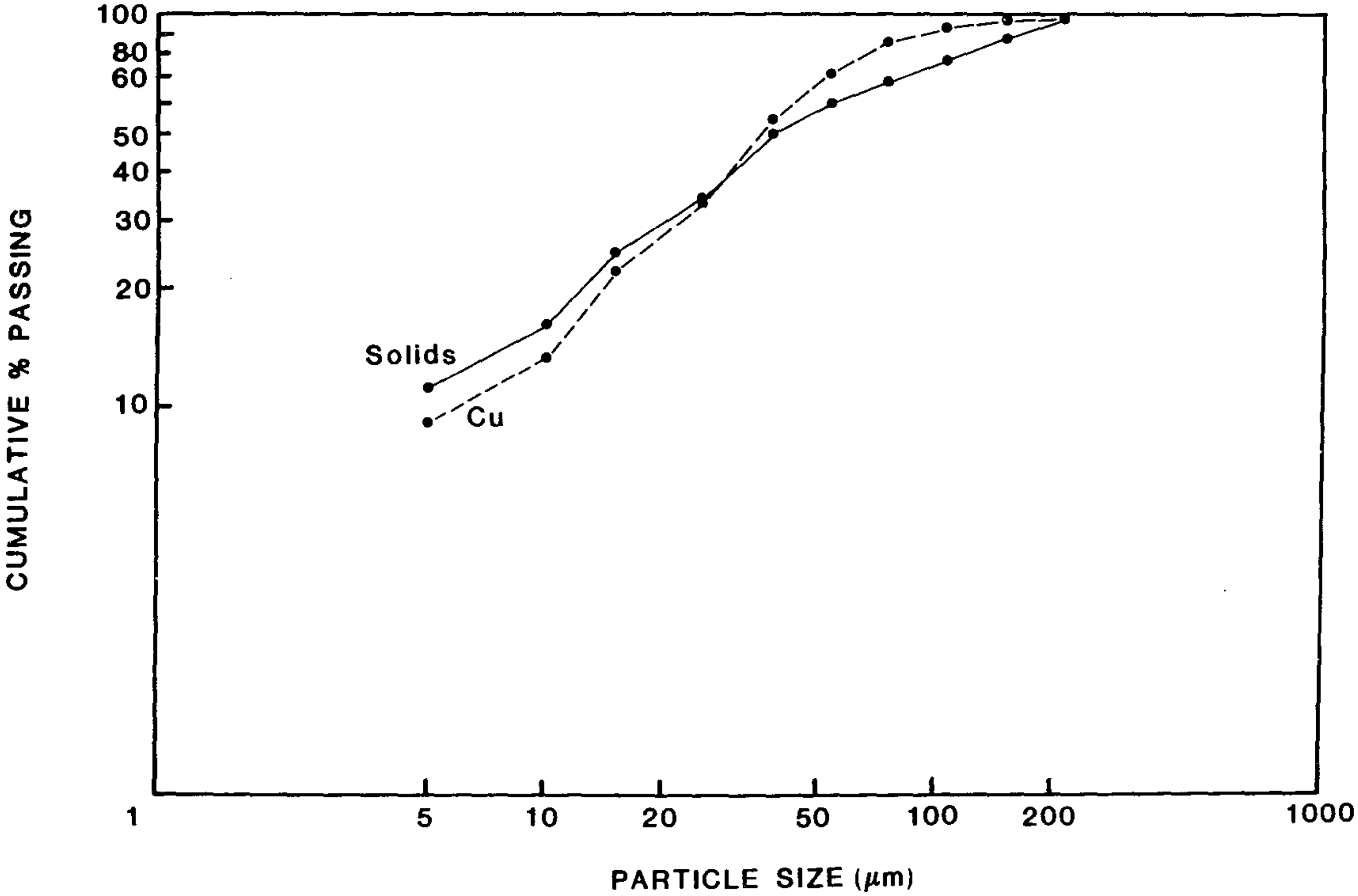


Figure 4-24. Overall Solids and Copper Distributions by Size in Cyclone Overflow, Selbaie Survey No. 2

flotation circuit. Based on the estimated actual recovery of 95.9%, we can therefore attribute a 2.1% loss in recovery to the "non-ideal" mineral size distribution in the flotation feed. Assuming a price of $2.205 per kg. for copper, 65 percent of which is payable in the smelter contract, the commercial meaning of these data can be summarized as shown in Table 4-9. All values are given in terms of dollar value per metric ton of ore fed to the flotation circuit during the survey.

Table 4-9. Copper Values in Grinding Circuit Product, Selbaie Survey No. 2

Feed Grade, % Cu:	3.24%
Value:	$46.44 per ton
"Ideal" Flotation Recovery:	98.0%
Value:	$45.51 per ton
Estimated Actual Recovery:	95.9%
Value:	$44.54 per ton
Total Recovery Loss due to Non-Ideal Flotation Feed:	2.1%
Value:	$0.97 per ton

Operating and Laboratory Work Indices

The relevant data for calculation of Bond operating work indices are given in Table 4-10, as are the results from Bond laboratory ball mill work index tests performed at McGill University. For survey no. 1, it was assumed that the surge tank discharge (crusher fines) effectively passed through the

Table 4-10. Ball Mill Circuit Performance Summary

Selbaie Surveys No. 1 and 2

Survey No.	1	2
Date	Nov. 12/86	Nov. 13/86
Circuit Feed		
New Feed Rate (DMTPH)	67.1	70.3
Feed Size, F80 (µm)	1,150 µm	1,160 µm
% - 106 um (150 mesh)	29.3%	30.3%
Test W.I. @ 150 µm (kwh/t)	12.8	11.8
Bond Test Grindability (g/rev)	2.08	2.31
Surge Tank Discharge (DMTPH)	16.9	n/a
K80 size, µm	131 µm	n/a
Circuit Product (new)		
Product Size, P80 (µm)	112 µm	115 µm
% - 106 um (150 mesh)	78.5%	77.6%
Circuit Performance		
Power Draw (kw)	527 kw	523 kw
kwh/t	7.85	7.44
Operating W.I. (kwh/t)	12.08	11.65
W.I. Eff. (Test/Op.), %	106%	101%
New - 106 µm solids (DMTPH)	33.0	33.3
106 um Classification Eff.	73.0%	71.0%
106 um Specific Production Rate (kg/kwh)	62.6	63.7
106 um Specific Grinding Rate, SGR (kg/kwh)	85.8	89.7
Grinding Rate Ratio (SGR/Bond Gr.)	41.2	38.8

circuit without being ground, so that the performance is based only on grinding of the rod mill discharge material. The crusher fines were excluded from the circuit feed and were subtracted from the cyclone overflow for calculation of the circuit grinding performance. The surge tank discharge material is quite similar in size distribution to that of the cyclone overflow so virtually all of the size reduction in this circuit will be carried out on the feed from the rod mill circuit.

The work index efficiency shows only a slight decrease of 5% from survey no. 1 to no. 2. Assuming a relative error of between 3 and 4% for each efficiency determination, this difference is just significant, at the 95% confidence level.

Circuit Classification Efficiencies

As discussed previously, the circuit "target" product size may be defined arbitrarily, but it is convenient to select the circuit P80. This size is close to the modal value of typical size distributions of ground products (Gaudin, 1939), and also often approximates the grinding circuit hydrocyclone classifier corrected d50 cut size (Arterburn, 1982). For the Selbaie ball mill circuit, the functional objective of the circuit operation may then be defined as the production of minus 106 um (150 mesh) material. The overall classification efficiency is then indicated by the relative amount of minus 106 um material present in the mill at any given time. The fines inventory of the ball mill, which represents the relative volume of wasted space and wasted power delivered by the grinding charge, can be estimated as follows.

	Survey No. 1 (% - 106 µm)	Survey No. 2 (% - 106 µm)
Ball Mill Feed	20.3	21.8
Ball mill Discharge	33.7	36.1
Average Relative Fines Inventory	27.0%	29.0%

The overall circuit classification efficiency is the relative mill volume available for grinding of the coarse solids, or 73% and 71% respectively. Note that while 100% of the mill steel load draws the mill power, the portion effectively used on coarse solids is controlled by the circuit classification efficiency.

The relative efficiencies measured between the two surveys suggests that the crusher fines and water influx from the surge tank has no major net effect on circuit classification efficiency.

Specific Grinding Rates and Grinding Rate Ratios

Using survey no. 2 as an example, the circuit feed contains 30.3% minus 106 μm and the circuit product contains 77.6% minus 106 μm material. The new minus 106 μm production rate is (.776 - .303) x 70.3 t/h feed, or 33.3 t/h. On the basis of power consumed, this represents:

$$\frac{33.3 \text{ t/h}}{523 \text{ kwh/h}} = 0.0637 \text{ t/kwh, or, } 63.7 \text{ kg/kwh.}$$

This is the "specific production rate" or the units of new product size produced per unit of work applied. This is related to both the efficiency of the process, and the grindability of the ore. However, since the mill inventory of coarse solids is only 71%, the "specific grinding rate" (i.e., per unit of effective mill power) is calculated by:

$$\frac{33.3 \text{ t/h}}{523 \text{ kwh/h} \times 0.71} = 89.7 \text{ kg/kwh}$$

Note that the specific grinding rate represents the grinding rate from plus 106 μm to minus 106 μm material in the ball mill per unit of effective mill power. As this parameter will still include the grindability characteristic of the ore, the true effectiveness of the mill in terms of its functional objective of size reduction of plus 106 um material can be determined by factoring out the grindability of the ore itself as determined, for example, in the Bond laboratory test. This yields the "grinding rate ratio", or the specific grinding rate in the plant ball mill relative to the grinding rate in a standard test mill under standard test conditions.

$$\text{Grinding Rate Ratio} = \frac{89.7 \text{ kg/kwh}}{2.31 \text{ g/rev}} = 38.8$$

This ratio represents the grinding or size reduction effectiveness of the mill environment after the grindability characteristic of the ore (as measured in a standardized test grinding to a similar product size) is taken into account.

It may be noted that the relative grinding rates in the plant and test mills are at two slightly different reference product sizes. The reference size for the plant mill is the nearest standard screen size to the prevailing circuit P80, in this case 106 μm (150 mesh). The Bond test circuit was operated with a closing screen size of 150 μm (100 mesh), which provides a test circuit P80 product size as close as possible to that of the plant. The amount of new minus 106 μm material produced in the test mill could be calculated and used to define the grinding rate ratio if this refinement was considered necessary. The former method, however, alleviates the problem of screening and interpolation error in the estimate of the test circuit product size distribution. In any case, for similar product size distributions, relative grinding rates (or grindabilities, as defined in the Bond test procedure) calculated at different mesh sizes will be similar.

The classification efficiency and specific grinding rate for a particular ore as defined above, when combined with the mill power, define the production rate of the circuit, as follows.

$$\text{Mill Power Draw} \times \text{Classification System Efficiency} \times \text{Specific Grinding Rate} = \text{Circuit Fines Production Rate}$$

For survey no. 1:

$$527 \text{ kw} \times 0.73 \times 85.8 \text{ kg/kwh} = 33.0 \text{ t/h}.$$

Grinding Rate Ratio = 41.2

For survey no. 2:

$$523 \text{ kw} \times 0.71 \times 89.7 \text{ kg/kwh} = 33.3 \text{ t/h}.$$

Grinding Rate Ratio = 38.8

From survey no. 1 to survey no. 2, it can be seen that there was a slight loss in both the measured classification efficiency and breakage efficiency. These are reflected in an overall loss in work index efficiency of the circuit. Although the differences between the figures are barely significant, they collectively demonstrate that only minor differences in circuit classification and breakage efficiency existed between the two sampled conditions.

4.5.2.2 Ball Milling at Kidd Creek Mines

Mineral Distributions in Ground Products

When the final copper concentrate and tailings are reconstituted in their relative proportions, the final product mineral distributions from grinding, excluding zinc regrinding, can be back calculated. This was done for both of the sets of data from runs no. 1 and 2, and is reported in Table 4-11. The cumulative mass distributions for both copper and zinc for the two sample runs are also plotted in Figure 4-25.

Table 4-11. Mineral Distributions in Ground Products, Kidd Creek Surveys No. 1 and 2

Size Class (μm)	Survey No. 1 Cu	Survey No. 1 Zn	Survey No. 2 Cu	Survey No. 2 Zn
+75	1.6%	3.1%	4.8%	4.3%
53-75	2.6	5.3	9.2	7.4
38-53	14.5	16.8	20.3	18.0
22-38	17.8	16.5	15.2	14.7
16-22	14.7	13.9	12.4	12.7
10-16	12.7	11.1	10.1	11.3
7-10	8.6	7.4	6.8	7.5
-7	27.5	25.9	21.2	24.0
	100.0%	100.0%	100.0%	100.0%

By applying the size by size recovery curves given in Figures 2-4 and 2-5 to these mineral distributions, the overall recovery of each to the copper concentrate can be calculated. By considering more generally derived size-recovery relationships, as discussed in section 4.3, the overall relationship between fineness of grind and recovery to the concentrate

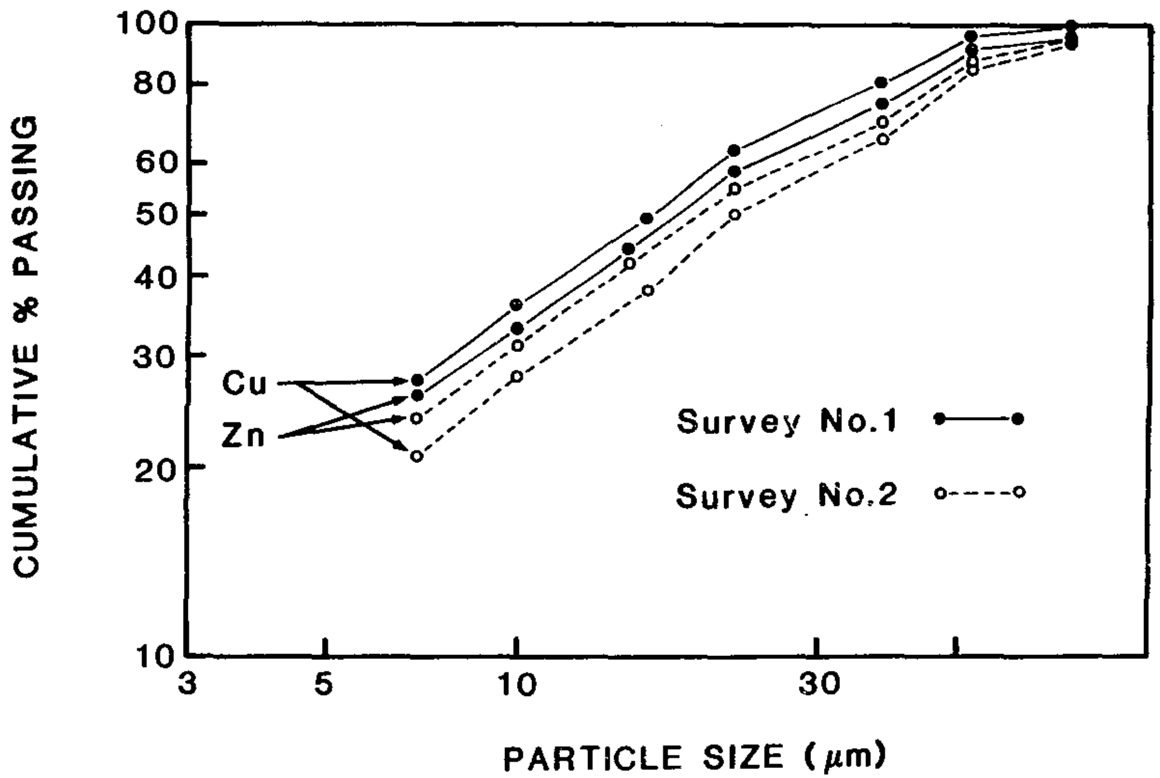


Figure 4-25. Mineral Distributions in Ground Products, Kidd Creek Surveys No. 1 and 2

streams can be explored. Concentrate values will be based on the specified smelter contact terms. By then introducing operating costs, a more general model of plant economic efficiency which considers both flotation and grinding circuit performance can be derived. The complexities of this are beyond the scope of the present investigation, but this is an important subject for further development work.

The substantially finer overall grind during survey no. 2 is reflected in the mineral size distributions. Note, however, the reversal in the relative finenesses of copper and zinc minerals from survey no. 1 to survey no. 2. The cause for this is open to speculation, but may reflect a grain boundary effect which is predominant for the zinc mineral, or simply more preferential grinding of copper mineral for that particular circuit configuration.

Primary Ball Milling Circuit Performance

Operating and Laboratory Work Indices:

The relevant data for calculation of Bond operating work indices are given in Table 4-12, as are the results from Bond laboratory ball mill work index tests. The work index efficiency shows a slight increase of about 5% from survey no. 1 to survey no. 2, which again is just barely statistically meaningful.

Table 4-12. Primary Ball Mill Circuit Performance Summary

Kidd Creek Surveys No. 1 and 2

Survey No.	1	2
Date	Dec. 18/85	Dec. 19/85
Circuit Feed		
New Feed Rate (DMTPH)	126.6	131.8
Feed Size, F80 (um)	1460 µm	1650 µm
% - 106 µm (150 mesh)	28.6%	27.5%
Test W.I. @ 150 µm (kwh/t)	12.7	13.6
Bond Test Grindability (g/rev)	2.07	1.86
Circuit Product		
Product Size, P80 (µm)	126 µm	149 µm
% - 106 µm (150 mesh) in Product	76.2%	72.1%
Circuit Performance		
Power Draw (kw)	955	924
kwh/t	7.54	7.01
Operating W.I. (kwh/t)	12.0	12.2
W.I. Eff. (Test/Op.), %	106%	111%
New - 106 um (DMTPH)	60.3	58.8
106 um Classification Eff.	65.6%	69.3%
106 um Specific Production Rate (kg/kwh)	63.1	63.6
106 um Specific Grinding Rate, SGR (kg/kwh)	96.2	91.8
Grinding Rate Ratio (SGR/Bond Gr.)	46.5	49.4

Circuit Classification Efficiencies:

The functional objective of the primary ball mill circuit operation may be defined once again as the production of minus 106 µm (150 mesh) material. The fines inventory of the ball mill can be estimated as follows.

	Survey No. 1	Survey No. 2
	(% - 106 µm)	(% - 106 µm)
Ball Mill Feed:	28.9	26.2
Ball mill Discharge:	39.9	35.2
Average Relative Fines Inventory:	34.4%	30.7%

The overall circuit classification efficiency is the relative inventory of coarse solids, or 65.6% and 69.3% respectively.

Specific Grinding Rates and Rate Ratios:

From survey no. 1, the circuit feed contains 28.6% minus 106 um and the circuit product contains 76.2% minus 106 um material. The new minus 106 um production rate is (.762 - .286) x 126.6 t/h feed, or 60.3 t/h. On the basis of power consumed, this represents:

$$\frac{60.3 \text{ t/h}}{955 \text{ kwh/h}} = 0.0631 \text{ t/kwh, or, } 63.1 \text{ kg/kwh.}$$

This is the "specific production rate". Since the mill inventory of coarse solids is only 65.6%, the "specific grinding rate" (i.e., per unit of effective mill power) is calculated by:

$$\frac{60.3 \text{ t/h}}{955 \text{ kwh/h} \times 0.656} = 96.2 \text{ kg/kwh}$$

The grinding rate ratio is calculated as follows:

$$\text{Grinding Rate Ratio} = \frac{96.2 \text{ kg/kwh}}{2.07 \text{ g/rev}} = 46.5$$

Note once again that:

Mill Power x Classification System Efficiency x Specific Grinding Rate = Circuit Fines Production Rate

For survey no. 1:
955 kw x 0.656 x 96.2 kg/kwh = 60.3 t/h.

Grinding Rate Ratio = 46.5

For survey no. 2:
924 kw x 0.693 x 91.8 kg/kwh = 58.8 t/h.

Grinding Rate Ratio = 49.4

From survey no. 1 to survey no. 2, the classification efficiency was noted to increase from about 66 to 69%. At the same time, the grinding rate ratio increased from 46.5 to 49.4. These factors are reflected in an overall increase in work index efficiency of the circuit.

Secondary Ball Milling Circuit Performance

Table 4-13 provides similar data on the secondary ball milling circuit performance during surveys no. 1 and 2. Unfortunately, the Bond ball mill work index test at 75 µm (200 mesh) for survey no. 1 was not completed successfully. As well, the actual secondary ball mill circuit feed, consisting of primary copper rougher flotation tailings, plus copper regrind circuit product, was not subjected to grindability testing. In hindsight, a combined sample large enough for such a purpose would have been appropriate.

The plus 75 µm (200 mesh) solids in the secondary ball mill circuit feed is dominated by primary ball mill discharge material, of which only 4% reports to copper concentrate in

Table 4-13. Secondary Ball Mill Circuit Performance Summary

Kidd Creek Surveys No. 1 and 2

Survey No.	1	2
Date	Dec. 18/85	Dec. 19/85
Circuit Feed		
New Feed Rate (DMTPH)	149.4	126.5
Feed Size, F80 (μm)	116 μm	147 μm
% - 270 m (53 μm) in Feed	62.1%	53.9%
Test W.I. Equivalent, Approx.	-	12.2
Bond Test Grindability (g/rev), Equivalent, Approx.	-	1.59
Circuit Product		
Product Size, P80 (μm)	50 μm	68 μm
% - 270 m (53 μm) in Product	82.9%	73.7%
Circuit Performance		
Power Draw (kw)	775	767
kwh/t	5.19	6.06
Operating W.I. (kwh/t)	10.7	15.6
W.I. Eff. (Test/Op.), %, Approx.	-	78%
New - 53 μm (DMTPH)	31.1	25.0
53 um Classification Eff.	60.3%	58.0%
53 um Specific Production Rate (kg/kwh)	40.1	35.6
53 um Specific Grinding Rate, SGR (kg/kwh)	67.0	56.2
Grinding Rate Ratio (SGR/Bond Gr.), Approx.	-	35.3

rougher flotation, thereby bypassing the secondary milling circuit. The ore grindability of the secondary circuit feed is therefore reasonably well represented by the work index over the range from 100 and 200 mesh as determined by the two work index tests, as follows. Let W_1 represent the work applied to 100 mesh, and W_{1+2} the work applied to 200 mesh.

100 mesh laboratory W.I. Test: $F80 = 1492\ \mu m$
$P80 = 112\ \mu m$
$WI_1 = 13.6$ kwh/t

$$WI_1 = W_1 \Big/ \left(\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}} \right)$$

$$13.6 = W_1 \Big/ \left(\frac{10}{\sqrt{112}} - \frac{10}{\sqrt{1492}} \right)$$

$$W_1 = 9.3 \text{ kwh/t}$$

200 mesh laboratory W.I. Test: $F80 = 1492\ \mu m$
$P80 = 61\ \mu m$
$WI_{1+2} = 13.1$ kwh/t

$$13.1 = W_{1+2} \left(\frac{10}{\sqrt{61}} - \frac{10}{\sqrt{1492}} \right)$$

$$W_{1+2} = 13.4 \text{ kwh/t}$$

Then, from 100 to 200 mesh, which approximates the grinding range of the secondary circuit;

$$W_2 = 13.4 - 9.3 = 4.1 \text{ kwh/t}$$

The work index in this range is;

$$WI_2 = 4.1 \Big/ \left(\frac{10}{\sqrt{61}} - \frac{10}{\sqrt{112}} \right) = 12.2 \text{ kwh/t}$$

The equivalent grams per revolution is calculated from the relationship between the test work index and the grindability at 200 mesh, namely;

$$\frac{K}{1.45^{0.82}} = 13.1 \quad kwh/t$$

Solving, $K = 17.8$

For the above WI_2;

$$17.8 \,/\, Grp^{0.82} = 12.2 \quad kwh/t$$

Solving, $Grp = 1.59$ gms/rev.

This figure can be used to calculate the approximate work index efficiency and grinding rate ratio, as shown in Table 4-13.

Although comparative ore grindabilities are not available for these two tests, the large increase in operating work index from survey no. 1 to no. 2 exceeds that likely due to an ore change alone, indicating a serious loss in circuit efficiency.

Note that the circulating load ratio dropped from about 230 to 120 percent from survey no. 1 to survey no. 2. Traditional reasoning would suggest that this is the cause for the increase in operating work index. However, the measured change in classification efficiency was actually quite small. The specific grinding rate, on the other hand, dropped substantially. This indicates that the breakage environment, rather than the classification system, is the likely problem area. A comparison of the mill discharge samples reveals a possible source of the problem, as shown in Table 4-14.

Table 4-14. Secondary Ball Mill Discharge Slurry Characteristics, Kidd Creek Surveys No. 1 and 2

	Survey No. 1	Survey No. 2
% solids by weight	71.6	73.2
Solids S.G.	3.6	3.2
% solids by volume	41.0	46.0
% -400 mesh	26	34

Assuming the samples reflect the relative nature of the mill contents, both a significant increase in percent solids (by volume) and fineness (minus 400 mesh) have occurred from survey 1 to 2. These changes could cause an increase in slurry viscosity, resulting in a reduction in the breakage rates (Klimpel, 1984).

4.5.3 Preliminary Economic Analysis of Grind Product Size vs Tonnage at Les Mines Selbaie

The operating data that has been collected can be used to provide an approximation of the best product size (P80) from the grinding circuit on the basis that (a) survey no. 2 is a typical operating condition which reflects average operating costs, (b) that overall recovery can be estimated from the product size P80, and (c) that the Bond work index relationship is valid for the overall grinding plant. A similar analysis has been performed at Gilbraltar by Steane (1976).

In order to establish the relationship between the grind size and overall recovery, a standard shape for the copper mineral distribution function must be assumed. The distribution modulus, n, was calculated for the cyclone overflow copper mineral distributions for each of the four months (anon., 1981), as shown in Table 4-15. A high value of n means a narrow distribution, and vice versa.

Table 4-15. Distribution Moduli for Copper Distributions

Month		n
Sept., 1984		1.326
Oct., 1984		1.184
Nov., 1984		1.036
Dec., 1984		1.005
	Mean	1.138

Since the distribution function for the October data was closest to the mean, it was selected as the most typical. This distribution was plotted on a large scale graph and a series of parallel lines, representing coarser and finer overall distributions, then drawn. The relative copper masses on individual mesh sizes for the different size distributions could then be estimated. Overall recoveries for the different distributions were then calculated based on mean recoveries of each particle size range from Table 2-2. The results are plotted in Figure 4-26.

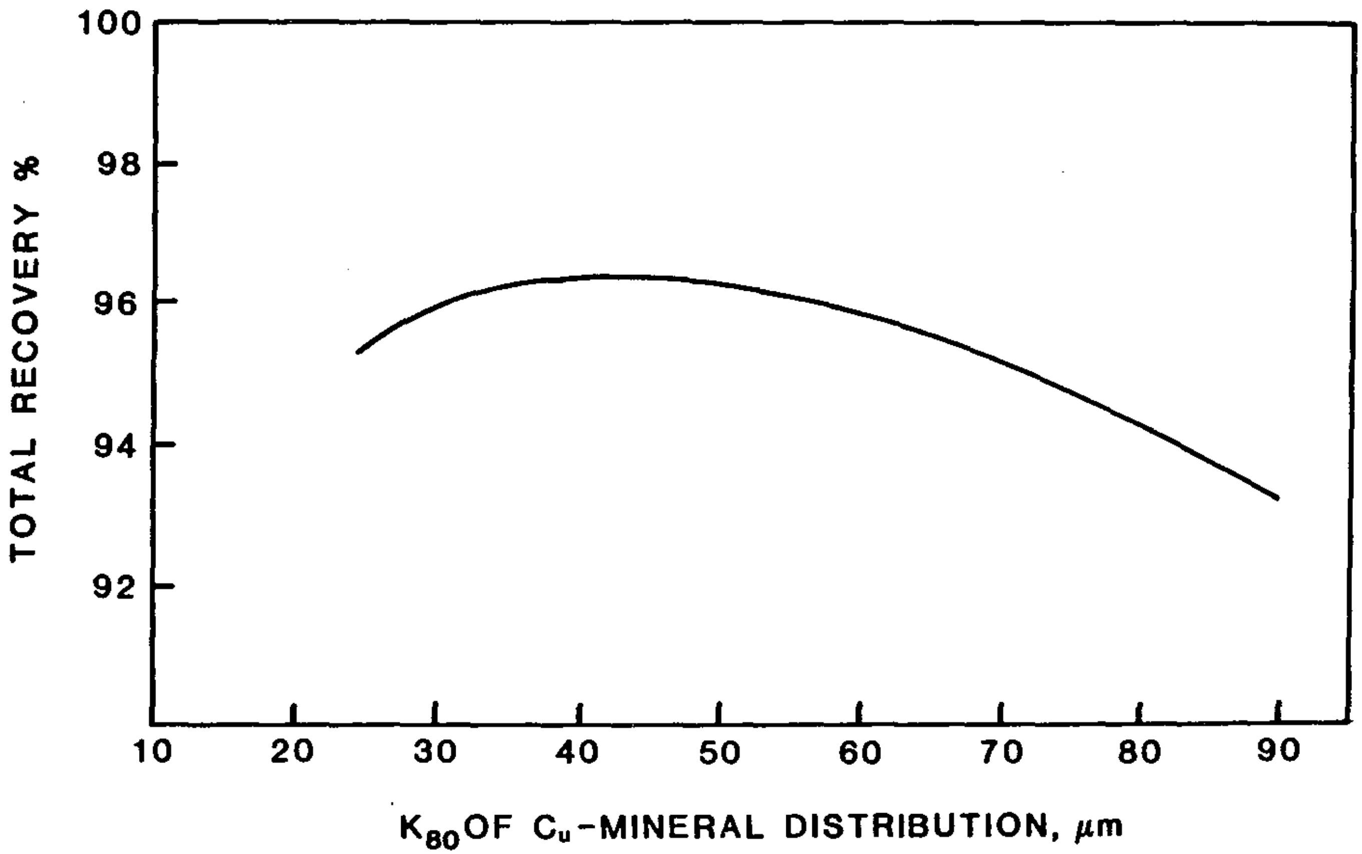


Figure 4-26. Les Mines Selbaie Total Recovery vs. K80 of Copper Mineral in Flotation Feed

The ratios of the solids P80 to the copper mineral K80 in the composite cyclone overflow samples for each of the four months are given in Table 4-6. The average ratio of 1.74, which was also the actual value during survey 2, was used to generate the estimated recoveries for the different overall grind P80s that are shown in Table 4-16.

Table 4-16. Recovery vs. Solids P80 and Copper K80 in Cyclone Overflow

K80 Copper Dist (μm)	Approx. P80 C.O.F. (K80 Copper X 1.74)	Calculated Overall Recovery %
90	157	93.08
74	129	94.68
68	118	95.08
64	111	95.46
56	97	96.03
45	78	96.26
38	66	96.28

The overall operating work index (rod and ball mill combined) was calculated to be 12.0 kwh/t. Direct unit grinding costs (power and media) at 70.3 DMTPH were taken as $0.97 per ton. The estimated product value per unit feed is based on the payable (assumed 65%) copper values only, which at 3.24% copper in the feed and $2.205/kg copper is the recovery times $46.44 per ton of feed. Daily plant revenue is based on 23 hours of operating time. The results are shown in Table 4-17.

Table 4-17. Preliminary Economic Analysis of Grinding Product at Les Mines Selbaie

Rod Mill Feed Tonnage	Product P80 (μm)	Mineral K80 (μm)	Product Value per Unit Feed	Direct Unit Grinding Costs	Net Value per Unit Feed	Daily Plant Revenue
55	73	42	44.70	1.24	43.46	$55,000
58	81	47	44.70	1.18	43.52	$58,060
61	89	51	44.67	1.12	43.55	$61,000
64	97	56	44.58	1.07	43.51	$64,050
67	105	60	44.47	1.02	43.45	$66,950
70.3 (Survey no.2)	115	66	44.23	0.97	43.26	$69,950
73	123	71	44.07	0.93	43.14	$72,430
76	132	76	43.88	0.90	42.98	$75,130
79	142	82	43.65	0.86	42.79	$77,750
82	151	87	43.37	0.83	42.54	$80,250

The results show that although recovery increases down to a grind product size P80 of approximately 80 µm, increasing unit grinding costs mean that the maximum net value per ton processed is obtained at a grind of approximately 90 um. Increasing tonnage increases daily plant revenue up to a tonnage that is both more than the grinding circuit can likely handle, and for which the flotation recovery characteristic used in the calculations is probably invalid. Note, however, that this analysis indicates that the value per unit of flotation feed decreases at higher tonages (for example, $0.47 is lost per ton processed at 79 versus 70 tons per hour). This means that increased daily plant revenues at higher tonnage must be offset by future lost revenues (recoverable reserves) that have reported to tailings. Depending on the mine life (the discount period) and the discount rate applied, the present value of these losses can effect the optimum milling rate for the operation.

When milling capacity is limited by the mining rate, as it is at Selbaie, net value per unit processed should be maximized. This means the target product size for the grinding circuit should be set at approximately 90 um, and the circuit adjusted and operated to achieve as close to this target as possible. These figures thus support the actual operating objectives used at Selbaie (Wood, 1987). Any significant change in grinding costs or the metal price will cause this target to change (Finch, 1976).

4.5.4 Selection of the Optimum Economic Circulating Load for Primary Ball Milling at Kidd Creek Mines

4.5.4.1 Introduction

Because maintenance costs on the primary grinding circuit cyclone feed pumps are very high, it was suggested that operating the circuit at a reduced circulating load could result in a net reduction in operating costs, even though it was recognized that there will be an offsetting loss in grinding efficiency. This analysis was therefore carried out to arrive at an estimate of the optimum economic circulating load, and the operating and design conditions under which it will be achieved. One of the requirements of this evaluation is to define the needed design modifications to the classification equipment.

4.5.4.2 Process Application Design of Hydrocyclones

Process application design of any hydrocyclones (Arterburn, 1982; McIvor, 1984b) must be divided into two steps, namely;

a) determination of the basic process specifications (separation characteristic and capacity requirements); and,

b) selection of equipment dimensions and number of units to meet the process specifications under desirable operating conditions.

The basic process specifications can be defined by completion of two material balances around the classifiers, namely;

a) solids and water masses in each of the three (feed, overflow, and underflow) streams; and,

b) size distributions of solids in each of the three streams.

The sequence of the general procedure is therefore:

1. Complete the solids/water balance.
2. Estimate the size distributions.
3. Calculate the cyclone cut size (d50c) requirement from the size distributions.
4. Select the cyclone dimensions that will give the required cut size at reasonable feed pressure. Feed pressure is usually specified at 35-70 kPa to minimize energy requirements, on the one hand, and to ensure stable operation, with some degree of flexibility, on the other.

   At this point all dimensions except apex size, (to which cut size is fairly insensitive) are first selected.
5. Based on the slurry feed capacity of a single unit of the stated dimensions at, say, 60 kPa feed pressure, the number of units to handle the entire flow is calculated.
6. The apex size is selected to handle the anticipated underflow rate at the desired percent solids.
7. Minor adjustments to dimensions and feed pressure are made to fine tune the dimensions (for cut size and apex capacity) and the number of units (for total feed capacity).

The above procedure applies to all hydrocyclone application engineering exercises, including closed-circuit grinding. Specific details relevant to this procedure as applied to closed circuit grinding are described by McIvor (1984a and b). The options of modifying or replacing the existing hydrocyclones can be evaluated by comparing their current performance with the new process performance specifications.

4.5.4.3 Process Performance Specifications for Grinding Circuit Hydrocyclones

General

The specifications for the hydrocyclones for this application are predicated by selection of the most suitable design circulating load for the circuit.

It is clearly desireable to produce the narrowest possible valuable mineral distribution by size for flotation feed. It is known that the K80 of the denser mineral is generally finer than that for the overall solids in the cyclone overflow, and that the circulating loads of denser minerals can be several times that of the overall solids (Finch and Matwijenko, 1977). As previously discussed, it has also been observed that dense minerals in the classifier overflow exhibit significantly narrower size distributions than the overall solids (Myers and Bond, 1957; Dean and Baker, 1984). This is also shown in Figure 4-24. This suggests that the narrowness of the heavy mineral distribution approaches the limit imposed by the breakage characateristics of the ore at any reasonable overall ball mill circulating load. For this analysis, it was therefore assumed that the slope of the mineral distribution would be independent of circulating load ratio over a reasonable range. The circuit product monetary value would therefore not be affected as long as the K80 was held constant. The optimum circulating load could therefore be selected on the basis of relative grinding circuit operating costs.

Constraints on Circulating Load

Two possible constraints on the achievement of the desired circulating load which cannot be overcome by classification system adjustments are those imposed by ball mill overload conditions, or by low circuit reduction ratio. The first has been widely suggested to be related to flow velocity through the mill (McIvor and Rimmer, 1983), which limit can be roughly estimated from overload conditions experienced at two substantially different operations, namely Bougainville (Gauvin, 1978) and Braden (Taggart, 1945). See Table 4-18.

The limiting flow velocity in these installations is almost double an equivalent figure of 4.3 m/min. (14 ft./min) previously suggested by Weller (1980). However, using 7.6 m/min. as the approximate limiting criterion for the Kidd Creek primary ball mill, the corresponding limiting circulating load at 127 mt/h of new feed could be calculated as follows.

Mill size: 3.66 m dia. by 5.49 m long

Residence time: $\dfrac{5.49 \text{ m}}{7.6 \text{ m/min.}}$ = 0.72 minutes

Table 4-18. Ball Mill Overload Conditions at Bougainville & Braden

	Bougainville	Braden
a) Ball mill size:	5.49 m dia. x 6.40 m long (18' dia. x 21' long)	2.44 m dia. x 3.66 m long (8'dia. x 12'long)
b) Mill throughput rate: (at overload)	2440 M.T.P.H. (2500 S.T.P.H.,) @ 78% solids	417 M.T.P.H. (460 S.T.P.H.,) @ 75% solids
c) Slurry throughput rate:	1590 m^3/hr (7,000 U.S.G.P.M.) (935 c.f.m.)	340 m^3/hr (1500 U.S.G.P.M.) (200 c.f.m.)
d) Mill slurry volume: (40% charge level 40% voids in charge)	21.5 m^3 (760 ft.)^3	2.72 m^3 (96 ft.)^3
e) Mean residence time:	0.81 min.	0.48 min.
f) Flow velocity: (mill length/ residence time)	7.9 m/min. (26 ft./min)	7.6 m/min. (25 ft./min)

Mill slurry volume: (40% charge level) 8.47 m^3

Slurry throughput rate: $\frac{8.47\ m^3}{0.72\ min.} = 11.76\ m^3/min.$

Mill throughput rate: 1120 mt/h at 75% solids

This corresponds to a circulating load of almost 900%, which is very likely far in excess of the economic optimum.

The data in Table 4-19 indicate a second constraint, that low circuit reduction ratios and low circulating load ratios go hand in hand. This might well be suspected, since a large portion of new circuit feed will report diretly to the classifier fine product stream.

When the reduction ratio is about 4 or greater, as is the case in this application, obtaining a high circulating load is feasible.

Table 4-19. Low Reduction Ratio Ball Milling Circuits

	Circuit	Red. Ratio (F80/P80)	Circulating Load Ratio	Verified*
1.	Miami (Taggart, 1945)	1.3	0.4	
2.	Mt. Lyell (Taggart, 1945)	1.4	0.2	Yes
3.	Mt. Lyell (Taggart, 1945)	1.7	0.50	Yes
4.	Mex (Taggart, 1945)	2.0	0.6 - 1.2	
5.	Kidd Secondary (present work)	2.0	1.3	Yes
6.	Brunswick Secondary (del Villar, 1987)	2.0	1.3	Yes
7.	Heath Steele (Houdoin et al, 1978)	2.0	2.0	Yes
8.	Heath Steele (Houdoin et al, 1978)	3.2	3.2	Yes
9.	Id. Mary (Taggart, 1945)	3.0	0.4	Yes
10.	United E. (Taggart, 1945)	3.0	1.3	
11.	United V. (Taggart, 1945)	3.0	1.5	Yes
12.	Pb-Zn (Taggart, 1945)	3.0	4.8	
13.	Permanente (Taggart, 1945)	3.0	5.0	
14.	Bunker (Taggart, 1945)	4.0	7.0	Yes
15.	BHS (Taggart, 1945)	5.0	3.3	Yes
16.	ASR (Taggart, 1945)	5.0	3.5	

* Reported circulating load ratio checked by reported size distribution data.

Optimum Economic Circulating Load

The relationship between circulating load and circuit capacity as shown by Davis (1925) in Figure 4-3 can be related to the mill fines inventory, as discussed earlier. As circulating load increases, the mill inventory size distribution becomes coarser, and more mill volume is effectively applied to size reduction of coarse (i.e., larger than desired product size) particles. Because the coarse particle inventory relates directly to mill capacity (or circuit efficiency), it provides a quantitative parameter for economic comparisons of operating at different circulating loads. Reducing grinding costs can be weighed against increasing circulating load material handling costs until the economic optimum is found. Because power and media costs for grinding outweigh pumping costs considerably, the optimum is normally several hundred percent.

For this case, the economic comparison for selection of the design circulating load was performed as follows. A base for comparison was arbitrarily chosen at 450% circulating load. A new feed rate of 127 mt/h, which corresponds to that at the time of sample survey no. 1, was also used as the basis for comparison. Only direct, variable grinding and pumping costs were considered. Grinding costs (for power and steel) for the base case were taken to be 53.6¢/ton.

Direct variable cyclone feed pumping costs were considered to consist of power and maintenance costs. From the pump performance curve, the "water" horsepower for pumping is about 48.5 kw at 517 m^3/hr. and a total dynamic head of 18 m, which is close to the 450% circulating load condition. With a slurry S.G. of 1.94, the power draw of the pump motor will be 94 kw, or approximately 0.7 kwh consumed per ton processed. At 2.94c/kwh, this equates to 2.1c per metric ton milled.

Yearly no. 24 pump and primary cyclone maintenance costs were given as:

Pumps:	$59,000
Cyclones:	8,000
Total:	$67,000

For a year's production, this equates to approximately

$$\frac{67{,}000.00}{127 \times 24 \times 360} = 6.1¢ \text{ per ton milled}$$

Pumping and classification costs then total 8.2¢ per ton milled.

The fines inventory for circulating loads varying from 250 to 550 percent was calculated from the circuit size distribution data. A simple spreadsheet program incorporating Plitt's (1971) cyclone performance equation was used for the reiterative procedure to estimate the cyclone size distributions (McIvor, 1984a and b). Survey no. 1 was taken as the base operating case. The cyclone model parameters were determined from sampling survey no. 1, as shown in Appendix M.

The hydrocyclone mass balance is worked out for each alternative circulating load using the same cyclone overflow water and solids as in sample survey no. 1, and 75.5% solids by weight in the cyclone underflow. The percent solids by volume in the cyclone feed can then be calculated for each circulating load. The model parameter m was then estimated using the value obtained from sample survey no. 1, and a general trend from a number of field observations (McIvor, 1984a) that show m decreasing with higher percent solids by volume in the feed (Cv), as follows.

Table 4-20. Estimated sharpness of Separation (m) at Different Circulating Loads

Case	C.L. Ratio	Cv (%)	m
1	250	37.7	1.30
2	350	39.6	1.10
SS #1	436	40.8	0.95
3	450	40.9	0.95
4	550	41.9	0.80

The bypass fraction was estimated for each circulating load condition as 5/6 times the water bypass, which was the case for sample survey no. 1. The cyclone size distributions were estimated using the trial and error method described by McIvor (1984a), and the same spreadsheet program described earlier. The cyclone feed size distribution was assumed to maintain the same general shape as observed for sample survey no. 1. The resulting cyclone mass balances and size distribution data for each case are given in Appendix N. The cyclone feed is the ball mill discharge, and the cyclone underflow plus the new circuit feed is the ball mill feed.

Grinding costs were assumed to vary in proportion to the relative power efficiency of the circuit, as determined by the mill "fines inventory" at different circulating loads. In all cases, the rod mill discharge size distribution, the cyclone underflow and overflow densities, and the K80 of the cyclone overflow were held constant. Pumping (and cycloning) costs were assumed to vary in direct proportion to the pumping throughput rate.

A summary of the results is presented in Table 4-21. The fines inventory was taken as the average minus 150 um (100 mesh) material between ball mill feed and ball mill discharge, although another specific mesh size could have been chosen. The relative circuit classification efficiency is the ratio of coarse mill inventory compared to the base case. The results show that total minimum operating cost occurs at approximately 350 percent circulating load, which is indeed somewhat lower than the current figure. This is well below the likely mill overload condition mentioned earlier. Also note that total costs increase rapidly for circulating loads below the optimum due to the serious negative effect on the relative circuit efficiency.

Process Specification Summary

From the 350% circulating load condition, at a new feed rate of 127 DMTPH, the process requirements of the classification equipment can be summarized as follows.

1. Process flows

Feed:	437 m^3/hr at 68.4% solids by weight Slurry S.G. = 1.91
Overflow:	158 m^3/hr at 51.5% solids by weight
Underflow:	279 m^3/hr at 75.5% solids by weight

Note that apex size should be selected to maximize cyclone

Table 4-21. Economic Analysis for Selection of Circulating Load, Kidd Creek Primary Ball Milling

Case	Cyclone d50c	C.L. Ratio (%)	Fines Inventory % -100 mesh	Class. Eff. %	Relative Circuit Eff. (%)	Costs (¢/ton) Milling	Pumping	Total
1	136	250	46.6	53.4	93.4	57.4	5.2	62.6
2	116	350	43.9	56.1	98.1	54.6	6.7	61.3
3	94	450	42.8	57.2	100.0	53.6	8.2	61.8
4	88	550	41.4	58.6	102.4	52.3	9.7	62.0

underflow density (without roping) through its wear life. An underflow density of close to 80% solids should be obtainable in this application (see Appendix P).

2. Classifier cut size: d50c = 116 µm.

4.5.4.4 Hydrocyclone Modifications

Selection of hydrocyclones for specific grinding circuit process specifications can be carried out from information provided in the literature (Arterburn, 1982; McIvor, 1984b). However, in this case, in order to determine the modifications needed it is appropriate to scale from process performance information on the existing equipment.

From the hydrocyclone performance information reported from sample survey no. 1, the cyclone cut size characteristic can be related to cyclone dimensional and operating characteristics using Plitt's (1976) model. The constant, K, in this equation can be calculated from the given performance data as follows. Using the units specified for this equation (Plitt, 1976):

$$d50c = \frac{K \times Dc^{0.46}\ Di^{0.6}\ Do^{1.21}\ e^{0.063\phi}}{Du^{0.71}\ h^{.38}\ Qi^{45}\ (\rho s-\rho_1)^{0.5}}$$

d50c is cut size, µm
Dc, Di, Do, Du, and h are cyclone dimensions, inches
Q is flow rate, c.f.m.
ϕ is feed solids by volume, percent
ρs is the S.G. of solids
ρl is the S.G. of liquid

$$126 = \frac{K\,(13.75)^{0.46}\,(3.625)^{0.6}\,(2.50)^{1.21}\,e^{.063 \times 40.8}}{(3)^{0.71}\,(40)^{.38}\,(49.6)^{.45}\,(3.3 - 1.0)^{0.5}}$$

Solving, K=26. An average apex (or underflow spigot) dimension (Du) over its wear life was used in the above equation.

The reduced flow rate for the new circulating load suggests reducing the number of operating cyclones to 5. As well, the average underflow density will be substantially increased by staying with the same apex sizing. To determine the vortex finder sizing for the new cut size of 116 um, we can go back to the above equation, using a value of K=26.

$$116 = \frac{26 \times 13.75^{0.46} \times 3.625^{0.6} \times Do^{1.21} \times e^{0.063 \times 39.6}}{3^{0.71} \times 40^{0.38} \times 54.3^{0.45}\ (3.3 - 1)^{0.5}}$$

Solving, Do = 3.2 inches, or 81 mm.

To summarize, the 5 operating cyclones with 81 mm vortex finders will achieve the desired circulating load of 350%. The cyclone feed pumping rate will be approximately 437 m^3/hr., with a feed slurry density of 1.91. Cyclone feed pressure will remain at close to 83 kPa. Final adjustments should be possible through the cyclone feed water addition rate.

Note that the above cyclone modifications should be accompanied by:

a) a slight reduction in the pump speed, and,

b) an increase in the mill charge level to obtain about a 2% increase in power draw to account for the loss in grinding efficiency at the reduced circulating load ratio.

4.5.5 Ball Mill Circuit Control

4.5.5.1 Les Mines Selbaie

The Need for Automatic Control

The general requirements and rationalization for automatic control of ball milling circuits are described in section 4.4. The need for automatic control for the Selbaie grinding circuit can be associated with the following factors.

A. Ore Grindability and Power Draw Variations

The circuit is normally operated at a feed tonnage to meet production requirements, and at a feed size well controlled by the crushing plant. However, work index variations of over 15 percent for both rod and ball milling have been measured. Coupled with rod mill power draw variations in the order of 5% between rod additions, the workload on the ball mill could vary quite easily by 20%.

B. Hydrocyclone Instability

In addition to the need for a control system to maintain operation at optimum design conditions (for example, at the desired circulating load), cyclone surging was often noted as a result of a low cyclone feed sump level. Sampling through the surging cycle (Appendix O) revealed variation in the underflow density from 63 to 77 percent solids by weight, with corresponding variations in size distribution of 40 to 24 percent minus 106 um (150 mesh). This compares to an average of about 21% minus

106 um during stable operation, and directly reflects the negative effect on classification system efficiency.

C. Surge Tank (Crusher) Fines

Although the crushing plant fines do not normally require an extensive amount of grinding, (K80 of 150 to 250 um) their substantial mass and high mineral content, as well as the large influx of associated water, cause a serious change in tonnage, volume, and total copper to the flotation circuit. As well, the grinding circuit performance data indicates that higher water input may increase the ball mill circuit efficiency, and the operators sometimes respond by increasing the rod mill feed rate when a feed rate reduction is more appropriate. Typical variations in flotation circuit feed characteristics during and without crusher plant operation are summarized below, with typical values of 3% copper in rod mill feed and 5% copper in crusher fines.

Crusher ON	Solids tonnage:	78 DMTPH
	Slurry volume:	129 m^3/h
	Cu in feed:	2.60 DMTPH
Crusher OFF (No adjustment)	Solids tonnage:	65 DMTPH
	Slurry volume:	108 m^3/h
	Cu in feed:	1.95 DMTPH

As a preliminary measure before a full circuit control plan is implemented, the operators should be advised to (a) increase the mill feed rate by approximately 10 TPH, and (b) allow the cyclone overflow density to fall about 2 percent, when the crushing plant is shut down, and vice-versa. This would result in the following flotation feed conditions with a minimum effect on grind size.

Crusher <u>OFF</u>	Solids tonnage:	75 DMTPH
(With adjustment)	Slurry volume:	134 m^3/h
	Cu in feed:	2.25 DMTPH

The influx of crushing circuit fines and water has no major negative effect on ball mill circuit classification or breakage efficiency, going by the by data from surveys no. 1 and 2. Although originally perceived as a design problem, this intermittent stream can now be seen as a subject of concern for the stabilizing control system for the ball mill circuit.

D. Ball Mill Overloading

Ball mill overload is characterized by a noticeable drop in power draw, accompanied by an extremely high flow of dense mill discharge. The loss in power draw is apparently caused by media charge swelling and centrifuging due to the build up of a thick, coarse slurry in the mill. It has been shown that mill hold-up volume increases with throughput rate, and that grinding rate suffers as a result (Austin et al, 1984). Because grinding slugs have a lower natural void space (about 4/5 than that of balls (McKim, 1987)), they may also contribute to mill overloading tendencies.

When the mill throughput rate (the circulating load) and feed density are too high, overloading may occur. These two factors are linked because higher solids flow through the fixed size apexes also means higher densities. When overloading occurs, the operator will increase mill feed water, and cut back (or off) the rod mill feed. Lost tonnage is small, but the upset to the grind product size, density, and feed rate to flotation may be significant.

E. Circuit Product Sizing

Circuit product sizing may be affected in several ways by lack of circuit control. First, the optimum product size (P80) in cyclone overflow at any moment in time cannot be maintained. Secondly, with poor classification the product size distribution will be broader at any time, with more extreme fines and coarse particles for the flotation circuit to deal with. Variations in the product size over time will also result in a wider distribution over an extended time period. Finally, less efficient grinding circuit operation will mean a coaser than possible grind size which could reduce total recovery in flotation.

Control System Requirements

Provisions for the control loops needed to implement the B grinding circuit control system are described as follows. Note that the two fundamental objectives of the system described in section 4.4 are:

1) to maintain desired circuit product sizing by maintaining constant classifier feed conditions; and
2) to maintain high grinding circuit efficiency by adjusting the new feed rate to maintain a desired level of circulating load.

The primary control loops needed to implement this system are shown in Figure 4-18.

The optimum economic circulating load has been estimated at approximately 300%, or 210 DMTPH based on a new circuit feed rate of 70 mtph. Note that for constant classifier feed conditions, the absolute circulating load value (rather than a ratio of the new feed rate) must be specified. The ultimate target grind size P80 can be taken as approximately 90 um for this circuit.

Control Loop No. 1:

The short-term control of cyclone overflow product size (overall K80) is through the cyclone feed density. A density gauge installed on the cyclone feed line, could provide the signal to control the flow rate of water (or water plus crusher fines) from the surge tank. A flow control valve with an actuator would be required on the line from the surge tank, or some other means to control the flow rate.

Control Loop No. 2:

The control of the circulating load is through the cyclone feed density and flow rate. The density may be controlled by loop no. 1. The cyclone slurry feed rate would be held constant by a constant sump level, which will be maintained by adjusting the new feed rate into the circuit. A sump level meter would be required on the cyclone feed sump. The signal from the level meter would adjust the rod mill feed rate set point, and hence the rod mill feed rate with the existing controller.

Note that the ball mill feed water can be held constant to maintain desired ball mill operating density when the circulating load is controlled at a constant value. This could be done manually, unless variations in water pressure are significant enough to warrant automatic flow control of this stream.

Also note that pump variable speed capability would not be used in this system. It could be reserved as an override for extreme high or low sump conditions which the control system does not accomodate, or to adjust for pump wear over the life of the liners and impellor.

Control Loop No. 3:

When the crushing plant is not running, the surge tank should be maintained at a suitable minimum level. This could allow the tank to provide needed surge capacity when the crushing plant is started, and a large flow of crusher fines and water is sent over from the screw classifier, while permitting the grinding circuit to call for only as much water as is needed to maintain the desired cyclone feed density.

A level meter on the surge tank could be used to control the make up water rate into the tank, through a flow control valve, installed at a point upstream before the make up water enters the spiral classifier overflow feeder pipe to the surge tank. The same level meter could provide the signal for an override to permit the grinding circuit to take more flow if the surge tank sump level gets dangerously high.

Preliminary Economic Evaluation

New equipment requirements for the above system consist of a density gauge, two level meters, two flow control valve and actuator arrangements, plus three controllers and associated electronics, overrides, etc. Installation and start-up costs must also be considered. The total was estimated to be in the order of $150,000.00.

From historical operating experience, the implementation of automatic control in grinding circuits has provided an average increase in operating efficiency in excess of 5%. This is most often exploited by increasing the circuit throughput rate. As well, reduction in the variation in the feed size to the flotation circuit has contributed to improvements in recovery in the order of 0.5 to 1.5 percent (Horst and Bender, 1979).

For the B grinding circuit, the expected operational life of this part of the plant will be an important aspect of the evaluation. The yearly economic improvements that could be expected on the basis of some of the factors discussed above can be reasonably expected to equal or exceed average historical figures.

During hydrocyclone instability due to cyclone surging fines inventory increases by 10 to 15 percent, having an equal negative effect on overall grinding efficiency and cost. On the assumption that this occurs approximately 25% of the operating time, an overall long-term loss of 3 to 4% in efficiency is estimated. Efficiency losses associated with ball mill overloading and varying circulating load can easily bring this total to 5 percent.

Data from survey no. 2 indicated roughly a 0.7 percent increase in copper recovery due to a narrower mineral size distribution (associated with short-term stable grinding circuit operation) compared to monthly composite results. With the added ability to optimize and maintain desired grind product size, a total gain of approximately 1% in copper recovery should be reasonably attainable.

Quantitatively, these improvements represent:

a) 5% of ball milling steel and energy costs, or;

   5% x 69.3 ¢ = 3.5 ¢/ton

b) 1% of copper revenues, or, assuming 3% copper head grade, and 65% payable at $2.205/kg. in the smelter contract;

   1% x 30 kg/t x $2.205/kg x 65% = 43.0 ¢/ton

Yearly, the total represents a savings of:

   46.5 ¢/ton x 1650 MTPD x 360 days/yr = $275,000/yr.

Very approximately, the investment of $150,000 in process control for the B grinding circuit could have a payback of $1.8 million over 5 years. This would also open the door to further savings through circuit optimization work.

4.5.5.2 Kidd Creek Mines

The Need for Automatic Control

The need for automatic control for the Kidd Creek 'B' zone grinding circuit, aside from longer term circuit optimization work, can be associated with the following factors.

A. Ore and Power Draw Variations

The grinding circuit is normally fed at a tonnage rate to meet monthly production requirements. Work index, rod mill feed size, and rod mill power draw variations, each in the range of 10%, all combined cause a varied workload on the ball mills. With ball milling power draw reasonably fixed, this will yield variations in ball milling efficiency and grind product size.

B. Classification System Performance

In addition to the need for a control system to maintain operation at the optimum economic circulating load, circulating load variations effect hydrocyclone performance, which can have a further detrimental effect on circuit efficiency. For example, the sampling data in Appendix P show that a primary cyclone underflow density of 80% solids by weight

(54% by volume) is achievable, which compares favourably to approximately 77% solids by weight (52% by volume) during surveys no. 1 and 2. This would improve the cyclone efficiency by decreasing the bypass fraction, and thus slightly reduce the ball mill fines inventory. Note that the data in Appendix P also show that fixed ceramic apexes can perform an excellent job of maintaining similar apex dimensions (and underflow densities and size distributions) between hydrocyclones in the cluster arrangement.

Sampling work by the plant staff during 1987 showed overall solids circulating ratios varying from approximately 260 to 440 percent in the three 'A' ore primary grinding circuits (Mars, 1987). This will have an effect on grinding efficiency and economy, as shown in Table 4-21.

Low sump level can also have a detrimental effect on classification system performance by allowing the cyclone feed to surge. This was studied in detail at Les Mines Selbaie (see Appendix O), where it was found that a loss in circuit efficiency in excess of 10 percent was encountered during cyclone surging.

C. Ball Mill Slurry Rheology

It has been shown that ball mill operating density has a direct influence on the breakage rates of solids inside the mill (Klimpel, 1982-83). This is also related to the classification equipment performance, which influences mill feed density by the apex sizing and mass flow of circulating load.

As discussed in section 4.5, extremes in pulp rheological characteristics can result in severe losses in grinding efficiency.

D. Ball Mill Overloading

Although ball mill overloading at Kidd Creek is very infrequent, it may be totally eliminated by suitable process control.

E. Grinding Circuit Product Sizing

Variations in the final grind size reflect lost effectiveness in the grinding processes in two ways. First, the best size distributions for mineral separation are not being maintained. Secondly, if the product size is finer than necessary, grinding energy has been wasted.

The nature of size variations due to absence of precise control can be observed by comparing monthly composite product samples to those obtained over a short sampling period. The size-weight distributions of copper concentrate samples from 1985 composites (Scheding, 1985) and from survey no. 1, which were similar in overall size, are given in Table 4-22. This shows a narrower size distribution (less +53 um and less -7 um material) for the survey sample than for the composite sample. However, further comparisons are needed to confirm the significance of these differences. Micro-sieve analysis may be preferred over cyclosizing for this purpose.

Table 4-22. Kidd Creek 1985 Composite and Survey No. 1 Copper Concentrate Size Distribution Data

Screen size or Cyclosizer	1985 Composite Cut Size (µm)	1985 Composite Ind. %	Survey No. 1 Cut Size (µm)	Survey No. 1 Ind. %
200 mesh	75	1.6	75	0.3
270 mesh	53	3.1	53	2.0
400 mesh	-	-	38	13.2
Cone 1	41	8.1	36	3.5
Cone 2	25	14.3	22	13.7
Cone 3	18	16.8	16	14.9
Cone 4	11	13.6	10	13.2
Cone 5	8	8.3	7	9.0
- Cone 5	-8	34.2	-7	30.2
		100.0		100.0

Control System Requirements

The optimum economic circulating load of solids has been estimated at 350%, or 469 DMTPH based on a new circuit feed rate of 134 DMTPH. Note that for constant classifier feed conditions, the absolute circulating load value (rather than a ratio of the new feed rate) must be specified. The ultimate target grind size is yet to be defined, but this can be taken as roughly 126 µm (overall P80).

The needed control loops are as follows (refer to Figure 4-18).

Control Loop No. 1:

The short-term control of cyclone overflow product size (overall K80) is through the cyclone feed density. The density gauge on the cyclone feed could be used to provide the signal to control the flow of rate of water to the cyclone feed sump.

Control Loop No. 2:

The control of the circulating load is through the cyclone feed density and flow rate. The density may be controlled by loop no. 1. The cyclone slurry feed rate can be held constant by a constant sump level, which can be maintained by adjusting the new feed rate into the circuit. A sump level meter is needed on the cyclone feed sump. The signal from the level meter can be used to adjust the rod mill feed rate set point, and hence the rod mill feed rate.

Preliminary Economic Evaluation

Total equipment requirements for the above system would consist of a density gauge, a level meter, required control valve and actuator arrangements, plus two controllers and associated electronics, overrides, etc. Total cost, including installation and start up, was estimated to be approximately $40,000.00.

Once again, introduction of automatic control in grinding circuits has historically provided an average increase in operating efficiency in excess of 5% (Horst and Bender, 1979). For the primary ball milling circuit, a gain in operating efficiency can be expected from stabilization of the classification system. This will also stabilize the mill rheological conditions. With constant product size, variations in ore feed size and grindability (observed to be in the range of 10%) will be taken up by adjustments in the feed tonnage rate. This will mean that feed tonnage may be increased roughly 5 percent on average. To maintain the same average tonnage as prior to control, media levels and power draw can be reduced by the same amount.

Quantitatively, savings of 5% in primary ball milling steel and energy costs represent 2.25¢/mt, or approximately $30,000.00 per year for each line. This would also open the door to further savings through circuit optimization and through enhancement and extension of grinding process control throughout the plant.

4.5.6 Other Considerations and Future Work

4.5.6.1 Grinding Media Utilization

The original project plan called for ball mill grinding media utilization to be a major part of this study. In retrospect, this must follow classification and control system considerations as these will determine and stabilize the material size distribution of the mill contents on which media sizing must be based. The stabilization period for the media charge after a change is implemented is also very lengthy, in the order of 2 to 3 times longer than it takes the complete charge to be replaced by new media (Leclercq, 1987).

Improvements in grinding efficiency of 2 to 5% have been reported with implementation of graded ball charging at Climax Molybdenum (Dorfler et al, 1981). Laboratory scale tests may be useful for determining the effects of different media charging practices on mill breakage efficiency (Lo and Herbst, 1986). Along with material selection for improving media cost effectiveness, grinding media optimization offers important potential for cost reductions.

4.5.6.2 Ball Mill Speed

A previous general investigation showed that maximum impact conditions occur near 75 percent of critical speed in ball mills, but could not provide any clear correlation between mill speed and grinding efficiency (McIvor, 1983). Although higher speeds in ball mills should have the same potential for lower media consumption as in rod mills, this is not at all clear, perhaps because of complex internal classification effects (Hardinge and Ferguson, 1950; Lewis and Goodman, 1957).

4.5.6.3 Interfacing of Rod and Ball Milling Stages

The rod mill product size (ball mill circuit feed size) can be varied within certain limitations set by the total power draw needed to deliver the final grind size, and the possible variations in the power draw of each mill through charge level and speed adjustments. Having observed lower operating work indices for ball versus rod milling means that overall grinding efficiency likely could be increased by shifting more of the work to the ball mill by coarsening the rod mill product size. Technical models, from one as simple as the work index approach to those as complex as population balance modelling, may lend themselves to this analysis. However, the net effect on total grinding costs will once again define the optimum.

4.5.6.4 Ball Mill Circuit Water Usage

Work spearheaded by the Dow Chemical Company has provided a direct method for calculation of the operating density which provides the maximum production rate of fines, for a given size distribution of the mill contents, in batch grinding mills (Klimpel, 1984). This work suggests that mill density should be a maximum, just before the slurry viscosity rises, for best grinding. However, fines removal considerations suggest the advantage of high water flow rates. As well, the generally poorly understood relation between mill feed (or discharge) density and the actual density inside the mill means that further investigation is needed. In plants, further testwork to derive the empirical relationship between mill head water addition and circuit efficiency would appear to hold the most promise.

The important effect of water addition rates on circuit efficiency is apparent from the circuit surveys. Despite the large influx of fines into the circuit from the crushing plant at Selbaie, classification efficiency was higher than without the crushing plant running, probably owing to the large additional amount of water supplied. The same held true for the Kidd Creek secondary ball mill when fines and water were added from the copper regrind circuit. Another important consideration is that improved grinding efficiency must be balanced against reduced flotation residence time when more water is added to the grinding circuit.

4.5.6.5 Secondary Milling and Flowsheet Configuration at Kidd Creek

In the present circuit configuration, the secondary ball mill circuit appears to offer good potential for improvements through implementing automatic control and through classification system adjustments. As well, recirculation of copper regrind circuit product through the secondary circuit may be reconsidered. However, size by size assay analyses of concentrator products and subsequent plant testwork carried out by the mill staff have shown that bypassing secondary ball milling has extremely interesting potential for improving copper metallurgy and reducing reagent consumption (Scheding, 1985; McLellan, 1986a and b). This would pre-empt the need to consider any modifications within the present circuit configuration.

4.5.6.6 Sulphur Mineral Behavior

Much of the data collected to date could be used as a starting basis for modelling the behavior of sulphur bearing minerals in the grinding circuits. Since their size characteristics will strongly influence where they report in the concentrator products, grinding is a logical starting point for researching the environmental effects of various processing plant practices.

4.5.7 Summary and Conclusions

4.5.7.1 Les Mines Selbaie

1. Overall grinding performance of the ball mill circuit is in line with that expected from the Bond work index scale-up approach.

2. When the crushing plant is running, there is a large influx of fines and water into the ball mill circuit. These appear to offset one another in terms of classification system efficiency.

3. A preliminary evaluation showed that overall recovery increases with finer overall product grind size (P80) down to about 80 um. When grinding costs are considered, the net value per tonne processed is maximized at a grind size of approximately 90 um. With fixed mining tonnage, this should be the target grind size for the milling operation.

4. During cyclone surging, the amount of fines being recirculated to the ball mill indicates a serious loss in both classification system and overall grinding circuit efficiency. This, along with ore grindability variations, the intermittent influx of crushing plant fines and water, the occurence of ball mill overloading, the need to control the circuit product sizing for good flotation performance, and the need for circuit stabilization to permit design

optimization all support the need for an automatic control system. The expected payback period for the proposed system costing roughly $150,000.00 would be about six months.

Other important considerations for this circuit that have not yet been addressed include grinding media and water utilization.

4.5.7.2 Kidd Creek Mines

1. Overall grinding performance of the primary ball milling circuit is in line with that expected from the Bond work index approach. A design circulating load ratio of close to 350% can be recommended based on a balance of grinding and classification costs. This can be achieved by an increase in the size of the cyclone vortex finders.

2. The secondary ball milling circuit has a low circulating load ratio, which is attributable to the low reduction ratio of the circuit. Additional water from the copper regrind circuit has a positive effect on the secondary circuit classification efficiency. However, this is largely offset by the associated influx of fines. During survey no. 2, the rheological characteristics of the secondary ball mill slurry was identified as the likely cause for a decrease in the grinding rate.

3. Ore variability, mill power draw variations, classification system performance, the need to control circuit product sizing, and the need to provide circuit stability to permit design optimization all strongly support the need for automatic process control. The proposed system for the primary ball mill circuit could provide a payback period of slightly over a year, subject to further detailed evaluation.

4. The most important other considerations for the ball milling circuits are the flowsheet configuration, grinding media, and water utilization. These must be accompanied by development of the process economic model. This economic model may be rather complex because of the complexity of the flowsheet, but it is needed to give commercial meaning to the technical analyses.

4.6 DISCUSSION

This chapter has attempted to present a general approach for the evaluation and improvement of closed circuit ball milling operations for the process engineer in a froth flotation concentrator. A modelling system which links specific design and operating variables to overall grinding circuit performance through separate overall classification and breakage parameters has been presented. To place grinding in the context of the process plant, it has also been proposed that its effect on the performance of the flotation operations can be assessed, at least to a reasonable first degree, by considering the mineral distributions by size in the grinding circuit product relative to the size by size recovery performance of the flotation plant on each mineral. Grinding circuit process control methodology is then proposed based on the performance objectives identified by both the grinding circuit modelling system and the selected approach to interfacing of the grinding and flotation operations.

The key new concepts being presented here are the ball mill circuit functional performance equation;

$$\text{Mill Power Draw} \times \text{Circuit Classification Efficiency} \times \text{Power Specific Grinding Rate} = \text{Circuit Fines Production Rate}$$

and the grinding rate ratio;

$$\frac{\text{Plant Specific Grinding Rate}}{\text{Test Specific Grinding Rate}}$$

The data needed to measure or calculate each of the factors in these equations are readily available from plant sampling and laboratory analysis. These factors can also be seen to have specific individual as well as collective meaning relative to circuit performance. Ultimately, within economic constraints, the objective of circuit modifications would always be to maximize the product of classification efficiency and the power specific grinding rate, whether to increase production (higher tonnage or a finer grind) or to reduce costs by reducing mill power draw for the same tonnage and grind.

Since the overall classification and breakage efficiencies can also be associated with specific design and operating variables, circuit optimization objectives can be more concisely defined, implemented, and then tested, whether in the plant or on a computer simulator. When the net effect of a design change on circuit performance is understood in terms of these two parameters, means of improving overall circuit performance may be more clearly perceived. For example, reducing ball mill feed water addition rate could possibly result in improved breakage but an offsetting loss in classification system efficiency. By adding water elsewhere in the circuit, classification efficiency may be restored, and the benefit from the improved breakage condition thus realized.

It is also recognized that neither the utility nor the limitations of these classification and breakage parameters are fully defined by the limited amount of data and analysis presented here. Calculation of the mill fines inventory could be more rigourous. Secondary effects, such as changes in breakage product size characteristics with different media sizes, and the effect this has on the fines inventory, are unexplored. Klimpel (1982-83) has shown the need for some fine material to achieve the ideal rheological slurry characteristic for grinding, which would limit the maximum desirable system classification efficiency as it has been defined here. The exact nature of the laboratory testwork which may be best for determining the test specific grinding rate (for example, batch testing of ball mill feed versus locked-cycle testing of circuit feed) has not been determined. Most importantly, the range of operating conditions considered in this study was quite limited. This applies to the observations made relative to grinding-flotation interfacing and process control, as well as the proposed grinding circuit modelling approach. Note that in both of the case studies the feed tonnage rates to the mineral processing plants were substantially fixed over the duration of these evaluations by the underground mining operations. Grinding circuit feed and product sizes were essentially constant. Also no major chemical effects, such as might be expected by a change in grinding media material, were dealt with. Extensive further development work will be required to establish how well the new concepts introducted in this thesis may contribute to a general systematic approach to industrial ball mill circuit performance optimization over the range of normally encountered operating conditions.

CHAPTER 5

OVERALL SUMMARY AND CONCLUSIONS

5.1 STUDY OBJECTIVES

One approach to the overall task of achieving maximum effectiveness from plant grinding operations can be conceived through the following three steps:

1. the overall efficiency of the size reduction process is characterized and measured;
2. individual circuit design and operating variables are related to the overall efficiency of grinding through suitable process modelling; and,
3. the size reduction process models are integrated into the general system of plant operations and overall process performance.

An essential aspect of the grinding process modelling approach is that it must be related to process economics so that optimum characteristics can be identified.

5.1.1 Error Analysis of Bond Work Index Efficiency

The Bond methodology offers a useful overall measure of rod mill or ball mill circuit efficiency for the plant process engineer. Because it is directly related to the major grinding costs, those for power and steel consumption, it also provides the link to grinding economics. It is therefore considered that Bond work index analysis still offers a practical means of testing for ways to improve grinding performance. However, the accuracy of work index efficiency determinations has not been well defined. Until the error in the measurements is known, it means that true changes in circuit efficiency cannot be clearly distinguished from variations due to noise in the environment. One of the objectives of the study therefore was to quantify the error of relative work index efficiency and to seek means to minimize it.

5.1.2 Ball Mill Circuit Steady State Modelling

As a single lumped parameter the Bond work index provides little insight into the complex system of interacting variables which determine overall circuit efficiency. Population balance modelling techniques for closed circuit ball milling provide a detailed system of ore and process characterization along with computational sophistication to deal with the size reduction process. The computerized grinding circuit steady state simulator could be used to carry out off-line experimentation in a completely controlled and noiseless environment. It was perceived, however, that the relationships between the system of technical parameters being used and overall grinding efficiency and economy were not well established. The initial approach was to try to combine Bond work index analysis with population balance models in an attempt to evaluate ball milling circuits. Unfortunately, it was found that at the present level of their development the accuracy of the process unit models and their interacting relationships is not adequate to make reliable predictions about the performance of a particular circuit following a major change in a design or operating variable. It was recognized, however, that particle classification and breakage could be considered as two separate functions in closed circuit ball milling. Development of means to relate overall circuit performance to individual circuit design and operating variables through separate overall classification and breakage parameters subsequently became the primary objective of this investigation.

5.1.3 Defining Grinding Circuit Product Value

Economic analysis of grinding alone was recognized as being incomplete in the context of the mineral concentrator. Rather, it is necessary to assess the effects of grinding circuit operation on plant metallurgical performance, as well as any significant effects it may have on downstream processing costs. Another objective, therefore, was to determine a means to assign an economic value to the grinding circuit product.

5.2 ROD MILLING INVESTIGATIONS

From a review of the technical literature on rod milling, mill speed, fines addition to mill feed and feed water addition rate were selected for more detailed evaluation of their potential for improving the economic performance of rod milling. An investigation of the sources of error associated with laboratory test and operating work index determinations showed that relative work index efficiencies can be determined to an accuracy of plus or minus 3 to 4 percent (95% confidence interval) from short-term plant testing. Mill power draw versus charge level relationships were established in each operation to provide a basis for estimation of power and steel consumption costs with changes in mill charge level.

Review of rod mill power draw characteristics and steel consumption data from operating plants produced strong evidence of the potential for reducing steel consumption by operating at high mill speed, as opposed to high charge level, to achieve desired mill power draw. Estimated savings in steel consumption for an increase in mill speed from 70 to 77% of critical at Kidd Creek would provide approximately a one year payback on the cost of the required equipment modifications.

A combined program of testing the effect of the addition of cyclone feed material, as a source of fines, was carried out at the two operations. These tests established suitable plant testwork procedures, particulary a minimum time period of about one hour for the rod mill to stabilize after making an adjustment to the feed conditions. No improvement in grinding efficiency was indicated at either location. Preliminary results from variations in feed water addition rate, however, were positive. Subsequent water addition testing showed that rod milling efficiency could be improved at both plants by operating at reduced densities. Savings in power consumption of approximately 8 and 13 percent were established at Kidd Creek Mines and Les Mines Selbaie respectively. The effects on steel consumption are unknown.

Comparison of the testwork results from the two operations indicates that there may be inefficiency associated with oversize feed in the Kidd Creek rod mill. Optimization of rod mill feed size, along with optimization of mill feed percent solids, should be included in future testwork.

5.3 BALL MILLING INVESTIGATIONS

5.3.1 Ball Mill Circuit Modelling

Through a review of classification effects in wet ball milling circuits it was recognized that the production rate of new product size material was related to the size distributions inside the grinding circuit. This had been observed in the past both directly, through the ball mill feed sizing (Coghill and DeVaney, 1938; Kelsall et al, 1969), and indirectly, through the circulating load ratio (Davis, 1925) and invariant cummulative specific rates of breakage (Laplante et al, 1986). This lead to the proposition that overall ball mill circuit classification efficiency could be characterized by a single parameter, namely the percentage inventory of coarse solids (i.e., larger than the defined circuit product size) in the ball mill as estimated by the average in the mill feed and discharge size distributions.

Because this parameter for circuit classification efficiency does not depend on the ore type (in particular, its grindability), improving grinding circuit efficiency through classification equipment or operating adjustments can be considered on its own merit. Furthermore, having identified an overall circuit classification characteristic, functional performance analysis of ball milling, as practiced in value engineering design (Miles, 1972), lead to the development of the ball mill circuit functional performance equation:

$$\text{Mill Power Draw} \times \text{Classification System Efficiency} \times \text{Power Specific Grinding Rate} = \text{Production rate of new product size material.}$$

The specific grinding rate of coarse solids in the ball mill would depend on the grindability characteristic of the ore in conjunction with the efficiency of the breakage environment. A single measure of overall breakage efficiency can be deduced by factoring out the ore grindability as measured in a standardized laboratory test. When both plant and laboratory grinding rates of the coarse material have been established on a power specific basis, then the relative efficiency of the breakage environment can be represented by the ratio of the grinding rates, as follows:

$$\text{Grinding Rate Ratio} = \frac{\text{Plant Specific Grinding Rate}}{\text{Test Specific Grinding Rate}}$$

It is proposed that the effects of specific ball mill circuit design and operating variables can be related to overall circuit efficiency through these two separate overall parameters, one related to classification alone (the coarse inventory) and the second related to only breakage (the grinding rate ratio). Relationships with the major grinding costs can be established through mill power draw versus charge level relationships and relative work index efficiencies for different design or operating conditions, in a similar fashion to rod milling.

5.3.2 Interfacing of Grinding and Flotation

The effects of the grinding circuit on the metallurgical performance of flotation operations were reviewed in an effort to assign a monetary value to the grinding circuit product, and thus to define at least all the major elements in the economic model for grinding. Current difficulties with measurement and modelling of mineral liberation appeared to limit the practicality of the approach of relating grinding to liberation and in turn relating liberation to flotation. However, the reasonably consistent and readily measurable size by size behavior of minerals in the flotation circuit offered a possible alternative. Combined with the mineral distribution by size in the grinding circuit product stream, a first estimation of the recoverable values in the flotation feed can be determined.

5.3.3. Process Control of Ball Milling

The important role that process control plays in mineral processing plant performance deemed it a suitable topic for detailed review as well. Control system objectives were defined on the basis of the desired quality of the flotation circuit feed and maintenance of the selected grinding circuit operating conditions which yield efficient size reduction performance. A simple control system which may be applied to achieve a base level of automatic control for conventional rod-ball or single stage ball milling circuits was proposed.

5.3.4 The Case Study Circuit Evaluations

For the simpler of the two plant flowsheets at Les Mines Selbaie, size by size copper mineral recoveries in flotation were calculated from size distribution and size by size assay data of monthly composite samples of flotation feed, concentrate, and tailings. By similar sizing and assaying of the cyclone overflow sample from a grinding circuit survey, the copper mineral distribution in the flotation feed was calculated. Based on a typical smelter contract, the monetary value of recoverable copper in the grinding circuit product during that survey could be calculated.

A typical mineral distribution by size in the Selbaie flotation feed was also selected from available monthly composite sample data. By shifting this distribution across a range of K80 sizes, it was possible to develop an estimated overall grind size versus recovery curve. Based on the expected product size and unit grinding costs at different tonnages, it was possible to estimate daily plant revenues, profits, and loss of copper values to tailings at different throughput rates.

The two general surveys carried out at Kidd Creek included sampling of the feed and product streams of the flotation circuits. Sizing and size by size assaying of these samples was used to determine the distribution of copper and zinc minerals in the concentrator products (copper concentrate, zinc concentrate, and final tailings) for each size class. The distribution by size of copper and zinc minerals in the ground products was also determined for each survey by reconstitution of copper concentrate and tailings streams. Although too major a task for this study, it is suggested that this type of information may provide the basis to eventually integrate mineral grind size and flotation performance data into a general economic model of plant performance at Kidd Creek Mines.

Two general circuit surveys were carried out at each plant. Overall work index efficiencies, classification system efficiencies, and grinding rate ratios were determined for the Selbaie ball mill and Kidd Creek primary ball mill circuits. Although differences in operating performance were small, both pairs of surveys showed that the net change in overall circuit efficiency from one survey to the next, as indicated by relative work index efficiencies, reflected the combined effect of changes in classification system and breakage efficiencies.

Comparative grindability data on the secondary ball mill circuit feed at Kidd Creek for the two surveys were lacking. However, a larger difference in operating work indices than would likely occur due to a change in the ore was observed. The difference in circuit classification system efficiency was quite small, while a much more significant drop in the power specific grinding rate was seen to occur. This probable loss in breakage efficiency could be attributed to a change in the breakage conditions in the mill, perhaps related to slurry rheology.

Changes in circuit classification efficiency with different circulating load ratios were used to develop a case for implementing a design change to adjust the circulating load in the primary ball mill circuit at Kidd Creek. Although overall grinding efficiency and related costs for power and steel consumption would be negatively affected by lowering the circulating load ratio, these were shown to be more than offset by expected reductions in pumping and classification equipment operating and maintenance costs. A methodology for selection and achievement of the optimum economic circulating load based on grinding cost minimization was thus presented.

Initial automatic process control systems based on the previously described review of process control in conventional ball milling were recommended for the Selbaie grinding circuit and through the first stage of ball milling at Kidd Creek. A very preliminary economic justification for implementation of grinding process control was given for each operation.

Other important considerations that were identified for future work include grinding media and water utilization at both operations, as well as the flowsheet configuration and development of the process economic model at Kidd Creek.

CONTRIBUTIONS TO ORIGINAL KNOWLEDGE

1. From analysis of operating rod mill data, a case is presented for increasing rod mill speed in order to reduce steel consumption.

2. The error associated with relative Bond rod mill work index efficiency determinations from short-term plant tests has been estimated. Sampling and experimental error, including variations in laboratory rod mill work index with test sample feed sizing, have been quantified. Knowledge of this error was used in the demonstration of an increase in rod mill circuit efficiency as a result of increasing the feed water rate.

3. An overall classification system efficiency index for a closed circuit ball mill was presented, namely, the proportion of the mill solids contents coarser than the target product size of the circuit, and referred to as the "coarse solids inventory".

4. An overall breakage efficiency index for a closed circuit ball mill was presented, namely, the ratio of the specific grinding rate of the coarse solids in the plant versus the laboratory ball mill, and referred to as the "grinding rate ratio".

5. By combining the overall classification efficiency with the overall breakage rate parameter for the ball mill circuit, a functional performance equation for closed circuit ball milling was derived. It is proposed that this algorithm can provide an intermediate level of circuit performance characterization which can be used to help relate individual design and operating variables to overall circuit performance.

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APPENDIX A

GENERAL AND HISTORICAL INFORMATION ON LES MINES SELBAIE CIRCUIT DESIGN AND OPERATION

General and Historical Information on Les Mines Selbaie

Circuit Design and Operation

Equipment Description (June, 1985)

Rod Mill:

Dominion Engineering, 1957, serial no. 340-281, 2.44 m (8.0 ft) inside shell diameter, 3.66 m (12.0 ft.) effective working length (inside end liners at the shell), overflow discharge, drawing no. 61603.

Liners, Shell; Centre-wave design, Ni-hard, 133 mm (5.25 in.) by 57 mm (2.25 in.), 6mm (0.25 in.) Linatex backing.
Ends; smooth Ni-hard, 89 mm (3.5 in.).

Media; 90 mm (3.5 in.) diameter by 3.5 m (11.5 ft.) long, AISI 1090 steel rods. Switched to 76 mm (3.0 in.) rods in late 1986.

Drive, Motor; C.G.E., Toronto, serial no. 723582, 298 kw (400 HP), 1.15 service factor, 4000 volts, 1185 r.p.m., 51 amps.
Fluid coupling; Vulcan-Sinclair SCR 5, size 26.
Gear reducer; Dominion Engineering, ratio 5.545 to 1.
Pinion, 29 teeth.
Ring gear, 320 teeth.
Mill speed is 18.8 r.p.m. (approx.), or 67.2% of critical based on 2.29 m (7.5 ft.) inside diameter.

Ball Mill:

Dominion Engineering, 1957, serial no. 340-280, 3.20 m (10.5 ft.) inside shell diameter, 3.96 m (13.0 ft.) effective working length (inside end liners at the shell), overflow discharge, drawing no. 61584.

Liners, Shell; Skega rubber, 50 mm (2.0 in.) shell plates, 135 mm (5.3 in.) lifters.
Ends; Ni-hard, 89 mm (3.5 in.) thick with 25 mm (1.0 in.) lifters.

Media; 38 mm (1.5 in.) nominal diameter Marmet grinding slugs. Switched to 38 mm (1.5 in.) diameter balls in late 1986.

Drive Motor; English Electric, St. Catherines, serial no. 244466, 597 kw (800 HP), 1.0 service factor, 4000 volts, 885 r.p.m., 101 amps.

Gear reducer; Dominion Engineering
Ring gear and pinion; Dominion Engineering
Mill speed is 18.2 r.p.m. (approx.), or 75.1% of critical based on 3.28 m (10.0 ft.) inside diameter.

Cyclone Feed Pumps:

Operating (variable-speed) Unit

Pump; Allen-Sherman-Hoff, 203 mm (8 in.) inlet and outlet, 533 mm (21 in.) diameter impellor.

Fluid coupling; American Standard Gyrol, Type VS, size 146, serial no. 533-214-123, 56 max. kw (75 HP), 1800 max. r.p.m.

Sheaves; 229 mm (9 in.) on fluid coupling, 660 mm (26 in.) approx. or pump.

Motor; Nameplate illegible.

Standby (fixed-speed) Unit

Pump; Worthington model 8M-193, serial no. B08010591, 203 mm (8 in.) inlet, 254 mm (10 in.) outlet, 489 mm (19.25 in.) diameter impellor.

Sheaves; 152 mm (6 in.) approx. on motor, 660 mm (26 in.) approx. on pump.

Motor; C.G.E., Toronto, 75 kw (100 HP), 95 amps, 550 volts, 3555 r.p.m., 1.15 service factor, model 8F1386XX, serial no. 721815.

Hydrocyclones (two operating plus one standby):

Krebs model D20B, extended body, 127 mm (5 in.) Ni-hard vortex finders, 89 mm (3.5 in.) ceramic apexes, in-line mounting arangement with manual isolation valves, pressure gauges.

Instrumentation and Control

Details of the B grinding circuit process control equipment, instrumentation and hardware as of October, 1985, are shown on the enclosed drawing number 11112-02. The

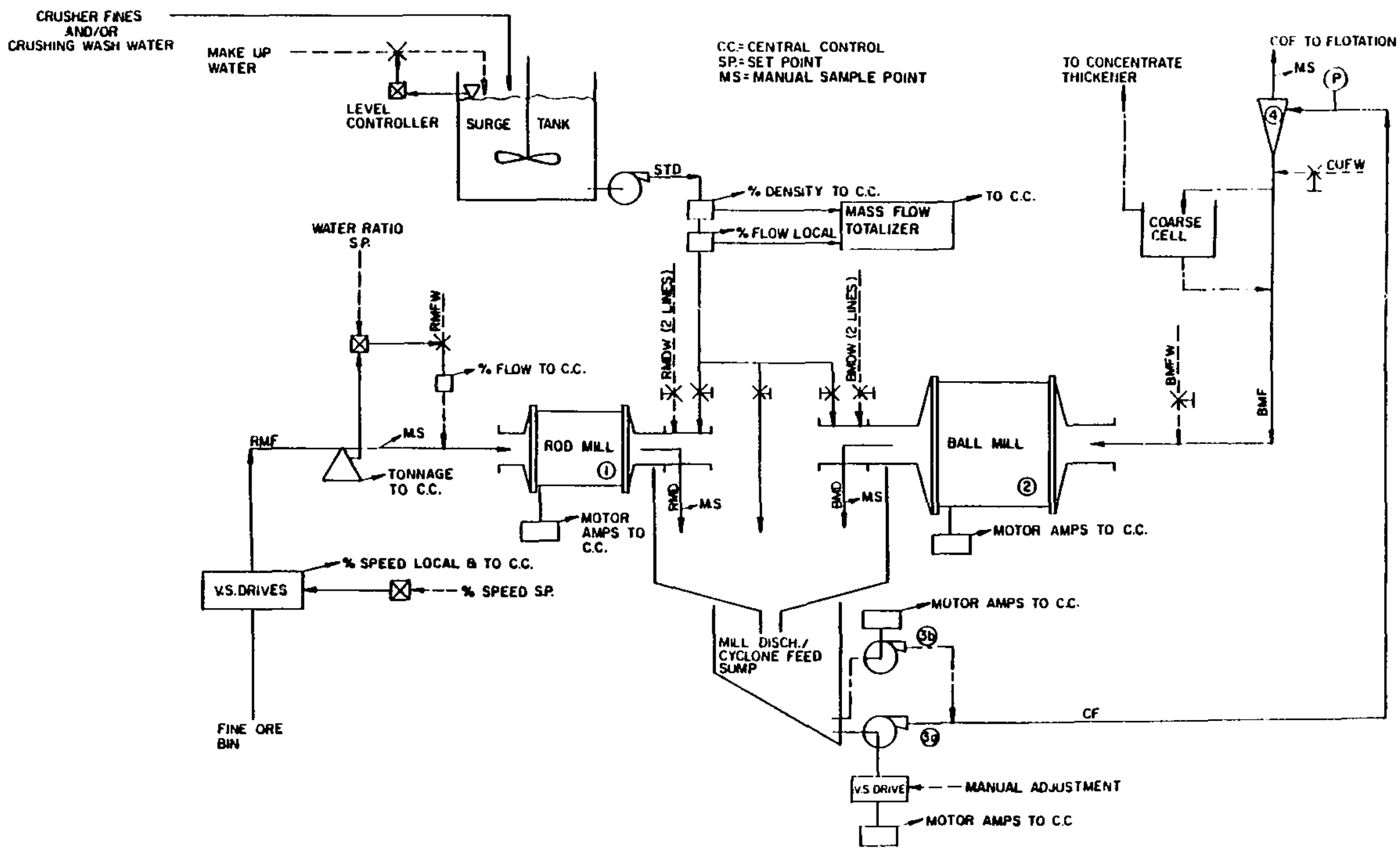

Figure A-1

DATE		REVISION	BY
11/10/85	0	NEW DRAWING	JK

LES MINES SELBAIE

GRINDING CIRCUIT- INSTRUMENTATION AND CONTROL

DATE: OCT 11/1985 SCALE: NONE

DWG NO. 11112-02 REV 0

following is a summary description of the grinding circuit instrumentation and control practices in the plant at that time. Subsequent modifications are discussed in the text of the report.

Rod Mill Feed (RMF):

Rod mill feed tonnage (solids plus 4-6% moisture) is normally manually set by the operator in the central control room through a speed controller/indicator on the 2 main feeder belts from the fine ore bin (North and South). The indicator is calibrated so the "% of scale" is approximately equal to the total rod mill feed tonnage (solids plus moisture).

Rod mill feed tonnage is measured on a Miltronics weightometer on the feed conveyor just before the rod mill (better than ± 1% accuracy). The tonnage is indicated on a digital read out and accumulator, as well as on a strip-chart indicator/ recorder in central control.

On automatic, the speed of the feeder conveyors is continually adjusted by the weightometer output to match the set point, but overcompensation of the feed rate leads to wide fluctuations, and so it is not normally operated in this mode.

Rod Mill Feed Water (RMFW):

Rod mill feed water is normally set by an automatic proportional controller which takes signals from the weightometer and a flowmeter on the water line, and adjusts the water flow through an automatic valve to meet the water ratio set point. Only occasional adjustments are needed to maintain the rod mill discharge density near 81% solids by weight.

Surge Tank Discharge (STD):
This stream is normally identified as "spiral water", as the solids in it come from the spiral classifier overflow sump, from pre-washing the crushing plant feed, as well as crushing plant conveyor spray and washdown water (continuous). As well, a level controller controls make-up water into the tank through an automatic valve to keep a high tank level.

The flow rate of this stream is controlled by manually adjusting the valve opening. This stream is normally sent to the rod mill discharge housing (to dilute the dense slurry from the rod mill discharge), but can be diverted to the ball mill, or a discharge pipe between the mills for better sampling access.

A Bestobel Meterflow Ltd. ultrasonic flowmeter and a Texas Nuclear density gauge are provided on this line. The flowmeter scale is calibrated so that 0-3 metres/second corresponds to 0-100% of scale, and indicates locally. The density scale is calibrated so that 1.0 - 1.5 slurry specific gravity corresponds to 0-100% of scale, and is displayed in the control room in the same box that gives a digital accumulation of the solids tonnage (mathematically determined from the density, the flow rate, the solids specific gravity of 3.0, and the inside pipe diameter of 137mm). Depending on the time since calibration, the instruments are accurate from 1 to 5%. However, the method of calibration (involving timed filling of a tank and sampling for slurry percent solids) is reported as no more than ± 5% accurate.

This stream is a major source of water into the circuit (note that a substantial flow from the spiral classifier sump was noted even with the crushing plant not operating), and is adjusted (manually) by the operators when cyclone overflow density varies outside of the range of 43-45% solids.

Density and flow rate are also recorded continuously on a strip chart.

Manually adjusted water addition points into the circuit include the following.

Rod Mill Discharge Water (RMDW)

Ball Mill Discharge Water (BMDW)

Ball Mill Feed Water (BMFW)

Cyclone Underflow Water (CUFW)

Cyclone Feed Pump Speed:

One cyclone feed pump is provided with a variable speed fluid coupling, so the speed of the pump can be adjusted manually at the pump location.

Cyclone Feed Pressure:

Feed pressure gauges are provided at each cyclone inlet.

Electrical:

Ammeters are provided on the control room for both mill motors and the cyclone feed pump motors.

Process Assays:

Also of interest to the operation of the grinding circuit is the Outokumpu Minexan on stream analysis system, which provides continuous copper assays on flotation feed, concentrate, and tailings. Of particular interest is the tailings assay, which, if too high, indicates a need to cut back on the total grinding tonnage rate.

Manual Sampling for Percent Solids:

Manual samples for slurry density are cut several times per shift at the following locations:

- Rod mill feed
- Rod mill discharge
- Ball mill discharge
- Cyclone overflow

Flotation Feed Size Analysis:

Size analysis (percent passing 75 um) of flotation feed is reported, usually daily, approximately 48 hours following the time the samples are cut.

Grindability Testwork

Prior to production, Bond work index determinations were carried out by Allis-Chalmers in 1979, and were reported as follows. The laboratory work index values are in kwh/mt, and were performed at 1180 um (14 mesh) for rod milling, and at 150 um (100 mesh) for ball milling (1).

Table A-1. Allis-Chalmers Work Index Test Results (kwh/mt)

Sample	Rod Mill W.I. (14 mesh)	Ball Mill W.I. (150 mesh)
C-210	14.6	9.9
C-211	13.3	11.0

A number of grindability tests were also carried out on site at the design stage and after plant start-up using a procedure developed by Pickands and Mather which closely follows the standard Bond procedure (2). The results are summarized below. The design stage tests are the first six listed (3), and the last two were performed in 1984.

Table A-2. Les Mines Selbaie/Picklands and Mather Test Result

Sample	Control Size	Ball Mill W.I.
C-201	150 um	14.4
C-202	150 um	12.3
C-203	125 um	9.1
C-204	125 um	11.7
Shaft grab	90 um	14.3
B ore & waste	125 um	13.0
South zone (C-228)	150 um	14.4
Rod mill feed (Sept. 9/84)	150 um	15.6

Several comparative grindability tests were also carried out in 1984 using the method of Berry and Bruce (4) with a small laboratory mill loaded with grinding rods. The results of these tests are summarized below. Note that although a rod charge is utilized, the feed and product size range is for Bond ball mill work index estimation, as are the comparative grindability tests and test samples.

Table A-3. Comparative Grindability Test Results

Sample	Ball Mill W.I. (approx.)
Main zone (C-226)	15.8
South zone (C-227)	16.1
Rod mill feed (June 4, 1984)	14.2
South zone (C-228)	15.4
Rod mill feed (Sept. 9, 1984)	13.7

The control size for the reference test ores was 150 um (100 mesh) in each case. The Bond ball mill grindability tests on the reference ores were performed at Lakefield Research. The Berry and Bruce method uses a single batch grind so no control size can be quoted, although the ultimate product sizes should be similar. Note the discrepancy between the Les Mines Selbaie/Picklands and Mather test results and the comparative grindabilities for the last two samples. The latter should be taken as approximate, being the less accurate of the two methods.

Rod Mill Feed Size

The range of sampled and reported rod mill feed size data (with the exception of tests for which the crusher screening size was intentionally changed) is summarized below.

Table A-4. Rod Mill Feed Size Data

Date	Rod Mill F80 (mm)
October 5, 1983	16.0
May 15, 1984	14.0
May 23, 1984	13.0
June 6, 1984	13.4
September 6, 1984	14.5
April 3, 1985	13.0
April 3, 1985	13.5

Barring times at which the fine ore bin is drawn very low, it appears that the rod mill receives a consistent top size of feed material, as provided by the closed circuit crushing plant.

Crusher Fines

For the single set of data obtained in April 1985 for which the crusher fines size distribution was available, the solids material was over 80% passing 75 um. With the crushing plant running, this stream contains from about 9 to as high as 20 DMTPH of solids. It is also responsible for a substantial flow of water into the circuit. For one data obtained in April, 1985, this stream (9.47 DMTPH) represented a quantity of fines equal to 16% of the rod mill solids feed rate, and 65% of the total water into the circuit. It appears that this stream could have a major effect on the performance of the ball mill, and the size of material observed in the final cyclone overflow.

A series of samples taken from July 24 to 29, 1985, revealed that the size distribution of the crusher fines varies widely, as shown below.

Table A-5. Crusher Fines K80 (um)

Date	Night	Day	Afternoon
July 24, 1985		100	255
25		85	255
26	137	163	187
27	148	150	165
28	148	127	137
29	182		170

Assay data also showed that these solids are significantly richer in sulphide minerals than the rod mill feed, averaging approximately 5% copper compared to 3% in the plant feed.

Circuit Product Characteristics

The range of reported overflow size distributions from various samples is summarized as follows.

Table A-6. Cyclone Overflow Size Data

Date	Cyclone Overflow P80 (um)
April 3, 1983	82
October 5, 1983	96
May 15, 1984	120
May 23, 1984	120
June 6, 1984	95
July 25, 1984	96
September 6, 1984	107
January 10, 1985	95
March 12, 1985	111
April 3, 1985	103
April 3, 1985	90
April 30, 1985	128

The circuit displays a considerable variation in final product size, at least part of which may be related to the intermittant crushing plant operation.

Cyclone overflow density is targeted at 43 to 45% solids by weight, but normally varies in the range of 38 to 45%. The target product size is approximately 70% minus 75 um (200 mesh), or an 80% passing size (P80) of approximately 100 um. Production capacity was originally designed at 1500 metric tons per day, but has subsequently been increased to average 1650 tons daily of combined rod mill feed and crusher fines.

Circuit Sampling Data

Rod mill discharge size distributions showed a considerable variation, from an 80 percent passing size of 900 to 1300 um covering all reported data, with one extreme at 1550 um.

Complete circuit size distributions were determined from three preliminary sampling surveys carried out in April, 1985, as well as two earlier sampling surveys (June 6, 1984, and October 5, 1983). The 80 percent passing sizes are summarized in Table A-7. The details of reported operating data are given in Table A-8. Note that the data from test no. 3, obtained April 30, 1985, was obtained under two unusual operating conditions, specifically, a high rod mill tonnage rate, and fine rod mill feed size.

Table A-7. Summary of Sample Operating Data

Test	TONNAGES (DMTPH) Rod Mill Feed	TONNAGES (DMTPH) Crusher Fines	80% PASSING SIZES (um) Rod Mill Feed	80% PASSING SIZES (um) Rod Mill Disch.	80% PASSING SIZES (um) Cyclone O.F.	80% PASSING SIZES (um) Crusher Fines
1, April 3/85	56.9	-	13,000	920	103	-
2, April 3/85	57.9	9.5	13,350	970	90	70
3, April 30/85	71.4	-	12,000	1290	128	-
June 6/84	55.3	-	13,400	1040	95	-
October 5/83	57.3	11.2	16,000	1040	96	N/Avail.

Table A-8. Sampled Circuit Size Distributions - Cumulative % Passing

Test #1 - April 3/85 (a.m.)

Rod Mill Feed = 82.0% - 13,500 um (4.3% moisture), 56.90 DMTPH

Screen Size (um)	Rod Mill Disch. (S.G.=2.83)	Ball Mill Feed (CUF)	Ball Mill Discharge	Cyclone Feed	Cyclone O.F.	Crusher Fines -NIL-
2,800	99.0	98.9	100			
2,000	97.2	97.4	98.7	100		
1,000	82.2	88.2	95.5	92.0		
500	61.3	74.5	88.4	81.1		
355	52.4	65.8	82.1	74.0	100	
250	45.1	55.3	72.7	65.0	98.5	
150	37.1	39.6	56.4	49.6	91.2	
125	34.1	33.9	49.0	45.6	85.2	
90	30.7	28.1	41.3	37.6	76.9	
75	28.5	24.8	36.7	33.4	71.1	
45	23.4	18.8	27.6	27.8	56.7	
% Solids by weight	84%	74%	72%	(poor sample)	45%	--

Table A-8 (Cont'd) Sampled Circuit Size Distributions - Cumulative % Passing

Test #2 - April 3/85 (p.m.)

Rod Mill Feed = 80.6% - 13,500 um (4.2% moisture), 57.86 DMTPH

Screen Size (um)	Rod Mill Disch.	Ball Mill Feed (CUF)	Ball Mill Discharge	Cyclone Feed	Cyclone O.F.	Crusher Fines 9.47 DMTPH
2,800	99.4	98.4	99.4	99.7		
2,000	97.8	97.1	98.7	98.6		
1,000	81.2	88.1	94.9	91.7		
500	59.6	75.6	87.5	80.7		
355	50.8	67.4	81.2	73.4	100	
250	43.7	57.1	72.0	64.1	98.5	100 (est.)
150	36.2	40.3	55.2	48.4	92.6	96 (est.)
125	33.3	33.9	48.0	41.7	87.4	92.6
90	30.0	27.3	40.1	34.2	79.9	85.8
75	28.0	23.6	35.6	29.8	74.6	81.5
45	23.1	17.3	26.5	20.9	62.0	64.9
25						53.3
% Solids by weight	84%	74%	73%		43%	14%

Sampled Circuit Size Distributions - Cumulative % Passing

Test #3 - April 30/85

Rod Mill Feed = 86.7% - 13,500 um (4.4% moisture), 71.38 DMTPH

Screen Size (um)	Rod Mill Disch. (S.G.=2.80)	Ball Mill Feed (CUF)	Ball Mill Discharge	Cyclone Feed	Cyclone O.F.	Crusher Fines -NIL-
2,000	93.6	95.9	95.2	96.0		
1,000	73.8	85.1	86.7	86.6		
710	62.8	77.5	81.4	80.3		
500	53.5	68.0	74.8	72.8		
355	45.7	56.1	65.6	63.3	100	
250	39.5	42.0	53.7	51.2	96.5	
150	33.1	27.1	38.9	38.9	85.2	
125	30.9	22.9	34.2	34.7	79.2	
90	27.7	17.6	28.0	29.0	69.9	
75	26.1	15.7	25.3	26.5	65.1	
45	21.4	11.1	18.9	20.3	51.0	
% Solids by weight	79%	74%	74%		42%	

Table A-8 (Cont'd) Sampled Circuit Size Distributions - Cumulative % Passing

June 6, 1984

Rod Mill Feed = 80.3% - 13,500 um (3.8% moisture), 53.33 DMTPH

Screen Size (um)	Rod Mill Disch.	Ball Mill Feed (CUF)	Ball Mill Discharge	Cyclone Feed	Cyclone O.F.	Crusher Fines
2,800						
2,000	95.6	89.8	94.1			
1,000	78.7	84.2	90.4			
710	66.8	77.6	85.7			
500	56.8	69.3	78.9			
355	48.2	58.0	68.5			
250	40.9	36.4	46.5		98.6	
150	27.2	30.2			92.2	
125					87.9	
90					78.6	
75					72.2	
45					57.5	
% Solids by weight	82%	79%	71%		39%	

Sampled Circuit Size Distributions - Cumulative % Passing

October 5, 1983

Rod Mill Feed = 79.8% - 16,000 um, 57.33 DMTPH

Screen Size (um)	Rod Mill Disch.	Ball Mill Feed (CUF)	Ball Mill Discharge	Cyclone Feed	Cyclone O.F.	Crusher Fines 11.20 DMTPH
2,800						
2,000	96.7	98.7	99.5	99.2		
1,000	79.1	91.9	97.4	95.2		
710	68.6	86.9	95.3	91.9		
500	59.5	81.3	92.3	87.8		
355	52.0	74.3	87.4	82.4		
250	46.0	65.4	79.8	75.0	98.5	Not Avail.
150	38.7	48.4	62.8	59.6	91.8	
125	36.0	42.2	55.9	53.4	86.4	
90	32.4	34.3	46.5	45.0	78.0	
75	30.0	28.4	40.7	39.7	72.0	
45	27.7	25.6	36.2	35.7	66.6	
% Solids by weight	81%		71%		44%	

Operating Work Indexes

Operating work index calculations (5) were carried out for the data sets mentioned in Table B1-7 with the exception of the test run of October 5, 1983, for which mill power draw measurements were not taken. Details of these calculations are given below. Note for test no. 2, the ball mill circuit P80 was assumed to be 93 microns, and crusher fines were not included in the circuit feed rate.

Table A-9. Operating Work Indexes (Preliminary)

Rod Milling (Uncorrected)

Test No.	Power Draw[1] (kw)	Feed Rate (DMTPH)	F80 (um)	P80 (um)	Op.W.I.[2] (kwh/mt)
1	206.1	56.9	13,000	920	15.0
2	207.9	57.9	13,350	970	15.3
3	210.6	71.4	12,000	1,290	15.8
June 6/84	215.6	55.3	13,400	1,040	17.4
		Ave:	12,940	1,055	15.9

1. Based on reported amperage readings, and assuming 90% drive efficiency to pinion shaft.

2. Based on

$$\text{Op.W.I.} = W \Big/ \left(\frac{10}{\sqrt{P80}} - \frac{10}{\sqrt{F80}} \right)$$

where

W = kilowatt hours/metric ton

Table A-9 (Cont'd.) Operating Work Indexes (Preliminary)

Corrections for Mill Operating Parameters

Test No.	Mill Diameter[1] (EF_3)	Reduction Ratio[2] (EF_6)	Corrected[3] Bond W.I.
1	1.013	1.016	14.5
2	1.013	1.024	14.8
3	1.013	1.27	12.3
June 6/84	1.013	1.052	16.3

1. Based on $EF_3 = \left(\dfrac{8}{I.D.}\right)^{0.2}$

2. Based on $EF = 1 + \dfrac{(Rr - Rro)^2}{150}$

   Where $Rro = \dfrac{8 + 5\,Lr}{I.D.}$

   $= 15.7$ (optimum reduction ratio)

3. Based on

$$\text{Bond W.I.} = \frac{\text{Op. W.I.}}{EF_3 \times EF_6}$$

Table A-9 (Cont'd.) Operating Work Indexes (Preliminary)

Ball Milling (Uncorrected)

Test No.	Power Draw (kw)	Feed Rate (DMTPH)	F80 (u)	P80 (u)	Op.W.I. (kwh/mt)
1	540	56.9	920	103	14.5
2	538	57.9	970	93	13.0
3	518	71.4	1,290	128	12.0
June 6/84	481	55.3	1,040	95	12.1
		Ave:	1,055	105	12.9

1. Calculation based on rod milling discharge tonnage and P80 adjusted for crusher fines.

Corrections for Mill Operating Parameters

Test No.	Mill Diameter (EF_3)	Corrected Bond W.I.
1	.9563	15.1
2	.9563	13.6
3	.9563	12.5
June 6/84	.9563	12.7

Combined Rod Mill - Ball Mill Work Index
(Average from 4 tests)

Rod Mill

F80 = 12,940
P80 = 1,055
Op.W.I. = 15.9

W = 15.9 (.3079 - 1.0879)
= 3.498 kwh/mt

Ball Mill

F80 = 1,055
P80 = 105
Op.W.I. = 12.9

W = 12.9 (.9759 - .3079)
= 8.617 kwh/mt

W total = 12.115 kwh/mt

Overall Op.W.I. = 12.115/(.9759 -.0879) = 13.64 kwh/mt

The results are summarized below.

Table A-10. Summary of Operating Work Index Data

Test	Operating Work Indexes (Uncorrected), kwh/mt	
	Rod Mill	Ball Mill
1, April 3, 1985	15.0	14.5
2, April 3, 1985	15.3	13.0
3, April 30, 1985	15.8	12.0
June 6, 1984	17.4	12.1
	Average: 15.9	Average: 12.9

The combined rod mill - ball mill work index based on average F80 and P80 sizes for the 4 tests is 13.64 kwh/mt.

When the above values were corrected for Bond's efficiency factors related to the grinding mill operating parameters, the corrected bond operating work indexes were obtained, as summarized below.

Table A-11. Summary of Corrected Operating Work Index Data

Test	Bond Operating Work Indexes (Corrected), kwh/mt	
	Rod Mill	Ball Mill
1, April 3, 1985	14.5	15.1
2, April 3, 1985	14.8	13.6
3, April 30, 1985	12.3	12.5
June 6, 1984	16.3	12.7
	Average: 14.5	Average: 13.5

Classifier Performance

Mass balances at the hydrocyclones were calculated for tests no. 1, 2, and 3 as were the actual recovery performances of the hydrocyclones on each size fraction. Details of sampling procedures were unknown. The overall mass split (circulating load) for each case was estimated using a simple mass balance technique (i.e., linear regression was applied to the values obtained in the numerator and denominator of the two-product formula estimate of recovery for each size fraction, to give the best overall estimate of circulating load from all the sampled classifier size distribution data). Recoveries to underflow of each size class were then calculated. Cyclone feed percent solids was also calculated from the overall mass split and cyclone product (overflow and underflow) densities.

Classifier recoveries to underflow are summarized below.

Table A-12. Hydrocyclone Recoveries (To Underflow)

	Test #1	Test #2	Test #3
Mesh Size (um)	Recovery	Recovery	Recovery
2,800	100%	100%	100%
2,000	100	100	100
1,000	100	100	100
500	100	100	100
355	100	100	100
250	95.9	98.2	93.4
150	87.7	95.8	82.2
125	75.8	90.8	71.0
90	69.8	87.6	66.6
75	65.2	84.8	58.1
45	57.9	80.0	53.3
-45	52.2	69.1	43.2

The classifier performance data for the three test runs are summarized as follows. Note that no actual d50 exists for tests 1 and 2, since over half of all solids report to underflow.

Table A-13. Classifier Performance Data

	Test no. 1	Test no. 2	Test no. 3
Cut size: Actual d50	-	-	40 um
Circulating load of solids (underflow/overflow)	430%	800%	350%
Bypass (water recovery to underflow)	55%	70%	47%
Percent solids (by weight)			
Feed	66%	68.5%	63.3%
Overflow	45%	43%	42%
Underflow	74%	74%	74%

Note that the hydrocyclones were observed to be surging during a later circuit reconnaisance. Feed pressure of the two operating units would run (in unison) at 90 kpa (13 psi) for about 10 seconds, and then drop to 41 kpa (6 psi) for 3 to 4 seconds. When the pressure dropped, the overflow virtually ceased as the cyclones appeared to flush their contents through the underflow.

Overall Plant Assays and Recoveries

Monthly composites on plant feed grade, fineness of grind, copper recovery, and copper concentrate grade for the months of September, 1984, through April, 1985, are summarized below, ranked by copper feed grade. Note that the coarse flotation cell was operated intermittently after January of 1985.

Table A-14. Grind Performance and Copper Assays and Recoveries

Month	Feed Grade % Cu	Total Feed Rate MTPH	Grind %-75um	Copper Recovery %	Concentrate Grade % Cu
Nov.84	3.60	70.0	68.3	95.7	29.38
Mar.85	3.60	68.1	64.8	96.0	27.75
Feb.85	3.28	69.3	64.9	95.8	28.75
Apr.85	3.12	66.4	60.8	95.3	27.88
Oct.84	3.11	73.0	69.9	96.1	26.22
Dec.84	2.86	74.6	72.8	95.5	25.93
Sept.84	2.81	68.6	70.6	95.6	26.02
Jan.85	2.33	72.7	70.3	94.7	29.19

Grinding Costs

The estimate of total grinding costs is based on the following figures:

a. Power Cost: 10.8¢/kwh, 1985 (generated on site).

b. Grinding rods: 61.2¢/kg, delivered to plant.

c. Grinding slugs: 64.8¢/kg, delivered to plant.

d. Average hourly tonnage: 70.3375, Sept. 1984 - April 1985.

e. Average power draw, Sept. 1984 - April 1985, motor input.

Rod Mill: 233.6 kw

Ball Mill: 581.0 kw

f. Average media consumption, Sept. 1984 - April 1985.

Rods: 0.28875 kg/mt

Slugs: 0.68750 kg/mt

g. Estimated "other" costs, including maintenance, non-grinding power, and overhead equal to 20% of total grinding costs.

Rod Milling:

Power: $\frac{233.6 \text{ kw}}{70.3375 \text{ tph}}$ x 10.8¢/kwh = 35.867¢/mt

Media: .28875 kg/t x 61.2¢/kg = 17.672¢/mt

Other (approx:) = 13.385¢/mt

Rod Milling, Total 66.924¢/mt

Ball Milling:

Power: $\frac{581.0 \text{ kw}}{70.3375 \text{ tph}}$ x 10.8¢/kwh = 89.210¢/mt

Media: .68750 kg/t x 64.8¢/kg = 44.550¢/mt

Other (approx:) = 33.440¢/mt

Ball Milling, Total 167.200¢/mt

Table A-15. Summary of Grinding Costs (¢/mt), 1985

	Power	Media	Other	Totals	%
Rod Mill	35.87	17.67	13.39	66.93	(28.6%)
Ball Mill	89.21	44.55	33.44	167.20	(71.4%)
Total	125.08	62.22	46.83	234.13¢/mt	
(%)	(53.4%)	(26.6%)	(20%)	(100%)	

Note that with the completion of a hydroelectric power line to the site, electric power costs were reduced to 3.0¢/kwh in 1986. This resulted in the following revised grinding cost breakdown, assuming "other" costs are reduced 10% overall by the reduced electricity costs.

Table A-16. Summary of Grinding Costs (¢/mt), 1986

	Power	Media	Other	Totals	%
Rod Mill	9.96	17.67	12.05	39.68	(28.5%)
Ball Mill	24.78	44.55	30.10	99.43	(71.5%)
Total	34.74	62.22	42.15	139.11¢/mt	
(%)	(25.0%)	(44.7%)	(30.3%)	(100%)	

References - Appendix A

1. Berry, T.F., and Bruce, R.W., 1966, "A Simple Method of Determining the Grindability of Ores", Canadian Mining Journal, v.87, p. 63-65.

2. Roloff, C.A., 1979, internal Allis-Chalmers correspondence, Oct. 1.

3. Rowland, C.A., 1982, "Selection of Rod Mills, Ball Mills, Pebble Mills, and Regrind Mills", ch. 23, Design and Installation of Comminution Circuits, AIME, p.393-438.

4. Wood, K., 1976, internal memorandum on "Grindability Work Index Testing", Oct. 14.

5. Wood, K., 1978, internal memorandum on "B-Zone Underground Composites Grinding Requirements," July 27.

APPENDIX B

GENERAL AND HISTORICAL INFORMATION ON KIDD CREEK MINES CIRCUIT DESIGN AND OPERATION

General and Historical Information on Kidd Creek Mines

Circuit Design and Operation

Equipment Description (June, 1985)

Rod Mill:

Allis-Chalmers, 3.2 m (10.5 ft.) inside shell diameter by 4.88 m (16.0 ft.) effective working length, overflow discharge.

Liners, Shell; Noranda wave design, Ni-hard, 22 rows, 102 mm (4 in.) by 204 mm (8 in.) thick.
Ends; Smooth chrome molybdenum steel, 63.5 mm (2.5 in.) thick.

Media; 102 mm (4 in.) diameter by 4.72 m (15.5 ft.) long, SAE 1090 steel.

Drive, Motor; Tamper, 597 kw (800 HP), 1.15 s.f., 225 r.p.m., 4000 volts, 115 amps.
Clutch; Fawick 42 VC 650
Gear and pinion; Falk, 274 and 21 teeth
Mill speed is 17.24 r.p.m. or 70.2% of critical based on 2.9 m (9.75 ft.) inside diameter.

Primary Ball Mill:

Allis-Chalmers, 3.66 m (12 ft.) inside shell diameter by 5.49 m (18.0 ft.) effective working length, overflow discharge.

Liners; B.F. Goodrich rubber, 51 mm (2 in.) shell plates with 25 mm (1 in.) ribs, 127 mm (5 in.) lifters, 26 rows.

Media; 46 mm (1.81 in.) steel grinding balls.

Drive, Motor; Tamper, 1119 kw (1500 HP), 1.15 s.f., 225 r.p.m., 4000 volts, 214 amps.
Clutch; Fawick 42 VC 1200
Gear and pinion; Falk, 241 and 19 teeth.
Mill speed is 17.74 r.p.m. or 78.5% of critical based on 3.51 m (11.5 ft.) inside diameter.

Secondary Ball Mill:

Allis-Chalmers, 3.66 m (12 ft.) inside shell diameter by 5.49 m (18.0 ft.) effective working length, overflow discharge.

Liners; B.F. Goodrich and Trelleborg rubber, same as primary ball mill.

Media; 38 mm (1.5 in.) forged steel grinding balls.

Drive, Motor; Tamper, 1119 kw (1500 HP), 1.15 s.f., 225 r.p.m., 4000 volts, 214 amps.
Clutch; Fawick 42 VC 1200
Gear and pinion; Falk, 241 and 18 teeth
Mill speed is 16.80 r.p.m. or 74.3% of critical based on 3.51 m (11.5 ft.) inside diameter.

Regrind Mills:

Allis-Chalmers, 2.44 m (8 ft.) inside shell diameter by 3.66 m (12 ft.) effective working length, overflow discharge.

Liners; B.F. Goodrich rubber, same as ball mills, 20 rows.

Media; 25 mm (1.0 in.) forged steel grinding balls.

Drive, Motor; Tamper, 261 kw (350 HP), 1.15 s.f., 1187 r.p.m., 4000 volts, 47 amps.
Clutch; Fawick 11.5 VC 650
Reducer; Hamilton 184 CS, 4.956 to 1, Zurn coupling
Gear and pinion; Falk, 302 and 28 teeth
Mill speed is 22.20 r.p.m. (approx.) or 79.3% of critical based on 2.29 m (7.5 ft.) inside diameter.

Hydrocyclones:

Primary; 8 (5 operating), Krebs model D15B, 92 mm (3.625 in.) inlet, 63.5 mm (2.5 in.) vortex finder, 70 mm (2.75 in.) ceramic apex, radial cluster arrangement.

Secondary; 15 (13 operating), Krebs model D10B (extended body), 63.5 mm (2.5 in.) inlet, 76 mm (3 in.) vortex finder, 38 mm (1.5 in.) ceramic apex, radial cluster arrangement.

Regrind; 12 (copper regrind) and 16 (zinc regrind), Krebs model D6B (extended body), 63.5 m (2.5 in.) inlet, 38 mm (1.5 in.) vortex finder, 19 mm (0.75 in.) ceramic apex, radial cluster arrangement.

Pumps:

Primary; Operating, Linatex 12 x 10 model M3
Standby, Allis-Chalmers 12 x 10 SRL-C

Secondary; Operating, Linatex 10 x 12 model M
Standby, Allis-Chalmers 12 x 10 SRL-C

Regrind; Operating and standby, Allis-Chalmers 10 x 8 SRL-C

Instrumentation and Control

Details of the B grinding circuit process control equipment, instrumentation and hardware as of November, 1985, are shown on the enclosed drawing number 11111-02. The following is a summary description of the grinding circuit instrumentation and control practices in the plant at that time.

A. Rod Mill Feed

Rod mill feed tonnage is indicated and recorded on a strip chart, as well as on a digital accumulator, from a signal from a Merrick weightometer on the rod mill feed belt. Feed rate tonnage is set by the operator in the central control room, and is controlled by two variable speed belts from the fine ore bins.

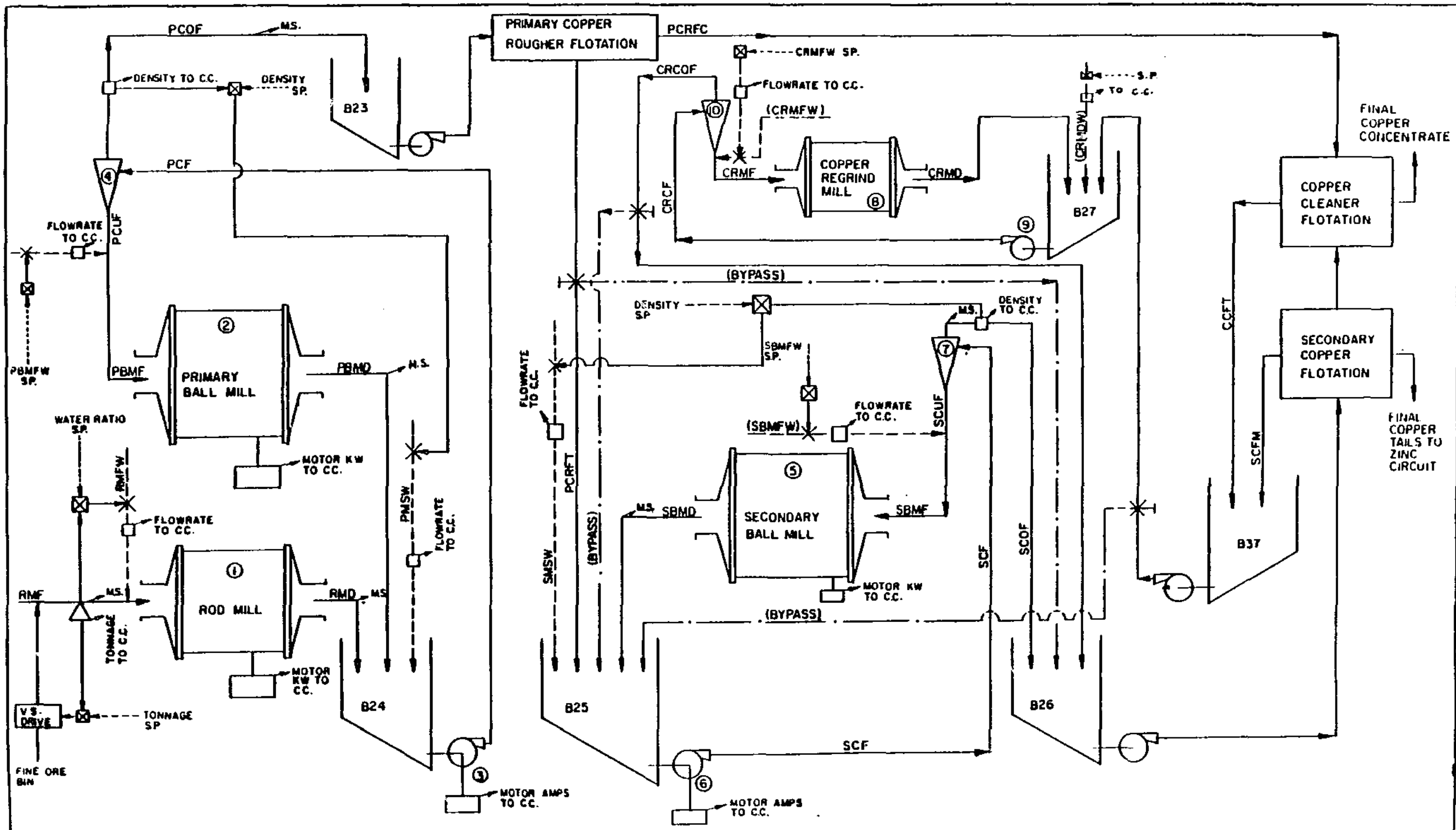

Figure B-1

CC = CENTRAL CONTROL
S.P. = SET POINT
M.S. = MANUAL SAMPLE POINT

DATE	REVISION	BY
28/3/87 1		
30/11/85 0	NEW DRAWING	

KIDD CREEK MINES LTD. LINE B
GRINDING CIRCUIT-INSTRUMENTATION AND CONTROL

DATE: NOV 30/1985 SCALE N.T.S.
DWG NO: 11111-02
REV. 1

Rod mill feed water is automatically proportioned with the rod mill feed to yield the desired rod mill feed density of 83-85% solids. Water addition is measured through an orifice plate and indicated in central control, and automatically adjusted through pneumatically actuated valves.

B. Primary Cyclone Overflow

Primary cyclone overflow density is measured by an Accuray density gauge, and indicated and recorded on a strip chart. The controller adjusts the primary cyclone feed water to maintain a setting of 52% solids in the overflow. Cyclone feed water is once again measured with an orifice plate and indicated in central control.

C. Primary Ball Mill Feed Water

Primary ball mill feed water is measured with a magnetic flowmeter and also indicated in central control. It is adjusted manually to maintain 81-82% solids in the primary ball mill discharge.

D. Secondary Cyclone Overflow Density

Secondary cyclone overflow density is measured by an Accuray density gauge, and indicated and recorded on a strip chart. The controller adjusts the cyclone feed water (measured by magnetic flowmeter) to maintain a maximum of 40% solids when there is otherwise insufficient water into the circuit.

E. Secondary Ball Mill Feed Water

Secondary ball mill feed water rate is indicated in central control, and is adjusted manually to maintain 78% solids in the secondary ball mill discharge, when necessary.

F. Copper Regrind Mill Feed Water

Copper regrind mill feed water is also indicated and can be controlled manually, but has never been added due to the low density of the feed into the regrind circuit.

G. Electrical

All grinding mill motor power input readings are provided in central control by kilowatt meters, and any one can be recorded on a strip chart at one time. Ammeter readings are provided for all pump motors. Electrical instrumentation in the motor control centre includes ammeters, power factor meters (where applicable), and line voltage for mill motors.

H. Other Instruments and Measurements

Other instrumentation not currently in use includes primary cyclone overflow particle size, primary cyclone mass flow (density and flow rate), and regrind cyclone feed densities. Manual samples are taken at the rod mill feed and discharge, and at each of the primary and secondary ball mill discharges.

A variety of mechanical and electrical indicators and alarms are also provided.

General Circuit Size Data

A series of sampling surveys were caried out in 1982 in which size distributions from rod mill feed through to secondary cyclone overflow were measured (1). Although precise details of the sampling procedures were not described, the data showed good consistency, and was therefore deemed suitable for preliminary evaluation purposes. A summary of the 80% passing sizes for the 7 complete surveys is given in Table B2-1.

Circuit Feed Variations

Total variation in rod mill feed size distribution for the 7 surveys was small, especially considering possible sampling variations on such a coarse stream. The plus 12.7 mm (0.5 in.) averaged 31%, with only 2 to 3 points variation when the most extreme (highest and lowest) values were excluded. The minus 105 um (100 mesh) averaged 11%, plus or minus 1 percentage point over the complete range of samples. The report on the 1982 sampling work also mentioned that there was no appreciable change in rod mill feed grindability, although details on the tests performed were not given.

Rod mill discharge (feed to the primary ball milling circuit), with the exception of test no. 412, which was exceptionally coarse, showed no more than 2 to 3 points of variation above or below the mean values of 39% plus 850 um (20 mesh), 46% minus 425 um (35 mesh), and 23% minus 53 um (270 mesh), all cummulative values.

Table B-1. Summary of 1982 Sampling Surveys

80% Passing Sizes (Microns) of Sampled Streams

Test No.	Rod Mill Feed	Rod Mill Disch.	Prim. Cyclone Feed	Prim. Cyclone Overflow	Prim. Ball Mill Feed	Prim. Ball Mill Disch.	Sec. Cyclone Feed	Sec. Cyclone Overflow	Sec. Ball Mill Feed	Sec. Ball Mill Disch.
408	16,200	1,585	725	129	820	520	111	35	135	112
411	16,100	1,475	530	124	800	385	101	35	139	105
412	15,900	1,840	705	132	985	490	109	35	145	109
414	15,600	1,705	690	130	955	505	113	40	154	113
418	15,800	1,385	765	131	835	485	112	28	153	103
419	17,400	1,410	565	125	705	390	96	25	132	93
420	16,800	1,710	820*	126	905	455	113	30	143	112
Mean	16,300	1,590	686	126	858	461	108	32.6	143	107
On-1	630	170	104	5.5	97	54.3	6.7	5.1	8.4	7.0

Most typical data are boxed
* Adjusted to 700u by mass balance calculations

Typical Size Distributions

Variations in the size distributions throughout the primary and secondary ball milling circuits can be observed from the data in Table 1, for which means and standard deviations were calculated as shown. While operating work index values can be calculated from combined averages, appropriate individual sets of size distribution data must be used for mass split (circulating load) and classifier cut size calculations.

No single test was found for which measured values of size distributions corresponded consistently (within one standard deviation) with calculated averages for all 10 sample points. However, test no. 408 appeared to be the best overall, with only the primary ball mill discharge falling on the coarse size of all other measured values for the same sample point.

A simple mass balance procedure (using linear regression analysis) was used to estimate the circulating load figures for this test run, and yielded values close to 400% for both the primary and secondary ball milling circuits. This revealed a second anomaly of this data set, namely, an unusually high circulating load, compared to the range of 130 to 230 percent for the other 6 tests on the secondary circuit. While not noticeable from inspection of the size distribution data, the unusually high figure results from a combination of slightly coarser cyclone feed, and slightly finer cyclone underflow, compared to the mean values.

As no single test yielded suitable size distribution data in both circuits, and because the time lag in primary flotation functionally separates the performance of the two circuits, primary and secondary ball milling circuits were considered separately. Data from test no. 420 gave excellent correlation with mean values, except for cyclone feed. Owing to the normal difficulty of obtaining reliable cyclone feed samples, this figure was checked by independent mass balance calculations (rod mill discharge plus ball mill discharge, and cyclone overflow plus underflow) which indicated a cyclone feed K80 size of 700 microns, instead of the reported 820. This then gave a good "typical" set of size distributions for the primary ball milling circuit, with an estimated circulating load of 390%.

Data from test no. 412 gave excellent correlation with the mean size distribution figures for the secondary ball milling circuit, with a circulating load of 160%, and so may be considered the "typical" case for the secondary circuit. The size distribution data for primary (test no. 420) and secondary (test no. 412) are summarized in Table B2-2.

Operating Work Indices

Preliminary operating work indices may be calculated from the average operating data as follows (2).

Rod Mill:	Size: 3.2m x 4.9m
	Power draw: 550 kw (assume 90% drive efficiency)
	Circuit tonnage: 131.5 dmtph
	F80 (average): 16,300 um
	P80 (average): 1,590 um
	Reduction ratio: 10.25

Table B-2.

Typical Size Distributions - "B" Circuit - Kidd Creek Mines

Cumulatiave Percent Passing

Primary Circuit - Test No. B4-420

Mesh Size	Rod Mill Feed	Rod Mill Disch.	Ball Mill Disch.	Prim. Cyc. Feed	Prim. Cyc.O.F.	Ball Mill Feed
3/4"	87.9					
1/2"	66.7					
3/8"	54.9					
1/4"	43.1					
4	38.7					
6	33.6	100		100		100
8	29.2	90.0	97.6	96.2		95.3
10	25.4	79.1	95.6	92.6		90.7
14	22.4	68.1	93.3	88.3		85.3
20	19.5	57.5	89.6	83.0		78.6
28	17.5	50.8	84.8	77.4	100	71.7
35	15.8	44.7	78.5	71.2	99.4	64.1
48	14.0	38.7	69.9	63.3	96.4	54.9
65	12.7	34.6	60.0	54.6	90.7	45.4
100	11.6	31.3	51.1	46.9	84.0	37.4
150	10.5	28.4	41.6	38.7	75.8	29.2
200	9.8	25.7(est)	32.5	30.9	66.9	21.8
270	8.7	23.6	26.2	25.7	59.1	17.1

Secondary Circuit - Test No. B4-412

Mesh Size	Ball Mill Disch.	Sec.Cyc.Feed	Sec.Cyc.O.F.	Ball Mill Feed
48	98.6	97.6	100	96.4
65	95.8	93.7	100	90.2
100	89.3	88.1	99.5	80.7
150	78.8	79.0	97.7	66.5
200	68.3	67.3	92.6	50.8
270	52.4	54.9	86.7	33.6
C5	10.9	14.6	38.5	6.11

Operating W.I.:

(W) Work requirement: $\frac{550 \times .90}{131.5} = 3.764$ kwh/mt

$$W.I. = W \Big/ \left(\frac{10}{\sqrt{P80}} - \frac{10}{\sqrt{F80}} \right)$$

$= \underline{21.82}$ kwh/mt

Correction Factors:

Diameter: $\underline{.956}$

Reduction Ratio: $Rro = \frac{8+5\ Lr}{D} = 15.75$

$$EF_6 = 1 - \frac{(Rr - Rro)^2}{150} = 1.20$$

Oversize Feed: (reiterative)

$$EF_4 = \frac{Rr - (WI-7)\ (F80 - Fo)\ /\ Fo}{Rr} = \underline{1.10}$$

Comparable Bond Laboratory W.I.:

$$\frac{21.82}{.956 \times 1.20 \times 1.10} = \underline{17.29 \text{ kwh/mt}}$$

Primary Ball Mill:

Size: 3.7m x 5.5m
Power draw: 666 kw (assume 90% drive efficiency)
Circuit tonnage: 131.5 dmtph
F80 (average): 1,590 um
P80 (average and test no. 420): 126 um
Reduction ratio: 12.6

Operating W.I.:

(W) Work requirement: $\frac{666 \times .9}{131.5} = 4.555$ kwh/mt

$$W.I. = W \Big/ \left(\frac{10}{\sqrt{P80}} - \frac{10}{\sqrt{F80}} \right)$$

$= \underline{7.116}$ kwh/mt

Correction Factors:

Diameter: $\underline{.930}$

Comparable Bond Laboratory W.I.:

$$\frac{7.116}{.930} = \underline{7.65 \text{ kwh/mt}}$$

Secondary and Copper Regrind Mills (combined):

Note that operating work indexes could not be calculated for the secondary ball milling and subsequent grinding stages from the given data. This is owing to the recycle streams from copper regrinding and copper secondary and cleaner flotation, which would influence the feed and product sizes for the circuit.

A combined secondary and copper regrind ball milling product size was estimated by reconstituting final copper concentrate and tailings monthly composite samples for 1984, which yielded an 80% passing size of approximately 68 um (3). The operating work index would then be estimated as follows.

Secondary Ball Mill and Regrind Mill (combined):

Sizes: 3.7 m x 5.5 m, 2.4 m x 3.7 m
Power draw: 645 + 154 = 799 kw (assume 90% drive eff.)
Circuit tonnage: 131.5 DMTPH
F80 (average): 126 um
P80 (average): 68 um

Operating W.I.:

(W) Work requirement: $\frac{799 \times .90}{131.5} = 5.47$ kwh/mt

$$\text{W.I.} = W \Big/ \left(\frac{10}{\sqrt{P80}} - \frac{10}{\sqrt{F80}} \right)$$

$$= \underline{17.0} \text{ kwh/mt}$$

Classifier Performance

Preliminary operating data based on the selected typical size distributions and normal operating densities for primary and secondary grinding circuit hydrocyclones are given in Tables B-3 and 4.

Table B-3

Preliminary Hydrocyclone Performance Data

	Primary Ball Mill Circuit	Secondary Ball Mill Circuit
Circuit Tonnage (DMTPH)	131.5	124.9
Densities (% solids by wt.)		
Feed	73.5	51.9
Overflow	54	35
Underflow	81	74.5
Circulating Load Ratio (%)	390	160

Table B-4

Hydrocyclone Recoveries From Typical Size Distributions

Mesh Size	% Recoveries to Underflow: Primary Cyclones (Test B4-420)	% Recoveries to Underflow: Secondary Cyclones (Test B4-412)
8	100	
10	100	
14	100	
20	100	
28	100	
35	98.0	
48	92.4	100
65	86.9	100
100	82.4	96.5
150	79.7	93.0
200	76.6	83.1
270	70.2	82.4
-270	53.4	41.9
	Bypass (water recovery to U.F.) = 52%	Bypass (water recovery to U.F.) = 23%

Grinding Costs

Details of historical power, grinding media, and "other", costs for all 5 stages of grinding is as follows:

Rod Mill

Power:	$\frac{550}{131.5}$ kwh x 2.944 ¢/kwh =	12.313 ¢/ton
Media:	.508 kg/t x 58.66 ¢/kg =	29.799 ¢/ton
Other:	Non-grinding power, labour, maintenance, overhead	22.132 ¢/ton
Total		64.244 ¢/ton

Primary Ball Mill

Power:	$\frac{666}{131.5}$ kwh x 2.944 =	14.910 ¢/ton
Media:	.617 kg/t x 62.77 ¢/kg =	38.729 ¢/ton
Other:		28.190 ¢/ton
Total:		81.829 ¢/ton

Secondary Ball Mill

Power:	$\frac{645}{131.5}$ kwh x 2.944 =	14.440 ¢/ton
Media:	.624 kg/t x 54.60 ¢/kg =	34.070 ¢/ton
Other:		25.495 ¢/ton
Total:		74.005 ¢/ton

Copper Regrind

Power: $\frac{154}{131.5}$ kwh x 2.944 ¢/kwh = 3.448 ¢/ton

Media: .130 kg/t x 46.58 ¢/kg = 5.925 ¢/ton

Other: 4.926 ¢/ton

Total: 14.299 ¢/ton

Zinc Regrind

Power: $\frac{154}{131.5}$ kwh x 2.944 ¢/kwh = 3.448 ¢/ton

Media: .115 kg/t x 46.58 ¢/kg = 5.357 ¢/ton

Other: 4.627 ¢/ton

Total: 13.432 ¢/ton

Summary - Total Grinding Costs

	Power	Media	Other	Totals	%
Rod Mill	12.313	29.799	22.132	62.244	(25.9)
Primary Ball Mill	14.910	38.729	28.190	81.829	(33.0)
Secondary Ball Mill	14.440	34.070	25.495	74.005	(29.9)
Copper Regrind	3.448	5.925	4.926	14.299	(5.8)
Zinc Regrind	3.448	5.357	4.627	13.432	(5.4)
Totals	48.559	113.880	85.370	247.809 (¢/ton)	
	(19.6%)	(46%)	(34.4%)		(100%)

References - Appendix B

1. de Friedberg, A.S., 1982, "Summary of Summer Project on B Grinding Circuit Data Collection", Kidd Creek Mines internal memorandum, Sept. 12.

2. Rowland, C.A., 1982, "Selection of Rod Mills, Ball Mills, Pebble Mills, and Regrind Mills", Ch.23, Design and Installation of Comminution Circuits, AIME, p. 393-438.

3. Scheding, W., 1985, Kidd Creek Mines test summary report BGS-2, Dec. 18.

APPENDIX C

TABLE OF REPORTED INDUSTRIAL ROD MILL OPERATING DATA

Note: Units used are those as reported.
Mill dimensions are nominal only.

ROD MILLING - OPERATING PLANT DATA

Plant	Old Dominion	Colquiri	Copper Queen	Moctezama
Reference	Taggart	Taggart	Taggart	Taggart
Date of Reference or Data	1945	1945	1945	1945
S.T.P.H. (ea. mill)	8.7	14	22.5	18.3
Size (dia. x length, ft.)	4 x 8	4 x 8	6 x 12	6 x 12
H.P.: Motor rating	50	50	150	-
Draw	39 H.P.	46 (est.)	125 (est.)	120 (est.)
% Solids (S.G. Ore)	74%	-	75%	78%
Feed: % + 3/4" / top size	- / 3 m	5-10%	37%	29%
20% retained (um)	4,100	6,000	25,000	26,000
Other	2% - 200 m	3% - 200 m	All ` 2" 13% - 200 m	7% - 200 m
Product: Top size (mesh)	14	10	20	6
20% retained (um)	350	600	250	840
other	30% - 200 m	25% - 200 m	46% - 200 m	28% - 200 m
Rod Size	-	3"	3"	3"
Speed: r.p.m.	25	24.7	17.5	17.5
% Cs	65%	64%	56%	56%
Recirculation	O.C.*	O.C.	O.C.	O.C.
Reduction Ratio	11.7	10	100	31
Power: kwh/t	3.34	2.49	4.14	5.0
Rod Consump.: lb/t	1.72	0.46	1.34	0.99
lb/kwh	0.51	0.18	0.32	0.20
Operating W.I.: kwh/st.	8.8	8.9	7.3	17.7

* O.C. = Open Circuit

ROD MILLING - OPERATING PLANT DATA

Plant	Fresnillo	Balmat		Chino
Reference	Taggart	Taggart	Myers et al	Taggart
Date of Reference or Data	1945	1945	1947	1945
S.T.P.H. (ea. mill)	15.5	52	42	62.5
Size (dia. x length, ft.)	6 1/2 x 12	6 1/2 x 12	same	7 x 10
H.P.: Motor rating	150	240	same	175
Draw	154	177	182	155
% Solids (S.G. Ore)	-	73%	-	65%
Feed: % + 3/4"/top size	21%	- / 1/2"	- / -1/2"	- / 3 m
20% retained (um)	19,000	5,600	6,000	1,600
Other	2% - 200	-	6% - 200 m	22% - 200
Product: Top size (mesh)	8	20	14	8
20% retained (um)	640	280	330	580
Other	30% - 200	25% - 200	23% - 200 m	33% - 200 m
Rod Size	3"	2"	2 1/2"	3"
Speed: r.p.m.	17	16.5	same	19.9
% Cs	56%	55%	same	69%
Recirculation	O.C.	O.C.	O.C.	O.C.
Reduction Ratio	30	20	18	3
Power: kwh/t	7.5	2.57	3.24	1.85
Rod Consump.: lb/t	1.66	1.3	1.08	0.4
lb/kwh	0.22	0.50	0.33	0.22
Operating W.I.: kwh/st	23.3	5.5	7.7	11.2

ROD MILLING - OPERATING PLANT DATA

Plant	Waite Amulet	TVA Fontana Dam	Reserve Mining	
Reference	Myers et al	Myers et al	Myers et al	Myers et al
Date of Reference or Data	1947	1947	1947	1947
S.T.P.H. (ea. mill)	52.1	136.4	2.3	4.8
Size (dia. x length, ft.)	9 x 11 1/2	9 x 12	3 x 6	9 1/2 x 12
H.P.: Motor rating				
Draw	300	305	23.4	23.4
% Solids (S.G. Ore)	83%	-	-	-
Feed: % + 3/4"/top size	9% / 1.5"	3% / 1"	4%	4%
20% retained (um)	16,000	11,000	13,000	13,000
Other	6% - 200 m	7% - 200 m	4% - 200 m	4% - 200 m
Product: Top size (mesh)	6	4	14	4
20% retained (um)	1,000	1,400	390	1,350
Other	24% - 200 m	16% - 200 m	4% - 200 m	4% - 200 m
Rod Size	3"	-	3 1/2"	3 1/2"
Speed: r.p.m.	16	15.5	35	same
% Cs	61.2%	59.4%	74.5%	same
Recirculation	O.C.	O.C.	O.C.	O.C.
Reduction Ratio	16	8	33	10
Power: kwh/t	4.3	1.67	7.54	3.62
Rod Consump.: lb/t	.94	-	-	-
lb/kwh	.22			
Operating W.I.: kwh/st	18.1	9.7	18.0	15.5

ROD MILLING - OPERATING PLANT DATA

Plant	J & L Benson	Kerr Addison		Quemont
Reference	Myers et al	Myers et al	Hawkes	McLachlan et al
Date of Reference or Data	1947	1947	1957a	1953
S.T.P.H. (ea. mill)	143	54.6	65	85
Size (dia. x length, ft.)	9 1/2 x 12	8 x 12	same	9 1/2 x 12
H.P.: Motor rating				400
Draw	418	236	300	-
% Solids (S.G. Ore)	-	81%	80%	-
Feed: % + 3/4"/top size	12% / 1.5"	13% / 1"	5% / 1"	25% /1 1/2"
20% retained (um)	16,000	16,000	16,000	20,000
Other	4% - 200 m	7% - 200 m	12% - 200 m	
Product: Top size (mesh)	8	3	3	
20% retained (um)	700	1,750	1,750	1,000
Other	15% - 200 m	22% - 200 m	38% - 200 m	
Rod Size	3 1/2"	3 1/2"	3 1/2"	3 1/2"
Speed: r.p.m.	14.6	17	17	15.7
% Cs	57.2	60.7		
Recirculation	O.C.	O.C.	O.C.	C.C.*
Reduction Ratio	23	9	9	20
Power: kwh/t	2.18	3.23	3.4	4.7
Rod Consump.: lb/t	-	0.67	0.74	.77
lb/kwh		.21	.22	.16
Operating W.I.: kwh/st	7.3	20.2	21.25	19.2

* C.C. = Closed Circuit

ROD MILLING - OPERATING PLANT DATA

Plant		Hayden / Ray (Kennecott)	
Reference	anon; Tuck	Myers et al	Myers et al
Date of Reference or Data	1938 / 1945	1947	1947
S.T.P.H. (ea. mill)	125	100	137.5
Size (dia. x length, ft.)	9 x 12	same	same
H.P.: Motor rating	350	same	same
Draw	352	345	345
% Solids (S.G. Ore)	70%	75%	75%
Feed: % + 3/4"/top size	4% / 1"	5%	6.5%
20% retained (um)	12,000	11,000	13,500
Other	15% - 200 m	34% - 200 m	27% - 200 m
Product: Top size (mesh)	4	6	4
20% retained (um)	1,150	1,000	1,450
Other	33% - 200 m	34% - 200 m	27% - 200 m
Rod Size	3"	same	same
Speed: r.p.m.	14	same	same
% Cs	55%	same	same
Recirculation	O.C.	O.C.	O.C.
Reduction Ratio	10	11	9
Power: kwh/t	2.13	2.57	1.87
Rod Consump.: lb/t	.38	.473	.334
lb/kwh	.178	.184	.179
Operating W.I.: kwh/st	10.5	11.6	10.6

ROD MILLING - OPERATING PLANT DATA

Plant	National Lead		
Reference	Myers et al	Strohl and Schewellenbach	
Date of Reference or Data	1947	1950	
S.T.P.H. (ea. mill)	41.5	38.4	42.4
Size (dia. x length, ft.)	6 1/2 x 12	same	same
H.P.: Motor rating			
Draw	240	230	230
% Solids (S.G. Ore)	65%		
Feed: % + 3/4"/top size		3/4" limiting	8/16" limiting
20% retained (um)	13,000	12,000	10,000
Other	4% - 200 m		
Product: Top size (mesh)	8		
20% retained (um)	400	500	400
Other	24% - 200 m	18% - 200 m	20% - 200 m
Rod Size	2.5"	same	same
Speed: r.p.m.	22	same	same
% Cs	71.3%	same	same
Recirculation		.48" screen	.29" screen
Reduction Ratio	30	24	25
Power: kwh/t	4.32	4.47	4.05
Rod Consump.: lb/t	1.35		
lb/kwh	.31		
Operating W.I.: kwh/st	10.6	12.6	10.1

ROD MILLING - OPERATING PLANT DATA

Plant	Tennessee Copper - London Mill			
Reference	Myers & Lewis	Myers & Lewis	Myers & Lewis	Myers & Lewis
Date of Reference or Data	1928	1929	Later	Later
S.T.P.H. (ea. mill)	12.5	13.75	52.25	51.83
Size (dia. x length, ft.)	6 x 12	same	same	same
H.P.: Motor rating				
H.P.: Draw	157	same	176	186
% Solids (S.G. Ore)	67% (3.3)		67%	67%
Feed: % + 3/4"/top size	4.3%			
Feed: 20% retained (um)	10,000		11,500	14,000
Feed: Other				
Product: Top size (mesh)				
Product: 20% retained (um)	130		350	350
Product: Other				
Rod Size	3"		3"	3"
Speed: r.p.m.	17	19	19	22
Speed: % Cs				
Recirculation	C.C. Rake Class		O.C.	O.C.
Reduction Ratio	69		33	40
Power: kwh/t	9.37		2.507	2.623
Rod Consump.: lb/t	2.74		.687	.675
Rod Consump.: lb/kwh	.292		.274	.257
Operating W.I.: kwh/st	12.1	10.8	5.7	5.8

ROD MILLING - OPERATING PLANT DATA

Plant	Tennessee Copper - London Mill		
Reference	Myers & Lewis	Myers & Lewis	Myers
Date of Reference or Data	Later	Later	1948
S.T.P.H. (ea. mill)	50	50	80
Size (dia. x length, ft.)	same	same	same
H.P.: Motor rating			
Draw	183	183	218
% Solids (S.G. Ore)	67%	67%	78%
Feed: % + 3/4"/top size			
20% retained (um)	11,000	21,000	13,500
Other			
Product: Top size (mesh)			
20% retained (um)	350	390	400
Other			
Rod Size	3"	3"	3"
Speed: r.p.m.	22	22	26.2
% Cs			
Recirculation	O.C.	O.C.	O.C.
Reduction Ratio	31	54	34
Power: kwh/t	2.72	2.72	2.03
Rod Consump.: lb/t			
lb/kwh			
Operating W.I.: kwh/st	6.2	6.2	4.9

ROD MILLING - OPERATING PLANT DATA

Plant	Tennessee Copper - London Mill		
Reference	Myers et al; Myers and Lewis		
Date of Reference or Data		1946	
S.T.P.H. (ea. mill)	52	62.5	83.3
Size (dia. x length, ft.)	6 x 12	same	same
H.P.: Motor rating			
Draw	220	same	same
% Solids (S.G. Ore)	77%	same	same
Feed: % + 3/4"/top size			
20% retained (um)	9,000	9,000	9,000
Other			
Product: Top size (mesh)			
20% retained (um)	330	380	415
Other			
Rod Size	3"	same	same
Speed: r.p.m.	24.4	same	same
% Cs	76.2%	same	same
Recirculation	O.C.	O.C.	O.C.
Reduction Ratio	27.3	23.7	21.7
Power: kwh/t	2.885	2.400	1.800
Rod Consump.: lb/t	.61		.46
lb/kwh	.211		.255
Operating W.I.: kwh/st	6.48	5.88	4.66

ROD MILLING - OPERATING PLANT DATA

Plant	Tennessee Copper - Isabella Mill		
Reference	Myers and Lewis		
Date of Reference or Data	1946		
S.T.P.H. (ea. mill)	39	44	44
Size (dia. x length, ft.)	6 x 9	same	same
H.P.: Motor rating			
Draw	140	165	174
% Solids (S.G. Ore)			
Feed: % + 3/4"/top size	5% / 1"	same	same
20% retained (um)	17,000	same	same
Other	5% - 200 m	same	same
Product: Top size (mesh)			
20% retained (um)			
Other			
Rod Size	3"	3"	2 1/2"
Speed: r.p.m.	24.4	same	same
% Cs	76.2%	same	same
Recirculation	O.C.	O.C.	O.C.
Reduction Ratio			
Power: kwh/t	2.68	2.80	2.95
Rod Consump.: lb/t	.61	.63	.77
lb/kwh	.228	.225	.261
Operating W.I.: kwh/st			

ROD MILLING - OPERATING PLANT DATA

Plant	Tennessee Copper - Isabella Mill			
Reference	Myers and Lewis			
Date of Reference or Data	1946			
S.T.P.H. (ea. mill)	44	44	44	44
Size (dia. x length, ft.)	6 x 9	same	same	same
H.P.: Motor rating				
Draw	165	174	174	201-205
% Solids (S.G. Ore)			78%	78%
Feed: % + 3/4"/top size				
20% retained (u)	12,000	17,000	12,000	12,000
Other	40% + 3m	54% + 3m		
Product: Top size (mesh)				
20% retained (um)	410	405	370	340-350
Other				
Rod Size	3"	2 1/2"	2 1/2"	same
Speed: r.p.m.	24.4	same		
% Cs	76.2%	same		
Recirculation	O.C.	O.C.	O.C.	O.C.
Reduction Ratio			25	29
Power: kwh/t	2.80	2.95	2.95	3.41 - 3.47
Rod Consump.: lb/t	.63	.77		
lb/kwh	.225	.261		
Operating W.I.: kwh/st	7.0	7.0	6.9	7.7

ROD MILLING - OPERATING PLANT DATA

Plant	Tennessee Copper - Isabella Mill	
Reference	Taggart	Myers and Lewis
Date of Reference or Data	1947	1949
S.T.P.H. (ea. mill)	44	46
Size (dia. x length, ft.)	6 x 9	6 x 9
H.P.: Motor rating		
Draw	172	181
% Solids (S.G. Ore)	78%	79%
Feed: % + 3/4"/top size	0 / 3/4"	
20% retained (um)	12,000	14,000
Other	6.5%-200 m	
Product: Top size (mesh)	10	
20% retained (um)	400	440
ther	30% - 200 m	
Rod Size	1 1/2"	
Speed: r.p.m.	24.4	24
% Cs	76.4%	
Recirculation	O.C.	O.C.
Reduction Ratio	29	32
Power: kwh/t	2.93	2.93
Rod Consump.: lb/t	.77	
lb/kwh	.263	
Operating W.I.: kwh/st	7.2	7.5

ROD MILLING - OPERATING PLANT DATA

Plant	Tennessee Copper - Isabella Mill			
Reference	Lewis and Goodman			
Date of Reference or Data	1957			
S.T.P.H. (ea. mill)	48	52	62.5	102
Size (dia. x length, ft.)	6 x 12	same	same	same
H.P.: Motor rating				
Draw	215			215
% Solids (S.G. Ore)				
Feed: % + 3/4"/top size				
20% retained (um)	15,000		14,500	
Other				
Product: Top size (mesh)				
20% retained (um)	390		475	
other				
Rod Size	3"	same	same	same
Speed: r.p.m.	24	same	same	same
% Cs	74%	same	same	same
Recirculation	O.C.	O.C.	O.C.	O.C.
Reduction Ratio	38		31	
Power: kwh/t	3.34		2.56	
Rod Consump.: lb/t	.72		.41	
lb/kwh	.22		.16	
Operating W.I.: kwh/st	7.9	6.2	6.8	4.0

ROD MILLING - OPERATING PLANT DATA

Plant	N.J. Zinc Eagle Mill	Gaspe	Inco Creighton	Cam. Chib.
Reference	Craig	Wearing	Staff	Siscoe
Date of Reference or Data	Oct. 1950	1957	1957	1957
S.T.P.H. (ea. mill)	23.5	141	130 (est.)	74 (est.)
Size (dia. x length, ft.)	4 x 10	11 x 13'9"	10 1/2 x 13	10 x 12
H.P.: Motor rating		800	800	600
Draw	80	(Ass. 90%)	(Ass. 600)	(Ass. 540)
% Solids (S.G. Ore)	-	-	70 - 72%	83% (3.2)
Feed: % + 3/4"/top size	- / 3/4"	37% / 1 1/2"	11% / 1"	18% / 1"
20% retained (um)	6,600	25,000	16,000	19,000
Other	9% - 65 m	4% - 200 m	6% - 200 m	8% - 200 m
Product: Top size (mesh)	14	3	4	3
20% retained (um)	340	2400	1400	2000
Oother	59% - 65 m	22% - 200 m	22% - 200 m	31% - 200 m
Rod Size	2 1/2"	3 1/2"	3"	3 1/2"
Speed: r.p.m.	23	14.9		16.8
% Cs	57%		59.5%	
Recirculation	O.C.	C.C. Spiral Class	O.C.	C.C. Spiral Class
Reduction Ratio	19.4	10.4	11.4	10
Power: kwh/t	3.4	4.3	3.4	5.4
Rod Consump.: lb/t	1.0	1.0	.98	.93
lb/kwh	.29	0.23	.28	.17
Operating W.I.: kwh/st	8.1	30.5	18.1	35.8

ROD MILLING - OPERATING PLANT DATA

Plant	Cominco Bluebell	Canadian Exploration	Can. Ex. Tungsten
Reference	Walton	Steane	Steane
Date of Reference or Data	1957	1957	1957
S.T.P.H. (ea. mill)	31 (est)	74 (est)	30 (est)
Size (dia. x length, ft.)	6 x 12	6 x 12	6 x 12
H.P.: Motor rating	250 (oversize)		
Draw	(Ass. 150)	(Ass. 150)	(Ass. 150)
% Solids (S.G. Ore)	83%	80%	76 - 80%
Feed: % + 3/4"/top size	35% / 1 1/2"	16% / 1 1/4"	16% / 1 1/4"
20% retained (um)	25,000	18,000	18,000
Other		10% - 200 m	10% - 200 m
Product: Top size (mesh)	8 - 10	10	8 - 10
20% retained (um)	850	600	1050
Other	20% - 200 m	30% - 200 m	28% - 200m
Rod Size	3"	3"	3"
Speed: r.p.m.	23.3	18	24.4
% Cs	70.4%	65%	75%
Recirculation	15% Atkins sand recov.	O.C.	O.C.
Reduction Ratio	29	30	17
Power: kwh/t	3.6	1.5	4.4
Rod Consump.: lb/t	.45	.2	.7
lb/kwh	.13	.13	.16
Operating W.I.: kwh/st	12.9	4.5	18.8

ROD MILLING - OPERATING PLANT DATA

Plant	Brenda	Craigmont	Granby
Reference	Bradburn	Hawthorn	Harwicke
Date of Reference or Data	1978	1978	1978
S.T.P.H. (ea. mill)	320 (est.)	112	62.5 (est.)
Size (dia. x length, ft.)	13 1/2 x 18	9 1/2 x 12	8 x 12
H.P.: Motor rating	1950		400
Draw	(Ass. 1600)	(Ass. 450)	(Ass. 350)
% Solids (S.G. Ore)			
Feed: % + 3/4"/top size			
20% retained (um)	16,000	15,000	12,500
Other			
Product: Top size (mesh)		3 m	
20% retained (um)	1,350	1,900	1,350
Other		22% - 200 m	
Rod Size	4"		3 1/2 "
Speed: r.p.m.			21
% Cs			76%
Recirculation	O.C.	O.C.	O.C.
Reduction Ratio	12	8	9
Power: kwh/t	5.0	3.0	4
Rod Consump.: lb/t	1.0		.70
lb/kwh	.20		.18
Operating W.I.: kwh/st	25.9	20.4	22

ROD MILLING - OPERATING PLANT DATA

Plant	Falconbridge Lac Dufault	Adams	Texada
Reference	Wright & Patel	Goode & Vedova	Haig-Smillie and Walker
Date of Reference or Data	1978	1978	1978
S.T.P.H. (ea. mill)	70 (est.)	150 (long tons)	1000 (long tons conc.)
Size (dia. x length, ft.)	9 x 12	11 1/2 x 18	9 x 14
H.P.: Motor rating	450	1250	
Draw	(Ass. 425)	(Ass. 1000)	(Ass. 450)
% Solids (S.G. Ore)			83% (4.4)
Feed: % + 3/4"/top size	5%		
20% retained (um)	15,000	14,000	16,000
Other			
Product: Top size (mesh)	8 - 10 m		
20% retained (um)	700	1200	800
Other			
Rod Size	3 1/2"	4 1/2"	3 1/2"
Speed: r.p.m.			
% Cs	70%		67%
Recirculation	O.C.	O.C.	O.C.
Reduction Ratio	21	12	20
Power: kwh/t	4.5	5 (long)	.5 (approx.)
Rod Consump.: lb/t	1.3	1.7	.7
lb/kwh	.29	.34	1.4
Operating W.I.: (kwh/st)	15.2	25	1.8

ROD MILLING - OPERATING PLANT DATA

Plant	Sullivan		
Reference	Banks	Brock	Jacobi
Date of Reference or Data	1953	1957	1978
S.T.P.H. (ea. mill)	333	208	156
Size (dia. x length, ft.)	11 1/2 x 12	same	same
H.P.: Motor rating			
Draw	950	950	670
% Solids (S.G. Ore)	83%	80%	
Feed: % + 3/4"/top size	29%	62%	
20% retained (um)	27,000	30,000	30,000
Other	(no fines in feed)		
Product: Top size (mesh)			
20% retained (um)	2,200	2,000	
Other			
Rod Size	3 1/2"	3 1/2"	3 1/2"
Speed: r.p.m.	19	19	17
% Cs	81%	81%	74%
Recirculation	O.C.	O.C.	Class Sands Recirc.
Reduction Ratio	12	15	
Power: kwh/t	2.12	3.40	
Rod Consump.: lb/t	.354	.24	
lb/kwh	.166	.070	
Operating W.I.: kwh/st	13.9	20.5	

<u>ROD MILLING - OPERATING PLANT DATA</u>

Plant	Butte	Twin Buttes	Yerington	Cananea
Reference	Bassarear	Bassarear	Bassarear	Bassarear
Date of Reference or Data	1985			
S.T.P.H. (ea. mill)	310	500	288	250
Size (dia. x length, ft.)	9 x 12	14 x 18 1/2	10 x 14	9 x 12
H.P.: Motor rating	450	1750	1250	450
Draw				
% Solids (S.G. Ore)	70	77	-	70
Feed: % + 3/4"/top size	22% / 1 1/2	/ 3/4"		
20% retained (um)	20,000	12,000	12,000	10,000
Other				
Product: Top size (mesh)	13% on 14 m 24% on 28 m			
20% retained (um)	1,000	1,700	2,000	3,000
Other				
Rod Size	3 1/2"	4"	3"	4"
Speed: r.p.m.	16.7	12.5	16.1	16.6
% Cs	65%	61%	66%	65%
Recirculation				
Reduction Ratio	20	7	5.5	3.3
Power: kwh/t	1.0	2.6	1.75	1.47
Rod Consump.: lb/t	.30	0.8	.42	.28
lb/kwh	.30	.31	.24	.19
Operating W.I.: kwh/st	4.1	17.0	13.2	17.8

ROD MILLING - OPERATING PLANT DATA

Plant	Mission	Bethlehem	Boliden	Craigmont
Reference	Bassarear	Bassarear	Bassarear	Bassarear
Date of Reference or Data				
S.T.P.H. (ea. mill)	160	625	200	110
Size (dia. x length, ft.)	10 1/2 x 15	12 1/2 x 15	10 x 15	9 1/2 x 12
H.P.: Motor rating	900	1250	800	600
Draw				
% Solids (S.G. Ore)	77	80	68	80
Feed: % + 3/4"/top size				
20% retained (um)	11,000	11,000	20,000	13,000 - 15,000
Other				
Product: Top size (mesh)				
20% retained (um)	1,800	1,900		1,600 - 1,900
Other				
Rod Size	4"	3 1/2"		3 1/2"
Speed: r.p.m.	15.8	15.9	15.6	
% Cs	66	72	65	75
Recirculation				
Reduction Ratio	6.1	5.8		8.1 - 7.9
Power: kwh/t	3.4	2.84	2.85	2.8
Rod Consump.: lb/t	.62	.90	.85	.55
lb/kwh	.18	.32	.30	.20
Operating W.I.: kwh/st	24.2	21.1		17.3 - 20.4

ROD MILLING - OPERATING PLANT DATA

Plant	Granduc	Christmas	Ray	Ray
Reference	Bassarear	Bassarear	Bassarear	Bassarear
Date of Reference or Data	1985			
S.T.P.H. (ea. mill)	187	250	162	210
Size (dia. x length, ft.)	10 1/2 x 16	11 x 16	9 x 12	10 1/2 x 14
H.P.: Motor rating	1000		500	900
Draw			(Ass. 450)	
% Solids (S.G. Ore)	80	70	51	52
Feed: % + 3/4"/top size				
20% retained (um)	21,000		10,000	
Other				
Product: Top size (mesh)				
20% retained (um)	2,000		2,000	
Other				
Rod Size	3"	4"	3"	2"
Speed: r.p.m.	15.0	14.3	18.3	16.5
% Cs	63	60	70	68
Recirculation				
Reduction Ratio	10.5		5	
Power: kwh/t	3.0	2.2	2.1	
Rod Consump.: lb/t	.75	.44	.44	.44
lb/kwh	.25	.20	.21	
Operating W.I.: kwh/st	19.4		17	

ROD MILLING - OPERATING PLANT DATA

Plant	Bonneville	Lake Dufault	San Manuel	San Manuel
Reference	Bassarear	Bassarear	Bassarear	Bassarear
Date of Reference or Data	1985			
S.T.P.H. (ea. mill)	300	69	180	130
Size (dia. x length, ft.)	12 1/2x18 1/2	9 x 12	10 x 13	12 1/2 x 16
H.P.: Motor rating	1250	450	700	1650
Draw				
% Solids (S.G. Ore)	72	82	71	71
Feed: % + 3/4"/top size				
20% retained (um)	20,000	10,000	15,000	
Other				
Product: Top size (mesh)				
20% retained (um)	1,500	1,300	1,400	
Other				
Rod Size	4"	3 1/2"	3 1/2"	3 1/2"
Speed: r.p.m.	13.7	19	15.8	14.1
% Cs	63	73	66	63
Recirculation				
Reduction Ratio	13.3		10.7	
Power: kwh/t	2.75	4.6	3.36	
Rod Consump.: lb/t	.63	1.3 - 1.55	.60	
lb/kwh	.23	.29 - .33	.18	
Operating W.I.: kwh/st	14.6	26	18.1	

ROD MILLING - OPERATING PLANT DATA

Plant	Marcopper	Mitsubishi	MT.Isa	Chingola East
Reference	Bassarear	Bassarear	Bassarear	Bassarear
Date of Reference or Data	1985			
S.T.P.H. (ea. mill)	400	48	385	210
Size (dia. x length, ft.)	13 1/2 x 20	8 x 12	12 1/2 x 18	11 1/2x14 1/2
H.P.: Motor rating	1800	350	1,500	1,250
Draw				
% Solids (S.G. Ore)	78	70	80	75
Feed: % + 3/4"/top size				
20% retained (um)	16,500		10,000	20,000
Other				
Product: Top size (mesh)				
20% retained (um)	1,200		3,000	700
Other				
Rod Size	4"	3"	3"	3 1/2"
Speed: r.p.m.	13.5	17	15	12.8
% Cs	63	62	67	57
Recirculation				
Reduction Ratio	13.75		3	28
Power: kwh/	3.7	3.6	2.69	2.0
Rod Consump.: lb/t	0.70	1.0	0.8	0.75
lb/kwh	0.19	.28	.30	.38
Operating W.I.: kwh/st	17.6		33	6.5

ROD MILLING - OPERATING PLANT DATA

Plant	Rokana	Geco	Virtasalmi	Palabora
Reference	Bassarear	Bassarear	Bassarear	Bassarear
Date of Reference or Data	1985			
S.T.P.H. (ea. mill)	85	230	30	400
Size (dia. x length, ft.)	9 x 12	12 x 14	71/4 x 151/2	12 x 16
H.P.: Motor rating	400	900	210	1250
Draw				
% Solids (S.G. Ore)	82	81	80	80
Feed: % + 3/4"/top size				
20% retained (um)	35,000	12,000	10,000	14,000
Other				
Product: Top size (mesh)				
20% retained (um)	1,000	800	900	900
Other				
Rod Size	3 1/2"	3 1/2"	3"	4"
Speed: r.p.m.	19	14	25.5	13.9
% Cs	73	62	78	63
Recirculation				
Reduction Ratio	35	15	11	16
Power: kwh/t	2.4	5.4	3.7	2.3
Rod Consump.: lb/t	0.43	1.53	0.7	0.30
lb/kwh	0.18	0.28	0.19	0.13
Operating W.I.: kwh/st	9.2	21	16	9

ROD MILLING - OPERATING PLANT DATA

Plant	Pima	Toquepala	Kakanda	Kidd
Reference	Bassarear	Bassarear	Bassarear	Bassarear
Date of Reference or Data	1985			
S.T.P.H. (ea. mill)	170	230	55	144
Size (dia. x length, ft.)	10 x 16	10 x 14	7 x 12	10 1/2 x 16
H.P.: Motor rating	800	800	300	800
Draw				
% Solids (S.G. Ore)	75	75	70	82
Feed: % + 3/4"/top size				
20% retained (um)	13,000	19,000	25,000	25,000
Other				
Product: Top size (mesh)				
20% retained (um)	1,700	1,500	2,400	1,800
Other				
Rod Size	4"	3 1/2"	2 1/2", 3"	4"
Speed: r.p.m.	16.8	17	22	16
% Cs	67	69	77	63
Recirculation				
Reduction Ratio	7.6	12.7	10.4	14
Power: kwh/t	3.75	2.45	5.0	4.04
Rod Consump.: lb/t	.60	.50	1.18	1.05
lb/kwh	0.16	.20	.24	.26
Operating W.I.: kwh/st	24.3	13.2	35	28

ROD MILLING - OPERATING PLANT DATA

Plant	White Pine	White Pine	Mattagami
Reference	Bassarear	Bassarear	Bassarear
Date of Reference or Data			
S.T.P.H. (ea. mill)	130	278	168
Size (dia. x length, ft.)	10 1/2 x 14	13 x 21	11 x 13
H.P.: Motor rating	700	1500	1000
Draw			
% Solids (S.G. Ore)	79	80	83
Feed: % + 3/4"/top size			
20% retained (um)	14,000		15,000
Other			
Product: Top size (mesh)			
20% retained (um)	18,000		900
Other			
Rod Size	3"	3"	3 1/2"
Speed: r.p.m.	16	13	15.8
% Cs	67	60	67
Recirculation			
Reduction Ratio	7.8		17
Power: kwh/t	4	4	3.5/
Rod Consump.: lb/t	.32	.32	0.69
lb/kwh	.1	.1	0.20
Operating W.I.: kwh/st	26		14

ROD MILLING - OPERATING PLANT DATA

Plant	-	-	-
Reference	Bond	Bond	Bond
Date of Reference or Data	1956	1956	1956
S.T.P.H. (ea. mill)	185.4	112	120.6
Size (dia. x length, ft.)	10 1/2 x 16	10 1/2 x 12	9 x 12
H.P.: Motor rating	800	800	450
Draw	654	768	440
% Solids (S.G. Ore)	83/3.5	80/3.5	55/3.0
Feed: % + 3/4"/top size			
20% retained (um)	14,000	16,000	14,500
Other			
Product: Top size (mesh)			
20% retained (um)	1620	1400	1120
Other			
Rod Size	4"	4"	3"
Speed: r.p.m.	14.7	18	18.1
% Cs	61	74	69
Recirculation	o.c.	o.c.	o.c.
Reduction Ratio	8.65	11.4	13
Power: kwh/t	2.63	5.12	2.72
Rod Consump.: lb/t	1.03	1.35	0.22
lb/kwh	0.36	0.26	0.08
Operating W.I.: kwh/st	16.1	27	12.6

ROD MILLING - OPERATING PLANT DATA

Plant	-	-	-
Reference	Bond	Bond	Bond
Date of Reference or Data	1956	1956	1956
S.T.P.H. (ea. mill)	167.5	92.7	146.3
Size (dia. x length, ft.)	10 x 13	9 x 14 1/2	9 1/2 x 12
H.P.: Motor rating	700	450	600
Draw	557	490	575
% Solids (S.G. Ore)	75/2.6	/3.5	80/3.5
Feed: % + 3/4"/top size			
20% retained (um)	24,000	15,600	15,000
Other			
Product: Top size (mesh)			
20% retained (um)	1160	1800	640
Other			
Rod Size	3 1/2	3 1/2"	4"
Speed: r.p.m.	15.8		16.8
% Cs	64.8	67	65.8
Recirculation	o.c.	o.c.	o.c.
Reduction Ratio	13	8.7	23.4
Power: kwh/t	2.48	3.94	2.93
Rod Consump.: lb/t		1.12	
lb/kwh		0.28	
Operating W.I.: kwh/st	10.83	25.3	9.35

ROD MILLING - OPERATING PLANT DATA

Plant	Frood-Stobie			
Reference	Zickar et al			
Date of Reference or Data	1981			
S.T.P.H. ea.mill)	300	350	300-375	300-375
Size (dia. x length, ft.)	13 1/2 x 18	same		
H.P.: Motor rating	1750	same		
Draw	1675	1700		
% Solids (S.G. Ore)	80%			
Feed: % + 3/4"/top size				
20% retained (um)	17,500	18,000		
Other				
Product: Top size (mesh)				
20% retained (um)	1150	2300		
Other				
Rod Size	4"	4"		
Speed: r.p.m.				
% Cs	66%	66%		
Recirculation	o.c.	o.c.		
Reduction Ratio	15.2	7.8		
Power: kwh/t	4.17	3.63		
Rod Consump.: lb/t	0.86			
lb/kwh	.21			
Operating W.I.: kwh/st	19.0	27.1	27.85	23.61

APPENDIX D

ALLIS-CHALMERS BOND WORK INDEX TEST PROCEDURES

AND

SUPPLEMENTARY NOTES ON PROCEDURES USED AT MCGILL UNIVERSITY

Bond Rod Mill Work Index Test

Laboratory Procedure

<u>Rod Mill Grindability Test</u>

The feed is crushed to -1/2 in., and 1250 cc packed in a graduated cylinder are weighed, screen analysed, and ground dry in closed circuit with 100 per cent circulating load.

The mill is a 12 in. dia. by 24 in. long tilting rod mill with a wave-type lining and revolution counter, running a 46 rpm.

The grinding charge consists of six 1.25 in. dia. and two 1.75 in. dia. steel rods 21 in. long and weighing 33,380 grams.

In order to equalize segregation at the mill ends, it is rotated level for eight revolutions, than tilted up 5° for one revolution, down 5° for another revolution, and returned to level for eight revolutions continuously throughout each grinding period.

Tests can be made at all mesh sizes from 4 to 65 mesh (normally 8 to 28 mesh). At the end of each grinding period the mill is discharged by tilting downward at 45° for 30 revolutions, and the product is screened on sieves of the mesh size tested. The sieve undersize is weighed, and fresh unsegregated feed is added to the oversize to make its total weight equal to that of the 1250 cc originally charged into the mill. This is returned to the mill and ground for the number of revolutions calculated to give a circulating load equal to the weight of the new feed added. The grinding period cycles are continued until the net grams of sieve undersize produced per revolution reaches equilibrium and reverses its direction of increase or decrease. Then the undersize product and circulating load are screen anaylsed, and the average of the last three net grams per revolution (Grp) is the rod mill grindability.

Where F is the size in microns which 80 per cent of the new rod mill feed passes, and P_1 is the opening of the sieve size tested in microns, then the rod mill work index Wi is calculated from the following revised (1960) equation:

$$Wi = 62 \Big/ (P_1)^{0.23} \times (Grp)^{0.625} \left(\frac{10}{\sqrt{P}} \quad \frac{10}{\sqrt{F}} \right)$$

This Wi value should conform with the motor output power to an average overflow rod mill of 8 ft. interior diameter grinding wet in open circuit.

For dry grinding the work input should be multiplied by 1.30.

The accompanying table and procedure can be used to shorten the calculations required with the new equation above. The values of $62/P_1^{0.23}$ are listed for each mesh size, and the values of $Grp^{0.625}$ can be found by interpolation from the table. The average P values are given as a check on the value found by plotting the screen analysis of the test product.

BOND ROD MILL GRINDABILITY LABORATORY PROCEDURE

1. Prepare sample to -1/2" by stage crushing and screening
2. Determine Screen Anaylsis
3. Determine Bulk Density, Lbs./Ft^3
4. Calculate weight of Material Charge

$$\text{Material Charge (gms)} = \frac{\text{Bulk Density (Lbs/Ft}^3)}{62.4 \text{ Lbs./Ft}^3} \times 1250 \text{ cc}$$

$$\text{Material Charge} = \frac{\text{Bulk Wt. (gm/lit.)}}{1000} \times 1250 \text{ cc}$$

5. Calculate IPP (Ideal Potential Product) for 100% Circulating Load

$$\text{IPP (grams)} = \frac{\text{Wt. Mill Charge (gms)}}{2.0}$$

6. Split feed sample to obtain 8 to 12 samples slightly smaller than IPP. Also split out sample for Material Charge.
7. Place Material Charge and Rod Charge in Mill Run x revolutions

   x = number of revolutions based on estimate of work index; usually 50, 100, 150 or 200 revolutions

8. Dump Material Charge. Screen all the material at Mesh of Grind - Weigh Product.
9. Product = weight of Material Charge - weight of Screen Oversize.
10. Net Product = Product - weight of undersize in Mill Feed.
11. Net grams of product per revolution = $\frac{\text{Net Product}}{\text{No. of Rev.}}$
12. Add new feed to Screen Oversize (circulating load) to bring up to original weight of Material Charge using split samples.
13. Calculate No. of Revolutions for next period.

$$\text{No. Rev.} = \frac{\text{IPP (gms) - Wt. undersize in feed}}{\text{Net grams/rev. for previous period}}$$

14. Repeat Steps 8-14 until net gms/rev. comes into equilibrium (May be steady or jumping, minimum of 5 periods)

$$\text{Circulating Load} = \frac{\text{Material Charge} - \frac{\text{(Last 2-3 periods)}}{\text{Average Product}}}{\frac{\text{Average Product}}{\text{(Last 2-3 periods)}}}$$

A-C ROD MILL GRINDABILITY TABLE

$$Wi = 62 \Big/ (P_1)^{0.23} \times (Grp)^{0.625} \times \left(\frac{10}{\sqrt{P}} \quad \frac{10}{\sqrt{F}}\right)$$

Grp	Grp.625	Grp	Grp.625	Grp	Gbp.625	Mesh Test	$62/P_1^{0.23}$	Average P
1	1.000	12.0	4.730	20.0	6.50			
2	1.542	12.2	4.775	20.2	6.54	Feed	-	10,000
3	1.988	12.4	4.820	20.4	6.58			
3.5	2.189	12.6	4.870	20.6	6.62	3	8.18	5100
4	2.379	12.8	4.915	20.8	6.66			
4.5	2.560	13.0	4.965	21.0	6.71	4	8.85	3640
5.0	2.739	13.2	5.015	21.2	6.75			
5.2	2.803	13.4	5.060	21.4	6.79	6	9.60	2560
5.4	2.866	13.6	5.110	21.6	6.83			
5.6	2.936	13.8	5.155	21.8	6.87	8	10.38	1815
5.8	3.000	14.0	5.210	22.0	6.90			
6.0	3.065	14.2	5.255	22.2	6.94	10	11.25	1280
6.2	3.127	14.4	5.300	22.4	6.98			
6.4	3.190	14.6	5.340	22.6	7.02	14	12.20	908
6.6	3.253	14.8	5.380	22.8	7.06			
6.8	3.315	15.0	5.430	23.0	7.10	20	13.20	640
7.0	3.376	15.2	5.48	23.2	7.14			
7.2	3.433	15.4	5.52	23.4	7.18	28	14.30	450
7.4	3.492	15.6	5.57	23.6	7.22			
7.6	3.550	15.8	5.62	23.8	7.26	35	15.45	320
7.8	3.613	16.0	5.66	24.0	7.29			
8.0	3.670	16.2	5.70	24.2	7.33	48	16.70	226
8.2	3.724	16.4	5.74	24.4	7.37			
8.4	3.780	16.6	5.79	24.6	7.[illegible]1	65	18.11	160
8.6	3.837	16.8	5.83	24.8	7.45			
8.8	3.890	17.0	5.87	25.0	7.48			
9.0	3.945	17.2	5.91	25.5	7.57			
9.2	4.000	17.4	5.95	26.0	7.66			
9.4	4.060	17.6	6.00	26.5	7.75			
9.6	4.110	17.8	6.04	27.0	7.84			
9.8	4.166	18.0	6.09	27.5	7.94			
10.0	4.215	18.2	6.13	28.0	8.03			
10.2	4.260	18.4	6.17	28.5	8.12			
10.4	4.315	18.6	6.22	29.0	8.21			
10.6	4.370	18.8	6.26	29.5	8.29			
10.8	4.420	19.0	6.30	30.0	8.37			
11.0	4.470	19.2	6.34	35.0	9.23			
11.2	4.520	19.4	6.38	40.0	10.01			
11.4	4.570	19.6	6.42	50.0	11.52			
11.6	4.630	19.8	6.46	60.0	12.94			
11.8	4.680	20.0	6.50	70.0	14.[illegible]			

Supplementary Notes on Bond Rod Mill Work Index Test Procedures used at McGill University

The test procedure used was intended to replicate as closely as possible that developed by Allis-Chalmers at their test facilities in Oak Creek, Wisconsin. The Bond rod mill test apparatus, constructed by the Colorado School of Mines Research Institute to F.C. Bond's specifications, was purchased for this purpose. The Allis-Chalmers test facilities were visited and the staff there consulted frequently to ensure compliance with established procedures. The details of the procedures used at McGill University are outlined as follows.

1. Oven dry the complete sample (approx. 15 to 20 kg).

2. Scalp off all the plus 12.7 mm (0.50 inch) material using the square Gilson screen deck. Crush this material in stages using the jaw crusher until it all passes the screen.

3. Weigh this material. Perform dry screen analysis to 425 um (35 mesh), after riffling (if necessary) to approximately 1 kg, using a 20 minute screening time.

4. From the above screen analysis, and the previously determined screen analysis of the plant rod mill feed, calculate the combined test material feed size distribution.

5. Recombine complete sample, and split on the 12 way spinning riffle. Riffle one of these samples to about 1 kg, and perform dry screen analysis for 20 minutes to 425 um (35 mesh) to check the feed screen size distribution, as calculated in step 4. Repeat on a second and third sub-sample, if necessary.

6. Determine the material specific gravity, if not previously known, using one of the sub-samples. This can be done using a large graduated cylinder, water, and an accurate balance.

7. The mill charge volume of 1250 cc of bulk volume can be obtained by packing material into a graduated cylinder. Alternatively, the weight of the mill solids charge can be calculated using the solids specific gravity and assuming a 67 percent volumetric packing density, which was measured in some of the earlier tests. Note that it was also determined that the test results are not sensitive to the exact weight of the solids charge.

8. Calculate the "ideal potential product" to achieve a circulating load of 100% (one-half of the mill load).

9. For period one, grind for 100 revolutions. Remove the mill solids, and screen from 9500 um (0.371 inch mesh) to the test control size, dry, for 15 minutes on the sieve shaker.

10. For each period, replace the undersize removed by new feed to make up the initial mill charge. The new feed should be riffled or otherwise split as carefully as possible to avoid segretation. Note that after reaching equilibrium, a circulating load of 100 plus or minus 2% should be achieved.

11. Dry screen the undersize from the last run down to 425 um (35 mesh) for 20 minutes. Dry screen the oversize from 9500 um (0.371 inch mesh) to the test control size for 15 minutes.

12. Determine the 80% passing feed and product sizes by log-log interpolation for calculation of the work index.

Revised on: May 17, 1983

BOND BALL MILL WORK INDEX TEST LABORATORY PROCEDURE

Ball Mill Grindability Test

The standard feed is prepared by stage crushing to all passing a 6 mesh sieve, finer feed can used only when necessary. It is screen analysed and packed bulk weight in grams per liter obtained by shaking and tapping. The weight of 700 cc is placed in the mill and ground dry at 250 per cent circulating load.

The mill is 12 in. x 12 in. with rounded corners, and a smooth lining except for a 4 in. x 8 in. hand hole door for charging. It has a revolution counter and runs at 70 rpm.

The grinding charge consists of 285 iron balls weighing 20,125 grams. It consists of about 43-1.45-in. balls, 67-1.17-in. balls, 10-1-in. balls, 71-0.75-in. balls, 94-0.61-in. balls with a calculated surface area of 842 sq. in.

Tests are made at all sieve sizes below 28 mesh. After the first grinding period (usually of 100 revolutions) the mill is dumped, the ball charge is screened out, and the 700 cc of material is screened on sieves of the mesh size tested, with coarser protecting sieves if necessary. The undersize is weighed, and fresh unsegregated feed is added to the oversize to bring its weight back to that of the original charge. Then it is returned on to the balls in the mill and ground for the number of revolutions calculated to produce a 250 per cent circulating load, dumped and rescreened. The number of revolutions required is calculated from the results of the previous period to produce sieve undersize equal to 1/3.5 of the total charge in the mill.

The grinding period cycles are continued until the net grams of sieve undersize produced per mill revolution reaches equilibrium and reverses its direction of increase or decrease. Then the undersize product and circulating load are screen analysed, and the average of the last three net grams per revolution (Gbp) is the ball mill grindability.

When F is the size in microns which 80 per cent of the new ball mill feed passes, P is the microns which 80 per cent of the last cycle sieve undersize product passes, and P_1 is the opening in microns of the sieve size tested, then the ball mill work index Wi is calculated from the following revised (1960) equation:

$$Wi = 44.5 \Big/ \left[(P_1)^{0.23} \times (Gbp)^{0.82} \left(\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}} \right) \right]$$

The average value of P at 100 mesh is 114 microns, at 150 mesh it is 76 microns, at 200 mesh it is 50, and at 325 mesh it is 26.7. These values of P are to be used in this equation when P cannot be found from size distribution analyses.

The Wi value from this equation should conform with the motor output power to and average overflow ball mill of 8 ft. interior diameter grinding wet in closed circuit.

For dry grinding the work input should normally be multiplied by 1.30. However, ball coating and packing can increase the work input in dry grinding.

Revised on: May 17, 1983

BOND BALL MILL GRINDABILITY LABORATORY PROCEDURE

1. Prepare sample to -6 mesh by stage crushing and screening.

2. Determine Screen Analysis (% passing the sieve size tested).

3. Determine Packed Bulk Density Lbs/Ft^3 or g/ℓ

4. Calculate weight of material charge

$$\text{Material Charge (gms)} = \frac{\text{Packed Bulk Wt. } (Lbs/Ft^3)}{62.4 \ Lbs/Ft^3} \times 700 \text{ cc}$$

$$\text{Material Charge (g)} = \frac{\text{Packed Bulk Wt. } (g/\ell)}{1000} \times 700 \text{ cc}$$

5. Calculate IPP (Ideal Potential Product) for 250% Circulating Load

$$\text{IPP (grams)} = \frac{\text{Wt. Material Charge (gms)}}{3.5}$$

6. Split feed sample to obtain 8 to 12 samples slightly smaller than IPP. Also split out sample for Material Charge.

7. Place Material Charge and Ball Charge in Mill Run x revolutions

   x = number of revolutions based on estimate of work index; usually 50, 100, 150 or 200 revolutions

8. Dump Mill, separate balls and Material Charge. Screen all the material at Mesh of Grind - Weigh Product.

9. Product = weight of Material Charge - weight of Screen Oversize.

10. Net Product = Product - weight of undersize in Mill Feed

11. Net grams of product per revolution = $\frac{\text{Net Product}}{\text{No. of Rev.}}$

12. Add new feed to Screen Oversize (circulating load) to bring up to original weight of Material Charge using split samples.

13. Calculate No. of Revolutions for next period.

$$\text{No. Rev.} = \frac{\text{IPP (gms) - Wt. undersize in feed}}{\text{Net grams/rev. for previous period}}$$

14. Repeat steps 8-14 until net gms/rev. comes into equilibrium (May be steady or jumping, minimum of 5 periods)

$$\text{Circulating Load} = \frac{\text{Material Charge} - \dfrac{\text{(Last 2-3 periods)}}{\text{Average Product}}}{\text{Average Product (Last 2-3 periods)}} \times 100$$

Supplementary Notes on Bond Ball Mill Work Index Test Procedures used at McGill University

The test procedure was intended to replicate as closely as possible that developed by Allis-Chalmers at their test facilities in Oak Creek, Wisconsin. The Bond ball mill grinding apparatus was supplied by Bico-Braun, but the ball charge had to be replaced by one as specified by Allis-Chalmers. Once again, the Allis-Chalmers staff was frequently consulted. Details on the procedure used at McGill University are described below.

1. Oven dry the complete sample (approx. 10 kg), and roll to remove any lumps.

2. Scalp off the plus 3350 um (6 mesh) material using the square Gilson screen deck. Crush this material in the gyratory crusher (or by hand, using a roller, if the weight is very small) until it all passes the screen.

3. If the weight of this material is significant (over 0.5% of the total), perform dry screen analysis for 20 minutes so the percentage passing the control screen size in the test feed material can be calculated, knowing the screen analysis of the plant rod mill discharge.

4. Recombine the sample, and split on the 12 way spinning riffle. Using one of these samples (approx. 1 kg), perform a dry screen analysis for 30 minutes from 2360 um (8 mesh) to 425 um (35 mesh). Riffle the undersize to about 200 grams. Wet screen and then dry screen for 20 minutes on 150 um (100 mesh), the test control size to determine the percentage of product size material in the test feed. Check this against the calculated value for step 3. Repeat if necessary.

5. Determine the weight of 700 cc by packing in a graduated cylinder. Alternatively, the mill solids weight can be calculated from the solids specific gravity, and assuming a 66 percent volumetric packing density. Note that it was determined that the test results are not sensitive to the exact weight of the mill charge.

6. Calculate the "ideal potential product" to achieve a circulating load of 250%.

7. After period 1, remove the mill contents and divide it into 8 batches for dry screening on 150 um (100 mesh) for 20 minutes each. This can be done quickly using a number of screens and extended lip pans.

8. For each period, replace the undersize removed by new feed to make up the initial mill charge weight. The new feed should be riffled or otherwise split as carefully as possible to avoid segregation.

9. Dry screen the undersize from the last run down to 2 mesh sizes finer than the control size for 30 minutes after riffling to about 100 gms. Dry screen the oversize from 2360 um (8 mesh) to the control size of 150 um (100 mesh) for 20 minutes after riffling down to about 200 gms.

10. Determine the 80% passing feed and product sizes by log-log interpolation for calculation of the work index.

APPENDIX E

BOND ROD MILL GRINDABILITY TEST DATA FROM SELBAIE SURVEY NO. 2

Rod Mill Work Index Test Results

Selbaie Survey No. 2

Test Feed: 18 kg of rod mill feed from Les Mines Selbaie Ltee, 95% passing 1.7 cm, crushed to 100% passing 1.2 cm.

Cycle mass: 1.25 L of ore = 2337 g

Fresh Feed Size Distribution

Size (um)	% Finer	% Retained
12500	100.00	0.00
9500	75.54	24.46
6300	54.86	20.68
4750	42.15	12.71
3350	34.16	7.99
2360	28.15	6.01
1700	24.13	4.02
1180	20.22	3.91
850	17.89	2.33
600	15.67	2.22
425	13.65	2.01

The mass of rods was equal to 32.9 kg

After each cycle, the mill discharge was screened down to 1.70 mm for 15 minutes. The undersize was removed, and replaced by an equal amount of fresh feed. A summary of the test results is given in the following table.

Cycle	Revs.	Mass of minus 1.7 mm (g) Feed	Product	Produced	Grp (g)
1	100	1524	564	960	9.60
2	83	1267	371	896	10.80
3	79	1233	307	926	11.72
4	74	1263	298	965	13.04
5	71	1146	305	841	11.85
6	75	1260	278	982	13.09
7	66	1186	304	882	13.36
8	Run discounted - rev. counter malfunction				
9	67	1161	268	893	13.32
10	67	1158	280	878	13.10

Average circulating for the last three cycles:
(2337 g - 1168 g) * 100% / 1168 g = 100%

Grp (last three cycles): 13.26 g

The minus 1.70 mm of the last cycle (1149 g) was dry screened for 20 minutes, yielding the following size distribution:

Size (um)	% Finer	% Retained
1180	73.26	26.74
850	58.91	14.35
600	47.44	11.47
425	38.45	8.99
-425	-	38.45

The plus 1.70 mm (1179 g) of the last cycle was dry screened for 15 minutes, yielding the following size distribution:

Size (um)	% Finer	% Retained
9500	97.68	2.32
6700	93.83	3.85
4750	87.18	6.65
3350	72.87	14.31
2360	42.28	30.59
1700	-	42.28

APPENDIX F

BOND WORK INDEX TEST RESULTS

ON SELBAIE AND KIDD CREEK ORE SAMPLES

Table F-1. Bond Rod Mill Work Index Tests on Selbaie Ore

(1700 um)

Test No.	Survey/ Sample	Test Date	Specific Gravity	Mill Load (gm)	Test Feed Size (um) F80	Test Feed Size (um) K50	Product (um) P80	Grindability (net gm/rev.)	W.I. (kwh/mt)
1	No.1	Oct.11/86	2.83	2200	10,513	5,760	1,307	12.0	14.6
2	No.1	Oct.13/86	2.83	2200	9,705	4,842	1,315	12.3	14.8
3	No.1	Oct.15/86	2.83	2200	8,105	4,030	1,315	12.6	15.4
4	No.1	Oct.15/86	2.83	2200	5,338	2,146	1,319	13.0	18.0
5	No.2	Apr.19/86	2.83	2337	10,161	5,936	1,308	13.3	13.8
6	No.4A	Sept.16/86	2.73	2205	10,015	5,842	1,330	12.0	15.0
7	No.4A	Sept.17/86	2.73	2337	10,015	5,842	1,326	11.9	15.0
8	No.4A	Oct.3/86	2.73	2337	5,465	2,625	1,319	12.2	18.5
9	No.4B1	Oct.1/86	2.69	2148	9,842	5,475	1,347	11.8	15.4
10	No.4B2	Sept.18/86	2.70	2337	9,471	5,219	1,350	11.8	15.5
11	No.4C	Sept.19/86	2.76	2160	9,675	5,021	1,330	12.4	14.9
12	No.4C	Sept.19/86	2.76	2400	9,675	5,021	1,332	12.4	14.9
13	No.8A	Aug.17/87	2.82	2400	9,407	4,972	1,286	12.0	14.9
14	No.8B	Aug.20/87	2.82	2400	9,254	5,217	1,297	11.0	15.9
15	No.9	July 6/87	2.66	2300	10,054	5,150	1,319	10.3	16.4

Table F-2. Bond Rod Mill Work Index Tests on Kidd Creek Ore (1700 um)

Test No.	Survey/ Sample	Test Date	Specific Gravity	Mill Load (gm)	Test Feed Size (um) F80	Test Feed Size (um) K50	Product (um) P80	Grindability (net gm/rev.)	W.I. (kwh/mt)
1	No.1	Aug. 14/87	3.3	2280	10,526	5,626	1,280	8.73	17.4
2	No.2	July 9/87	3.0	2250	9,987	5,378	1,240	8.34	17.8
3	No.2	July 12/87	3.0	2250	8,923	4,842	1,230	8.67	17.9
4	No.2	July 15/87	3.0	2250	7,945	4,086	1,302	8.68	19.4
5	No.3A	Sept. 17/87	3.06	2330	9,586	4,895	1,297	8.32	18.7
6	No.3B	Sept. 20/87	3.07	2500	9,586	4,788	1,294	9.08	17.7
7	No.4D	Jan. 12/87	-	2394	10,010	4,874	1,289	9.29	17.1
8	No.4E	Jan. 15/87	-	2394	9,760	4,640	1,288	9.35	17.2

Table F-3. Bond Ball Mill Work Index Tests on Selbaie Ore (150 um)

Test No.	Sample/ Survey	Test Date	Specific Gravity	Mill Load (gm)	Test Feed Size (um) F80	Product (um) P80	Grindability (net gm/rev)	W.I.
1	No. 1	July 24/87	2.83	1310	1130	108	2.08	12.8
2	No. 1	July 26/87	2.83	1180	1130	111	2.07	13.1
3	No. 1	July 27/87	2.83	1180	610	109	2.01	15.8
4	No. 2	July 28/87	2.83	1290	1132	109	2.31	11.8
5	No. 2	July 30/87	2.83	1290	818	109	2.32	12.8

Table F-4. Bond Ball Mill Work Index Tests on Kidd Creek Ore

A. At 150 um (100 mesh)

Test No.	Sample/ Survey	Test Date	Mill Load (gm)	Test Feed Size (um) F80	Product (um) P80	Grindability (net gm/rev)	W.I.
1A	No. 1	Aug. 1/87	1492	1361	112	2.07	12.7
2A	No. 2	Aug. 5/87	1478	1492	112	1.86	13.6

B. At 75 um (200 mesh)

Test No.	Sample/ Survey	Test Date	Mill Load (gm)	Test Feed Size (um) F80	Product (um) P80	Grindability (net gm/rev)	W.I.
1B	No. 1	Aug. 3/87	1492	1361	58	1.39	13.3*
2B	No. 2	Aug. 7/87	1478	1492	61	1.45	13.1

* Approximate (circulating load not yet stabilized when test stopped).

APPENDIX G

ROD MILL FINES ADDITION TESTS

(SURVEYS NO. 3 AND 4) AT LES MINES SELBAIE

Rod Mill Fines Addition Tests (Surveys No. 3 and 4) at Les Mines Selbaie

Preparatory Testwork

A set of preliminary tests were first carried out on May 27, 1986, to ensure the adequacy of sampling equipment and procedures. Rod mill fines addition was provided by tapping into the cyclone feed line, and running a hose to the rod mill feed chute. These tests were also used to determine the valve settings needed to achieve the required flow rates of water and fines.

A full sampling survey (no. 3) was carried out on May 29 under (a) normal conditions; (b-1) with cyclone feed fines added to the rod mill feed and normal water rate addition; (b-2) with cyclone feed fines and reduced water addition; and (c) normal conditions once again. Unfortunately, some problems were experienced with preparation of the samples in the laboratory, and screen analysis results could not be reproduced. This was therefore considered as a practice run only, and a complete repeat test was planned. The only clear observation from this test was a reduction in power draw of 6 to 7% when the fines were added versus normal rod mill operation.

The subsequent test, sample survey no. 4, was planned to reproduce the previous test, with the following modifications noted:

1. During the fines addition portion of the test, an adequate flow of fines would be used to ensure the needed level of total fines in the rod mill feed. This also would permit fines addition with the rod mill feed water off, and therefore a more accurate mass flow calculation.
2. During the extra dilution water portion of the test, the water flow rate should be substantially increased to try to create a more pronounced effect on the mill.
3. The rod size being charged to the mill was changed from 89 to 76 mm (3.5 to 3 in.) as of June, 1986 due to concerns about the rod mill foundation.

Test Description - Survey No. 4

Operating Modes

A: Normal operation (water only added to rod mill feed).

B1: Extra dilution water added to rod mill feed.

B2: Cyclone feed added (valve fully open less 6 revolutions) to rod mill feed and rod mill feed water off.

C: Normal operation.

Data Collection

For each of the 4 operating modes, the rod mill feed tonnage was determined by recording the time (to the nearest second) and the rod mill feed totalizer reading at the beginning and end of each period, as follows:

Mode	Time Start / Finish	Tonnage Start / Finish	Tonnage Rate (Wet m.t.p.h.)
A	9:46:00 / 10:14:00	22662.60 / 22691.49	61.9
B1	10:26:00 / 10:53:00	22703.92 / 22731.94	62.3
B2	13:34:00 / 13:55:00	22894.83 / 22914.92	57.4
C	14:05:00 / 14:23:00	22924.50 / 22942.28	59.3

During each period, samples were cut and combined at each of the streams as follows:

Rod mill feed:	4 x 2 cuts per sample.
Rod mill discharge:	4 x 2 cuts per sample.
Cyclone feed fines:	4 x 3 cuts per sample.

During each period, a series of "high-low" ammeter readings, each over a period of 3 to 4 seconds, was taken to obtain an average for the test condition, as follows:

	Mode A, 9:46	Mode A, 10:14	Mode B1, 10:26	Mode B2, 13:34
	33-38 amps	33-38	35-40	32-38
	33-39	31-40	33-38	31-36
	32-28	34-38	34-40	33-36
	32-39	32-37	34-39	32-37
	33-38	33-37	34-39	32-37
	31-39	33-37	33-38	31-37
	33-37			31-35
	35-39			30-37
Ave:	35.6	35.3	36.4	34.1

	Mode B2, 13:55	Mode C, 14:05	Mode C, 14:23
	31-34	35-40	35-39
	32-37	36-40	34-39
	33-39	35-39	33-36
	32-36	35-39	31-37
	30-37	35-39	34-37
	30-37	35-40	33-40
	32-38	33-41	34-38
	32-35		
Ave:	34.1	37.3	35.7

At the end of the test, the power plant was visited to record the line voltage (V=4060) and power frequency (60.0 c.p.s.).

Sample Analyses

The percent solids determinations (by weight) on all the collected samples are summarized below:

Table G-1. Percent Solids of Sampled Streams, Survey No. 4

	Mode A	B1	B2	C
Rod mill feed	93.9%	93.4	94.8	94.9
Rod mill discharge	80.8%	71.2	82.1	80.4
Cyclone feed (fines)	-	-	64.9	-

Bond rod mill work index tests were carried out at McGill University on rod mill feed samples from each operating mode, as follows. The control screen size was 1700um (10 mesh) throughout.

Table G-2. Bond Rod Mill Work Indexes, Survey No. 4

Mode	Specific Gravity	Test Feed K50(um)	Net Grams per Revolution	W.I. (kwh/mt)
A	2.73	5,842	12.0	15.0
B1	2.69	5,475	11.8	15.4
B2	2.70	5,219	12.4	15.5
C	2.76	5,021	12.0	14.8

The size distribution data from the test are given in the following Tables B7-3 to 5.

Table G-3. Rod Mill Feed Size Distributions, Survey No. 4

Screen Size (um)	Mode A Ind.%	Mode A Cum.% Pass.	Mode B-1 Ind.%	Mode B-1 Cum.% Pass.	Mode B-2 Ind.%	Mode B-2 Cum.% Pass.	Mode B-3 Ind.%	Mode B-3 Cum.% Pass.
26900	0.00	100.0	0.00	100.0	0.00	100.0	0.22	99.78
19000	2.09	97.91	2.39	97.61	1.62	98.38	1.64	98.14
13500	13.81	84.10	12.66	84.95	13.72	84.66	15.28	82.86
9500	20.64	63.46	20.40	64.55	21.32	63.33	20.78	62.08
6700	15.14	48.32	13.08	51.47	15.31	48.02	14.93	47.14
4750	9.71	38.60	12.00	39.48	10.82	37.21	9.61	37.54
3350	8.31	30.29	8.42	31.06	8.22	28.99	8.28	29.25
2360	5.77	24.53	6.26	24.81	6.08	22.91	5.72	23.53
1700	4.05	20.48	4.23	20.58	4.00	18.91	3.78	19.75
1180	3.44	17.04	3.49	17.09	3.23	15.68	3.18	16.57
850	2.38	14.66	2.39	14.69	2.18	13.50	2.17	14.40
600	2.00	12.67	2.06	12.64	1.83	11.67	1.87	12.53
425	1.76	10.90	1.75	10.89	1.64	10.03	1.70	10.83
300	1.25	9.66	1.25	9.64	1.22	8.81	3.73	7.10
212	0.84	8.81	0.82	8.82	0.84	7.97	0.68	6.42
150	0.97	7.85	0.94	7.88	0.97	7.00	0.78	5.64
106	0.80	7.05	0.79	7.10	0.78	6.23	0.64	5.00
75	0.30	6.75	0.31	6.79	0.68	5.55	0.57	4.43
53	1.02	5.74	1.02	5.77	0.55	5.00	0.43	4.00
38	0.86	4.88	0.93	4.85	0.79	4.21	0.65	3.36
- 38	4.88	0.00	4.85	0.00	4.21	0.00	3.36	0.00
Total	100.0		100.0		100.0		100.0	

Table G-4. Rod Mill Discharge Size Distributions, Survey No. 4

Screen Size (um)	Mode A Ind.%	Mode A Cum.% Pass.	Mode B-1 Ind.%	Mode B-1 Cum.% Pass.	Mode B-2 Ind.%	Mode B-2 Cum.% Pass.	Mode B-3 Ind.%	Mode B-3 Cum.% Pass.
3350	0.00	100.0	0.00	100.0	0.00	100.0	0.00	100.0
2360	2.46	97.54	1.22	98.78	1.88	98.12	1.25	98.75
1700	5.21	92.34	3.21	95.57	4.23	93.89	3.50	95.25
1180	10.90	81.43	8.04	87.53	9.64	84.25	8.35	86.91
850	11.39	70.04	10.57	76.96	10.03	74.21	10.41	76.50
600	11.00	59.04	11.74	65.22	9.86	64.35	11.27	65.23
425	9.82	49.21	10.96	54.26	9.08	55.28	10.46	54.77
300	7.12	42.09	8.18	46.08	7.24	48.04	8.15	46.62
212	4.91	37.18	5.55	40.53	5.54	42.50	5.23	41.39
150	5.06	32.12	5.60	34.92	6.37	36.13	5.61	35.78
106	3.98	28.14	4.19	30.74	4.97	31.15	4.32	31.46
75	3.34	24.80	3.46	27.28	4.06	27.09	3.65	27.82
53	2.41	22.40	2.63	24.65	2.97	24.13	2.83	24.99
38	3.24	19.16	3.36	21.29	3.75	20.38	3.45	21.53
- 38	19.16		21.19		20.38		21.53	
Total	100.0		100.0		100.0		100.0	

Table G-5. Cyclone Feed Fines Size Distributions, Survey No. 4

Screen Size (um)	Mode B-2 Ind.%	Mode B-2 Cum.% Pass.
3350	0.00	100.0
2360	1.34	98.66
1700	2.26	96.40
1180	4.78	91.62
850	4.78	86.84
600	5.60	81.23
425	7.15	74.09
300	8.48	65.60
212	8.74	56.86
150	11.77	45.09
106	9.15	35.94
75	7.10	28.84
53	4.27	24.58
38	4.94	19.64
- 38	19.64	
Total	100.0	

APPENDIX H

ROD MILL FINES ADDITION TEST (SURVEY NO. 3) AT KIDD CREEK MINES

Rod Mill Fines Addition Test (Survey No. 3) at Kidd Creek Mines

Test Description

This plant test on August 13, 1986, was carried out in parallel to other similar testwork at Les Mines Selbaie (see Appendixes A6 and A7). Rod mill feed fines were obtained from the primary cyclone feed by connecting into one of the outlets from the cyclone distributor. Preliminary tests were also carried out to determine needed valve settings to obtain desired approximate flow rates of fines.

Operating Modes

A: Normal operation.

B: Cyclone feed fines added to rod mill feed, and rod mill feed dilution water cut back to achieve near the same rod mill discharge density as normal operation.

C: Extra dilution water added to rod mill feed (no fines addition).

Data Collection

For each of the three operating modes, the rod mill feed tonnage was determined by recording the time and the rod mill feed totalizer reading at the beginning and end of each period, as follows.

Mode	Time Start/Finish	Tonnage Start/Finish	Tonnage Rate (Wet M.T.P.H.)
A	9:42:00/10:59:00	771518/771684	129.4
	12:42:00	771907	129.9
B	13:51:00	772056	129.6
	14:34:00	772149	129.8
C	15:28:00	772263	126.7

Note that for period C the tonnage appeared to have dropped slightly. However, the rod mill feed stopped momentarily when the electronic eye was re-engaged after the sampling was complete, but just as the final totalizer reading was being taken. Since the tonnage was otherwise very steady, as shown by all the other readings, the tonnage could be taken as constant throughout the test, taken as the average from 9:42 to 14:34, of 129.7 W.M.T.P.H.

During the test, rod mill power draw was monitored from the wattmeter and automatically recorded by the computer. The following readings were also taken in the motor control room during each of the test periods as a check on the calibration of the meters.

Mode	Voltage	Amperage	Power Factor
A	4190	99.5	0.74
B	4175	97	0.73
C	4130	100	0.74

During each period, samples were cut and combined at each of the streams, as follows:

Rod mill feed: 8 cuts for each of 2 samples.

Rod mill discharge: 8 cuts per sample.

Cyclone feed fines: 8 cuts per sample.

Several cuts of cyclone feed fines were also taken at the beginning and end of period B to check for consistency of the cyclone feed fines stream. Duplicate samples of rod mill feed were taken to provide one for percent solids and sizing, and one for work index testing. The rod mill feed water flow rate reading in the control room was also recorded, along with primary cyclone overflow densities to monitor stability of operation. The Marcy density meter was used to set water and cyclone feed fines flow rates near the desired levels by monitoring rod mill discharge density, and comparing to the estimated density based on water/solids mass balance calculations. A summary of the test timing and control room readings is given below.

9:25 a.m.	Initial power draw readings taken. Trommel spray turned off.
9:42	Start A. Cyclone overflow density, 51-54%. Cyclone feed water, 165 (USGPM scale).
10:59	End A. Cyclone overflow density 51-53%. Cyclone feed water, 177.
11:06	Cyclone feed fines added.
11:58	Rod mill feed water adjusted.
12:42 p.m.	Start B. Cyclone overflow density 52%. Cyclone feed water, 200.
1:51	End B. Cyclone overflow density, 51-53%. Cyclone feed water, 200.
2:00	Fines off. Rod mill feed water increased.
2:24	Rod mill feed water adjusted.
2:34	Start C. Cyclone overflow density, 52%. Cyclone feed water, 140.

3:28 End C. Cyclone overflow density, 53%. Cyclone feed water, 125.

Sample Analyses

The wet and dry weights were used to calculate the percent solids of each of the sampled streams, as follows.

Table H-1. Percent Solids of Sampled Streams, Survey No. 3

	A	B	C
Rod mill feed	97.8%	97.7%	97.8%
Rod mill discharge	81.7%	81.4%	76.1%
Cyclone feed (fines)	-	73.7%	-

Screen analyses of each of the above samples are given in Tables B8-2, 3 and 4. Note that check screen analyses were also performed on cyclone feed fines samples taken before and after the test, and these showed excellent stability throughout.

Bond rod mill work index tests were carried out on samples extracted for this purpose for modes A and B. As it was determined that insufficient time was allowed for the circuit to stabilize after adding extra water during mode C, the results for this stage of the test could only be considered approximate, and no work index test was performed on this feed sample.

Table H-2. Rod Mill Feed Size Distributions, Survey No. 3

Screen Size (um)	Mode A Ind.%	Mode A Cum.% Pass.	Mode B Ind.%	Mode B Cum.% Pass.	Mode C Ind.%	Mode C Cum.% Pass.
25,400	2.64	97.36	2.15	97.85	3.50	96.50
19,000	7.74	89.62	7.91	89.94	11.37	85.13
13,200	15.47	74.15	15.43	74.51	15.01	70.12
9,500	15.58	58.57	15.21	59.30	14.52	55.60
6,700	11.01	47.56	10.90	48.40	10.45	45.15
4,750	8.16	39.40	8.32	40.08	7.85	37.30
3,350	6.34	33.06	6.50	33.58	6.01	31.29
2,360	4.43	28.63	4.25	29.33	4.14	27.15
1,700	3.49	25.14	3.79	25.54	3.62	23.53
1,180	3.31	21.83	3.65	21.89	3.31	20.22
850	2.44	19.39	2.61	19.28	2.36	17.86
600	1.81	17.58	1.97	17.31	1.83	16.03
425	1.90	15.68	2.09	15.22	1.91	14.12
300	1.45	14.23	1.34	13.88	1.27	12.85
212	1.40	12.83	1.40	12.48	1.28	11.57
150	1.27	11.56	1.34	11.14	1.18	10.39
106	1.04	10.52	1.16	9.98	1.01	9.38
75	1.04	9.48	1.19	8.79	1.08	8.30
53	0.85	8.63	1.03	7.76	0.97	7.33
38	1.19	7.44	1.20	6.56	1.10	6.23
- 38	7.44		6.56		6.23	
	100.0		100.0		100.0	

Table H-3. Rod Mill Discharge Size Distributions, Survey No. 3

Screen Size (um)	Mode A Ind.%	Mode A Cum.% Pass.	Mode B Ind.%	Mode B Cum.% Pass.	Mode C Ind.%	Mode C Cum.% Pass.
3,350	1.71	98.29	1.15	98.85	1.14	98.86
2,360	3.89	99.40	2.89	95.96	3.39	95.47
1,700	7.81	86.59	6.05	89.91	6.76	88.71
1,180	11.87	74.72	10.44	79.47	11.75	76.96
850	10.55	64.17	9.89	69.58	11.00	65.96
600	8.21	55.96	7.87	61.71	8.75	57.21
425	7.88	48.08	7.81	53.90	8.29	48.92
300	5.63	42.45	5.86	48.04	5.62	43.30
212	5.07	37.38	5.89	42.15	5.27	38.03
150	4.37	33.01	5.54	36.61	4.54	33.49
106	3.39	29.62	4.52	32.09	3.56	29.93
75	3.34	26.28	4.38	27.71	3.59	26.34
53	2.72	23.56	3.73	23.98	2.44	23.90
38	3.28	20.28	3.97	20.01	3.89	20.01
- 38	20.28		20.01		20.01	
	100.0		100.0		100.0	

Table H-4. Cyclone Feed Fines Size Distribution, Survey No. 3

Screen Size (um)	Ind.%	Cum.% Pass.
3,350	0.38	96.62
2,360	1.06	98.56
1,700	2.53	96.03
1,180	4.49	91.54
850	5.07	86.47
600	5.27	81.20
425	7.29	73.91
300	7.36	66.55
212	9.33	57.22
150	10.22	47.00
106	8.65	38.35
75	7.74	30.61
53	5.69	24.92
38	5.18	19.74
- 38	19.74	

APPENDIX I

ROD MILL WATER ADDITION TEST (SURVEY NO. 8)

AT LES MINES SELBIAE

Rod Mill Water Addition Test (Survey No. 8) at Les Mines Selbaie

Test Description

This plant test on November 8, 1986, followed a similar procedure to survey no. 4, except water addition rate was the only variable studied, and the sampling periods and waiting periods between operating modes were extended. Separate rod mill discharge samples were also taken during each half of each period to check for circuit stability.

Operating Modes

A. Normal water addition rate.

B. Increased water addition rate (starting at 1:54 p.m.).

Data Collection

The rod mill feed tonnage was determined by recording the time and the totalizer reading at the beginning and end of each period, as follows:

Mode	Time Start / Finish	Tonnage Start / Finish	Tonnage Rate (WMTPH)
A	11:19:00/12:34:00	29749.50/29838.50	71.2
B	3:03:00/ 4:05:00	30007.99/30080.95	70.5

During each half of each period, 4 samples were cut and combined at the rod mill feed and discharge. Ammeter readings were also taken in the control room and voltage readings in the power plant, as follows:

Amps:	10:55	12:49	2:58	2:59	4:07
	36-40	35-38	37-42	37-40	36-40
	34-43	36-40	36-42	35-41	35-41
	35-41	36-40	37-43	34-41	38-42
	34-43	35-40	36-41	35-42	36-41
	36-40	35-40	36-40	36-40	36-39
	40-42	34-39	36-40	36-41	37-41
	35-40	34-40	36-40	36-43	36-41
	36-41	35-40	37-41	36-42	36-41
	37-42	35-40	37-41	37-41	37-42
	35-40	35-39	35-42	36-41	35-40
Ave:	38.5	37.25	38.75	38.5	38.5
Volts:	11:00	12:54	2:50		4:12
	3930	3980	3980		3980
	3990	4030	4020		4020
	3990	4030	4020		4020

Power frequency was noted to be 60 c.p.s. throughout the day.

Sample Analyses

The percent solids determinations (by weight) are summarized below.

Table I-1. Percent Solids of Sampled Streams, Survey No. 8

	Mode A	Mode B
Rod mill feed	96.0%	95.2%
Rod mill dishcarge - 1	80.6	77.0
Rod mill discharge - 2	82.0	77.2
Rod mill discharge - Ave.	81.4	77.1

Bond rod mill work index tests were carried out at McGill University on the rod mill feed samples, as follows, with a 1700um (10 mesh) control screen size.

Table I-2. Bond Rod Mill Work Indexes, Survey No. 8

Mode	Specific Gravity	Test Feed K50	Net Grams per Revolution	W.I. (kwh/mt)
A	2.82	4,972	12.0	13.5
B	2.82	5,217	11.0	14.4

The size distribution data from the test samples is given in the following tables.

Table I-3. Rod Mill Feed Size Distribution, Survey No. 8

	Mode A		Mode B	
Screen Size (um)	Ind.%	Cum.% Pass.	Ind.%	Cum.% Pass.
26,900	0.1	99.9	0	100
19,000	1.5	98.4	1.2	98.8
13,500	14.3	84.1	14.9	83.9
9,500	18.0	66.1	16.9	67.0
6,700	14.2	51.9	14.8	52.2
4,750	10.8	41.1	10.8	41.4
3,360	8.1	33.0	7.8	33.6
2,360	6.0	27.0	6.0	27.6
1,700	4.2	22.8	4.3	23.3
1,180	4.0	18.8	4.0	19.3
850	2.4	16.4	2.4	16.9
600	2.3	14.1	2.3	14.6
425	2.2	11.9	2.1	12.5
- 425	11.9		12.5	
Total	100.0		100.0	

Table I-4. Rod Mill Discharge Size Distributions, Survey No.8

Screen Size(um)	A1 Ind.%	A1 Cum.% Pass.	A2 Ind.%	A2 Cum.% Pass.	B1 Ind.%	B1 Cum.% Pass.	B2 Ind.%	B2 Cum.% Pass.
6700	0	100	0	100	0	100	0	100
4750	0.16	99.84	0.19	99.81	0.05	99.95	0.11	99.89
3360	0.55	99.30	0.55	99.26	0.49	99.46	0.43	99.46
2360	2.05	97.25	2.39	96.87	1.74	97.72	1.47	97.99
1700	5.58	91.67	5.97	90.89	4.23	93.50	4.20	93.79
1180	10.85	80.82	11.51	79.38	9.75	83.75	9.24	84.55
850	9.76	71.06	9.88	69.50	9.83	73.92	9.66	74.89
600	10.36	60.70	10.19	59.31	10.97	62.94	11.04	63.85
425	9.39	51.31	9.39	49.92	9.92	53.02	10.67	53.17
300	5.61	45.70			6.37	46.65		
212	5.12	40.58			5.32	41.33		
150	4.39	36.19			4.51	36.82		
106	3.77	32.42			3.74	33.08		
75	3.71	28.71			3.50	29.58		
53	3.61	25.10			3.63	25.95		
38	3.42	21.68			2.95	23.00		
Pan	21.68		49.92		23.00		53.17	
Total	100.0		100.0		100.0		100.0	

APPENDIX J

ROD MILL WATER ADDITION TEST (SURVEY NO. 4)

AT KIDD CREEK MINES

Rod Mill Water Addition Test (Survey No. 4)
at Kidd Creek Mines

Test Description

This plant test on October 23, 1986, was carried out as a continuation of the fines addition test of August 13, except that only the rod mill feed water addition rate was varied.

A. Operating Modes

D: Normal operation.

E: Extra dilution water added to rod mill feed.

F: Additional extra dilution water added to rod mill feed.

B. Data Collection

For each of the three periods, rod mill feed tonnage was determined by recording the time and the rod mill feed totalizer reading at the beginning and end of each period, as follows.

Mode	Time Start / Finish	Tonnage Start / Finish	Tonnage Rate (Wet MTPH)
D	8:45:52/ 9:53:05	992373/992524	134.8
E	10:57:12/11:55:57	992668/992800	134.8
F	1:17:00/ 2:20:12	992982/993124	134.8

The following readings were taken in the motor control centre during the test (averages).

Mode	Voltage	Amperage	Power Factor
D	4140	99.7	0.73
E	4133	100.5	0.72
F	4153	100.3	0.72

During each half of each period, 4 cuts were taken for each of the rod mill feed sample and two separate rod mill discharge samples. The feed rate was maintained at 135 metric tons per hour, and the Marcy density meter was used to set approximate rod mill discharge density levels. A summary of the test timing and control room readings is given below.

8:32 a.m.	Initial power draw readings taken. Trommel spray turned off.
8:46	Start D. Feed water setting, 102 (USGPM scale).
9:53	End D.
9:55	Rod mill feed water adjusted.
10:57	Start E. Feed water setting, 144.
11:56	End E.
11:58	Rod mill feed water adjusted.
1:17 p.m.	Start F. Feed water setting, 191.
2:20	End F.

Sample Analyses

The wet and dry weights were used to calculate the average percent solids of the sampled streams, as follows.

Table J-1. Percent Solids of Sampled Streams, Survey No. 4

	D	E	F
RMF	97.9%	97.9%	97.8%
RMD-1	81.4%	77.1%	71.7%
RMD-2	81.3%	76.8%	72.7%
Ave.	81.4%	77.0%	72.2%

Screen analyses of these samples are given in Tables 2 and 3. The rod mill feed screen analyses were performed in conjunction with the Bond rod mill work index tests at McGill. Rod mill discharge samples were analysed on site. Results of the work index tests for rod mill feed samples D and E are given in Appendix B6. Unfortunately, the test on sample F was ended before equilibrium was reached and the results had to be discarded.

Table J-2. Rod Mill Feed Size Distributions, Survey No. 4

Screen Size (um)	Mode D Ind.%	Mode D Cum.% Pass.	Mode E Ind.%	Mode E Cum.% Pass.	Mode F Ind.%	Mode F Cum.% Pass.
25,500	2.39	97.61	1.33	98.67	1.48	98.52
19,000	6.02	91.59	3.76	94.91	4.52	93.99
13,200	18.07	73.53	17.42	77.49	14.97	79.02
9,500	13.87	59.66	12.50	64.99	13.49	65.53
6,700	11.23	48.43	12.16	52.82	10.85	54.68
4,750	8.32	40.12	9.13	43.69	8.67	46.02
3,350	6.43	33.68	6.94	36.75	6.80	39.21
2,360	5.08	28.60	5.64	31.11	5.62	33.59
1,700	3.48	25.11	3.97	27.14	4.27	29.32
1,180	3.63	21.48	4.01	23.13	4.11	25.21
850	2.33	19.15	2.57	20.56	2.87	22.34
600	2.46	16.69	2.62	17.95	2.97	19.37
425	2.84	13.85	2.83	15.12	3.40	15.97
300	0.98	12.97				
212	1.23	11.64				
150	1.18	10.46				
106	1.11	9.36				
75	1.13	8.23				
53	1.14	7.09				
38	1.01	6.07				
Pan	6.07		15.12		15.97	
	100.0		100.0		100.0	

Table J-3. Rod Mill Discharge Size Distributions, Survey No. 4

(Cummulative % Passing)

Screen Size (um)	D - 1	D - 2	E - 1	E - 2	F - 1	F - 2
3,350	98.23	98.26	98.99	99.23	99.48	99.25
2,360	94.07	94.01	95.69	96.68	97.61	96.90
1,700	86.38	86.24	89.02	90.58	92.56	91.41
1,180	74.38	74.28	77.41	79.15	82.37	80.18
850	63.93	63.89	66.48	68.14	71.61	68.75
600	55.67	55.61	57.58	59.04	62.32	59.29
425	47.94	47.82	49.28	50.52	53.27	50.39
300	44.64	44.57	45.75	46.79	49.31	46.58
212	41.68	41.60	42.56	43.50	45.80	43.21
150	39.17	39.07	39.86	40.71	42.82	40.38
106	37.16	37.04	37.70	38.50	40.45	38.15
75	35.20	35.06	35.62	36.34	38.16	35.97
53	33.50	33.29	32.72	34.43	36.13	34.02
38	31.22	31.04	31.36	32.02	33.58	31.60

APPENDIX K

ROD MILL REDUCED CHARGE LEVEL TEST (SURVEY NO. 9)

AT LES MINES SELBAIE

Rod Mill Reduced Charge Level Test (Survey No. 9) at Les Mines Selbaie

Test Description, Survey No. 9

This plant test on November 11, 1986, was carried out at reduced charge level and fully worn rod mill liner conditions, just before the mill was shut down for a liner change. This work coincided with power draw and charge level measurements, as described in the text.

The rod mill feed tonnage was determined by recording the time and the totalizer reading at the beginning and end of the test, as follows.

Time Start / Finish	Tonnage Start / Finish	Tonnage Rate (WMTPH)
3:29:00/4:31:00	33954.60/34022.15	65.4

During each half of the test, 4 samples were cut and combined at the rod mill feed and discharge. Ammeter readings were taken in the control room and voltage readings in the power plant, as follows.

Amps:	3:55	4:33
	34-38	35-38
	33-38	32-38
	33-38	33-39
	33-38	31-40
	33-39	34-39
	34-36	32-40
	33-37	34-38
	33-38	35-41
	34-39	33-38
Ave.	35.5	36.1

Volts:	4:02	4:42
	3390	3930
	4030	4000
	4020	3980
Ave:	4013	3970

Power frequency was noted to be 60 c.p.s. throughout the test.

Sample Analyses

The percent solids determinations (by weight) were as follows.

Table K-1. Percent Solids of Mill Feed and Discharge, Survey No. 9

Rod mill feed	94.9%
Rod mill discharge - 1	81.4%
Rod mill discharge - 2	81.5%
Rod mill discharge - Ave.	81.5%

A Bond rod mill work index test was carried out on the rod mill feed at 1700um (10 mesh), as follows.

Specific gravity: 2.66 gms/cc
Net gms per revolution: 10.3
Test feed 50% passing size: 5,150um
Work Index: 16.4 kwh/mt

The size distribution data from the test are given in Table B11-2.

Table K-2. Selbaie Survey No. 9 Size Distribution Data

Screen Size(um)	Rod Mill Feed Ind.%	Rod Mill Feed Cum.% Pass.	Rod Mill Discharge-1 Ind.%	Rod Mill Discharge-1 Cum.% Pass.	Rod Mill Discharge-2 Ind.%	Rod Mill Discharge-2 Cum.% Pass.
26500	0	100				
19000	0.86	99.14				
13200	12.14	87.00				
9500	17.19	69.81				
6700	13.61	56.19	0	100	0	100
4750	10.33	45.86	0.12	99.88	0.13	99.87
3350	8.12	37.74	0.44	99.44	0.40	99.47
2360	6.44	31.30	1.56	97.88	1.88	97.59
1700	4.69	26.61	4.49	93.39	4.76	92.83
1180	4.50	22.12	10.51	82.88	10.62	82.21
850	2.72	19.40	9.96	72.92	9.92	72.28
600	2.49	16.91	10.80	62.13	10.69	61.59
425	2.29	14.62	9.69	52.43	9.69	51.90
- 425	14.62		52.43		51.90	
Total	100.00		100.00		100.00	

APPENDIX L.

CHRONOLOGICAL LISTING OF PLANT BALL MILL CIRCUIT CONTROL SYSTEM DATA

Ball Milling Circuit Control System Data

Source(s): Myers and Lewis, 1946.

Plant Identification: Tennessee Copper
Isabella & London

Ore: Copper

Type of Circuit: Rod-ball.

Control Objectives: Constant grind size.

Circuit Instrumentation:	No	Yes	Type/Comments
New ore feed rate		X	
Sump water addition			
Mill water addition			
Feed density			
Feed velocity			
Feed mass flow			
Overflow density		X	
Underflow density			
Overflow particle size			
Sump level			
Others			

Control Instruments:
1. Feed rate weightometers
2. Water addition by orifice plates
3. Pulp density indicator/recorder

Comments: Paper cites the size controlled product naturally occurring from the rod mill despite feed variations. May be of historical interest.

Ball Milling Circuit Control System Data

Source(s): Steffensen and Aubrey, 1957.

Plant Identification: Marmoraton, Marmora, Ont. Ore: Magnetite

Type of Circuit: Rod-ball (closed circuit with cyclones, magnetic separators).

Control Objectives: Constant product size (adjust circulating load to maintain grind with harder ore).

Circuit Instrumentation:

	No	Yes	Type/Comments
New ore feed rate		X	
Sump water addition			
Mill water addition			
Feed density			
Feed velocity			
Feed mass flow			
Overflow density			
Underflow density			
Overflow particle size			
Sump level			
Others			Vacuum of cyclone air core.

Control Instruments:
1. Circulating load used to adjust feed rate
2. Sump level by water addition

Comments: Vacuum @ apex increases as C.L. increases. This was sensed and used to cut back feed.

Data at various circulating loads presented.

Probably one of the first automatic control systems, especially with cyclone.

Reported Benefits: Control of -325 mesh fraction.

Ball Milling Circuit Control System Data

Source(s): Daniel, 1967; Kelly and Gow, 1966.

Plant Identification: East Malartic Ore: Gold

Type of Circuit: Rod-ball.

Type of Automatic Control: Noise level

Control Objectives: Constant grind size.

Circuit Instrumentation:	No	Yes	Type/Comments
New ore feed rate			
Sump water addition			
Mill water addition			
Feed density			
Feed velocity			
Feed mass flow			
Overflow density			
Underflow density			
Overflow particle size			
Sump level			
Others			

Control Instruments: 1. Sound level - feed rate

Comments: This did not catch on due to problems with use of this type of signal. However, it does show that even rudimentary attempts at control can show significant results.

Reported Benefits: Reduced variability of grind. Spread of size distribution did not change significantly.

Ball Milling Circuit Control System Data

Source(s): Bilon, 1967.

Plant Identification: Moose Mountain, National Steel, Capreol, Ont. Ore: Magnetite

Type of Circuit: Rod mill - mag sep - closed circuit ball mill (with mag sep).

Control Objectives:
1. Constant product size.
2. (varying feed rate).

Variable Speed Pump(s): No.

Circuit Instrumentation:

	No	Yes	Type/Comments
New ore feed rate		X	
Sump water addition	X		
Mill water addition	X		
Feed density		X	
Feed velocity	X		
Feed mass flow	X		
Overflow density	X		
Underflow density	X		
Overflow particle size	X		
Sump level		X	Float
Others			

Control Instruments:
1. New ore/water ratio.
2. New ore rate controlled by cyclone feed density.
3. Sump level controlled by water addition.

Comments: Basic equipment (slide-wires and timers), but sound set up.

Reported Benefits: Stabilized grind, increased recovery.

Ball Milling Circuit Control System Data

Source(s): Cross, 1967.

Plant Identification: West Driefontain, S.A. Ore: Gold

Type of Circuit: Rod-pebble.

Control Objectives: Density and quantity of cylcone underflow.

Variable Speed Pump(s): No.

Circuit Instrumentation:

	No	Yes	Type/Comments
New ore feed rate			
Sump water addition			
Mill water addition			
Feed density			
Feed velocity			
Feed mass flow			
Overflow density		X	
Underflow density		X	
Overflow particle size			
Sump level			
Others Density monitor on cone of cyclone. Pebble controller based on mill power draw.			

Control Instruments:
1. Cyclone overflow density to regulate sump water.
2. Cyclone underflow density to regulate apex opening.
3. Apex opening (C.L.) to regulate are feed rate.
   + water proportioned to rod mill feed.
   + pebbles added to maintain mill power draw.

Comments: Advantages of high circulating load cited, along with severe disadvantages of overgrinding.
This is one of the earliest, successful reported applications of grinding control.

Reported Benefits: More stable operation.
$70,000 per year in improved recovery (reduced tailings values).

Ball Milling Circuit Control System Data

Source(s): Watson et al, 1970.

Plant Identification: Pilot plant.

Type of Circuit: Single stage pebble (or ball) mill.

Type of Automatic Control: Computer.

Control Objectives: Constant product size.

Variable Speed Pump(s): Yes

Circuit Instrumentation:	No	Yes	Type/Comments
New ore feed rate		X	
Sump water addition		X	
Mill water addition			N/A
Feed density		X	
Feed velocity		X	
Feed mass flow		X	
Overflow density			
Underflow density		X	and flow!
Overflow particle size			
Sump level		X	
Others			

Control Instruments:
1. Cyclone underflow mass flow regulates new feed rate.
2. Sump level regulates pump speed.
3. Cyclone feed density regulates sump water.

Comments: Concluded single variable control cannot be achieved at maximum grinding efficiency - i.e. need to vary feed rate or tolerate variation in product size. Common interactions described.

Ball Milling Circuit Control System Data

Source(s): Diaz and Musgrove, 1973.

Plant Identification: Inspiration Consol., Arizona Ore: Copper

Type of Circuit: Single stage ball mill (grate type).

Type of Automatic Control: Analog.

Control Objectives: 1. Constant grind.
2. Maximum tonnage.

Variable Speed Pump(s): Yes

Circuit Instrumentation:

	No	Yes	Type/Comments
New ore feed rate		X	
Sump water addition		X	
Mill water addition		X	
Feed density	X		
Feed velocity	X		
Feed mass flow	X		
Overflow density		X	(from PSM)
Underflow density	X		
Overflow particle size		X	
Sump level			
Others			Mill load

Control Instruments:
1. Particle size of Cyclone overflow by water addition.
2. Ball mill load - new feed rate (i.e. by mill mass).
3. Ratio new water/feed.

Comments: They indicate that they think that low circulating load is superior (as at Craigmont). Their concept of cyclone operation is erroneous!
Noted pH had direct effect on Cyclone overflow size, but was really a viscoscity effect.

Reported Benefits: 4-10% improvement in throughout during testwork.

Ball Milling Circuit Control System Data

Source(s): Weber and Diaz, 1973; Gault et al, 1979.

Plant Identification: Craigmont. Ore: Copper

Type of Circuit: Rod-ball.

Control Objectives: Fixed product size.

Variable Speed Pump(s): No.

Circuit Instrumentation:

	No	Yes	Type/Comments
New ore feed rate		X	
Sump water addition	X		
Mill water addition	X		
Feed density	X		
Feed velocity	X		
Feed mass flow	X		
Overflow density		X	(from PSM)
Underflow density	X		
Overflow particle size		X	
Sump level		X	
Others			

Control Instruments:
1. Cyclone overflow particle size controlled by sump water addition.
2. Sump level controlled by new feed rate.

Comments: Mill water set manually.
Feed box level found reliable indication of circulating load.
Beautifully simple and precise!

Unfortunately, they thought running at low circulating load was good for capacity.

Reported Benefits: 5% increase in tonnage,
stable operation and consistent grind.

Ball Milling Circuit Control System Data

Source(s): Atkins et al, 1974.

Plant Identification: Rand Mines Ltd., S.A. Ore: Gold

Type of Circuit: Rod-pebble.

Type of Automatic Control: Analog.

Control Objectives: Constant product size.

Circuit Instrumentation:	No	Yes	Type/Comments
New ore feed rate		X	Weightometer
Sump water addition			
Mill water addition			
Feed density		X	
Feed velocity		X	
Feed mass flow		X	
Overflow density			
Underflow density			
Overflow particle size		X	
Sump level			
Others			Ammeter - pebble feeder

Control Instruments: (Discussed in general)

Comments: Reported simple analog systems work well - little need for anything more complex.

Cyclone feeder overflow density alone cannot control - need feed rate as well. Behavior of product size when feed shut off studied. Sump concluded to be critical component in behavior of circuit.

Ball Milling Circuit Control System Data

Source(s): Pewings, 1981; Balles et al, 1977; Lees and Lynch, 1972; Lynch, 1977; Gault et al, 1979.

Plant Identification: Mt. Isa, Australia Ore(s): Copper, lead, zinc.

Type of Circuit: Rod + 2 stage ball.

Type of Automatic Control: Computer.

Control Objectives: 1. Constant product size.
2. Maximum throughput*.

Variable Speed Pump(s): Yes.

Circuit Instrumentation:

	No	Yes	Type/Comments
New ore feed rate		X	
Sump water addition		X	
Mill water addition		X	
Feed density		X	
Feed velocity		X	(secondary cyclone)
Feed mass flow		X	(secondary cyclone)
Overflow density	X		Secondary cylone overflow density and volume (flotation feed)
Underflow density	X		
Overflow particle size	X		
Sump level		X	
Others			Ball mill amps for overload condition. Cyclone feed pressures.

Control Instruments: 1. Sec. cyclone mass flow regulates new feed
2. Pump speed controls sump level
3. Cyclone cut size regulated by cyclone model (feed conditions)

Comments: * Philosophy of constant feed rate and constant product size was abandoned in favor at maximizing throughput rate (10).

Reported Benefits: Circulating load 20% higher, capacity 5% for automatic versus manual control.
Cost of automatic control system was 1/4 of 1% of plant capital cost.
Considerably narrowerer product size distribution.

Ball Milling Circuit Control System Data

Source(s): Bradburn et al, 1976; Gault et al, 1979.

Plant Identification: Brenda   Ore(s): Copper, molybdenum

Type of Circuit: Rod-ball.

Type of Automatic Control: Computer.

Control Objectives: 1. Maximum tonnage.
2. Limit product size to less than 30% + 65 mesh.

Variable Speed Pump(s): No.

Circuit Instrumentation:

	No	Yes	Type/Comments
New ore feed rate		X	
Sump water addition		X	
Mill water addition		X	
Feed density		X	
Feed velocity	X		
Feed mass flow	X		
Overflow density		X	
Underflow density	X		
Overflow particle size		X	
Sump level		X	
Others			Rod mill sound, ball mill sound

Control Instruments: 1. Rod mill feed regulates cyclone feed density.
2. Ball mill water by mill sound.
3. Total water to circuit constant (sump water by difference). Water ratioed to new feed.

Comments: Variable crushing plant modes lead to highly variable feed size. Depending on feed, production constraint changes from rod to ball mill.
Analog system was slow to react to large ore variations, and conflict between need to increase vrs. decrease feed rate for 2 mills. System at constraints/overloads switch control/override to suit (eg. mill noise, sump level, etc.).

Reported Benefits: 4% increase in tonnage, with a 1 year payback period.

Ball Milling Circuit Control System Data

Source(s): Balles set al, 1977.

Plant Identification: Phelps Dodge, Morenci, Arizona. Ore: Copper

Type of Circuit: Single stage ball mill/duplex classifers.

Type of Automatic Control: Analog + computer.

Control Objectives: Vary product size, depending on feed grade.

Variable Speed Pump(s): N/A

Circuit Instrumentation:

	No	Yes	Type/Comments
New ore feed rate		X	
Sump water addition		X	(at classifier)
Mill water addition		X	
Feed density	X		
Feed velocity	X		
Feed mass flow	X		
Overflow density	X		
Underflow density			
Overflow particle size	X		
Sump level			
Others			X-ray analyser, classifier motor loadings

Control Instruments:

1. Classifier load to control circ. load.
2. Feed rate to control mill loading.
3. Part. size setpoint by computer algorithm.
4. Part size maintained by classifier water.
5. Feed water ratio.
6. Ball mill overload by power draw.
7. Manual ball mill headwater.

Comments: Unusual, complex.

Reported Benefits: Increased tonnage and reduced reagent consumption, stability, flexibility and process knowledge.

Ball Milling Circuit Control System Data

Source(s): Lynch, 1977.

Plant Identification: Bougainville, P.N.G. Ore: Copper

Type of Circuit: Single stage ball mills (primary).

Type of Automatic Control: Analog (conversion for computer underway).

Control Objectives: 1. Max. tonnage.
2. Finest poss grind.*

Variable Speed Pump(s): Mills 1-8, no. Mill 9, yes. All eventually changed

Circuit Instrumentation:	No	Yes	Type/Comments
New ore feed rate		X	
Sump water addition		X	
Mill water addition			
Feed density		X	
Feed velocity		(X)	With variable speed pumps only
Feed mass flow			
Overflow density		X	
Underflow density			
Overflow particle size			
Sump level		X	
Others			

Control Instruments:

Fixed Speed Pumps	Variable Speed Pumps
1. Sump level by water addition.	1. Sump by pump speed.
2. Feed rate by cyclone overflow density.	2. Cyclone overflow density by sump.
3. Cyclone overflow density by feed rate/ sump water ratio.	3. New feed by cyclone mas flow.
4. Mill water manually.	4. Mill water manually.

Comments: The normal grind is coarser than optimum for flotation recoveries, but is so to keep tonnage and copper production schedule.

Ball Milling Circuit Control System Data

Source(s): Manning and chang, 1977; Balles et al, 1977.

Plant Identification: Climax Mill. Ore: Molybdenum

Type of Circuit: Single stage ball mill.

Type of Automatic Control: computer.

Control Objectives:
1. Constant product size.
2. Maximum tonnage/fixed tonnage.
3. Maintain mill density (maintain floatation feed density).

Variable Speed Pump(s): No pumps - "duplex" classifiers on some lines.

Circuit Instrumentation:

	No	Yes	Type/Comments
New ore feed rate		X	Weightometer
Sump water addition		X	Orifice plates
Mill water addition		X	Orifice plates
Feed density		X	
Feed velocity		X	
Feed mass flow		X	
Overflow density		X	
Underflow density	X		
Overflow particle size	X		See note*
Sump level		X	Capacitone probe
Others			

Control Instruments:
1. Cyclone feed density regulated by fresh feed rate.
2. (Sump water) sump level kept high by maximum level set point.
3. Mill water regulated to maintain constant density from classifier feed and sump water calculation, to yield constant classifier overflow density.

Comments: * Use of PSM was discontinued after using it to help develop regression equation (control algorithm) for classifier product size control.

Note, flowsheet not displayed. Sump level control for cyclone circuits only.

Reported Benefits: Capacity increase of 9%, while variance of product size reduced very significantly.

Ball Milling Circuit Control System Data

Source(s): Lynch, 1977; Balles set al, 1977.

Plant Identification: Asarco Silver Bell, Arizona

Ore(s): Copper, molybdenum

Type of Circuit: Single stage ball mills (primary), grate discharge.

Type of Automatic Control: Computer.

Control Objectives:
1. Fixed grind.
2. Max throughout.

Variable Speed Pump(s): (converted).

Circuit Instrumentation:

	No	Yes	Type/Comments
New ore feed rate		X	
Sump water addition		X	
Mill water addition		X	
Feed density		X	
Feed velocity		X	
Feed mass flow		X	
Overflow density	X		
Underflow density	X		
Overflow particle size	X		
Sump level		X	
Others			Mill weight for grate blockage indication

Control Instruments:
1. Fresh feed - cyclone overflow product size.
2. Ball mill water - mill density.
3. Sump water - circulating load.
4. Pump speed - sump level

Comments: Control algorithm to maintain cyclone overflow size was developed and upgraded.

Difficulties with bubble tubes. Replaced with diaphragm pressure sensors.

Ball Milling Circuit Control System Data

Source(s): Watts et al, 1978.

Plant Identification: Pinto Valley, Arizona Ore: Copper

Type of Circuit: Single stage ball mill.

Type of Automatic Control: Computer.

Control Objectives:
1. Maximum tonnage.
2. Maintain product size.

Variable Speed Pump(s): No.

Circuit Instrumentation:

	No	Yes	Type/Comments
New ore feed rate		X	
Sump water addition		X	
Mill water addition		X	
Feed density		X	
Feed velocity		X	
Feed mass flow		X	
Overflow density	X		
Underflow density	X		
Overflow particle size	X		
Sump level		X	
Others			Cyclone feed pump amps, cyclone feed pressure

Control Instruments:
1. Cyclone mass flow regulated by new ore feed rate
2. Ball mill water to control mill density
3. Sump water by level measurement

Comments: Above reference describes preliminary development, wrought with problems, sump water input uncontrollable. Reported as "interim" state of development.

Ball Milling Circuit Control System Data

Source(s): Ribiero et al, 1979 and 1981.

Plant Identification: Arafertil, Brazil (Serrana Mining Company)

Ore: Phosphate (flotation process)

Type of Circuit: Rod-ball.

Type of Automatic Control: Analog.

Control Objectives:
1. Fixed product size.
2. Maximum tonnage.

Variable Speed Pump(s): Yes.

Circuit Instrumentation:

	No	Yes	Type/Comments
New ore feed rate		X	
Sump water addition		X	
Mill water addition		X	
Feed density		X	
Feed velocity		X	
Feed mass flow		X	
Overflow density		X	(through PSM)
Underflow density	X		
Overflow particle size		X	
Sump level		X	
Others			Cyclone feed pressure. Pump speed.

Control Instruments*:
1. Cyclone overflow particle size regulated by sump dilution water.
2. Sump level regulated by pump speed.
3. Cyclone feed mass flow controlled by new feed rate.

Comments: * Found to be best strategy after numerous, rigourous experiments with a large variety of strategies. This is an excellent monograph on alternative control schemes.

Reported Benefits: "The instrumentation paid for itself in the 3 month familiarization phase!"

Ball Milling Circuit Control System Data

Source(s): Hulbert and Barker, 1984.

Plant Identification: East Driefontein, S.A. Ore: Gold

Type of Circuit: Rod-pebble, 2 stage cyclones.

Type of Automatic Control: Multivariable/mini-computer.

Control Objectives:
1. Controlled product size.
2. Maximum throughput.

Variable Speed Pump(s): YES

Circuit Instrumentation:	No	Yes	Type/Comments
New ore feed rate		X	
Sump water addition		X	Both.
Mill water addition		X	Rod mill only.
Feed density		X	Primary only.
Feed velocity		X	" "
Feed mass flow		X	
Overflow density		X	Secondary, with PSM.
Underflow density	X		
Overflow particle size		X	
Sump level	X		PSM, secondary cyclone.
Others			

Control Instruments:

Multivariable Actions	Controlled
Rod mill feed water.	Overflow particle size (see cyclone)
Water to sumps (2).	Cyclone feed density (prom. cyclone)
	Cyclone feed flowrate (prom. cyclone)

- Rod mill feed rate controlled proportioned to R.M. water rate.
- Cyclone roping constraint.

Comments: Authors noted importance of particle size and regulation at most efficient operating point. Circuit efficiency very sensitive to circulating load.
Concluded satisfactory circuit operation requires that feed rate be allowed to vary, by + or - 10%

Reported Benefits:

Ball Milling Circuit Control System Data

Source(s): Leau and Baker, 1984; Lynch 1977.

Plant Identification: New Broken Hill — Ore(s): Lead & Zinc

Type of Circuit: Rod-ball-float-ball.

Type of Automatic Control: Computer.
Control Objectives: 1. Correct mineral distribution in flotation feed.
2. Max. circulating load in primary circuit (feed rate may be either fixed or maximum).

Variable Speed Pump(s): Yes.

Circuit Instrumentation:	No	Yes	Type/Comments
New ore feed rate		X	
Sump water addition		X	
Mill water addition		X	
Feed density		X	
Feed velocity		X	
Feed mass flow		X	
Overflow density		X	
Underflow density	X		
Overflow particle size	X		
Sump level		X	(Assumed-controls pump speed)
Others			Cyclone feed pressure.

Control Instruments:

Primary	Secondary
1. Cyclone overflow mineral distribution controls new feed & water	Water only.
2. Variable speed pump controls sump level.	Same
3. Limit feed rate to prevent ball mill overload due to high circulating load.	N/A

No. of cyclones varied at low tonnage in seconardy circuit.

Comments: Control of mineral distribution may be the most rational approach to flotation feed control, but this is the only plant known to practice it.

Ball Milling Circuit Control System Data

Source(s): Mansonti and Maio, 1984.

Plant Identification: Ozark Lead Ore: Lead

Type of Circuit: Rod-ball.

Type of Automatic Control: Computer - Distributed Digital.

Control Objectives: 1. Constant particle size.
2. Max. tonnage.

Variable Speed Pump(s): Yes.

Circuit Instrumentation:	No	Yes	Type/Comments
New ore feed rate		X	Weightometer
Sump water addition	X		
Mill water addition	X		
Feed density		X	Nuclear
Feed velocity		X	Magnetic
Feed mass flow		X	
Overflow density		X	
Underflow density	X		
Overflow particle size		X	
Sump level		X	Sonic
Others			

Control Instruments: 1. Cyclone overflow size regulated by sump water.
2. Circ. load controlled by new feed rate.
3. Sump level controlled by pump speed.

Also, rod mill water ratioed and flotation "make-up" water by cyclone overflow density.

Comments: Similar to Buick, except for variable speed pumps.

Reported Benefits: Increased tonnage 15%.
Increased Pb and Zn recoveres.

Ball Milling Circuit Control System Data

Source(s): Hals et al, 1986.

Plant Identification: Cuajone, Peru Ore: Copper

Type of Circuit: Single-stage ball mill.

Type of Automatic Control: Computer (distributed).
Control Objectives: 1. Product size constant.
2. Maximum capacity.
3. Ball mill density constant.

Variable Speed Pump(s): No.

Circuit Instrumentation:	No	Yes	Type/Comments
New ore feed rate		X	
Sump water addition		X	
Mill water addition		X	
Feed density		X	
Feed velocity		X	
Feed mass flow		X	
Overflow density		X	
Underflow density	X		
Overflow particle size		X	
Sump level		X	
Others			Mill feed/discharge density.

Control Instruments*: 1. Particle size controlled by feed rate.
2. Sump level controlled by water addition.
3. BM density controlled by calculated water addition.

Comments: * Operating constraints
a. maximum tonnage for downstream processes
b. cyclones roping
c. ball mill overload
d. cyclone feed pump overload

Reported Benefits: 7.5% increase in tonnages. However, a 1% loss in recovery due to the extra load on flotation is being suffered.

Ball Milling Circuit Control System Data

Source(s): Derster, 1986; Perkins and Marneweke, 1978.

Plant Identification: Buick, Amax. Lead, Missouri

Ore(s): Lead, Zinc

Type of Circuit: Rod-ball.

Type of Automatic Control: Analog, 1978; computer, 1986.

Control Objectives:
1. Constant product size.
2. Maximum tonnage.

Variable Speed Pump(s): No.

Circuit Instrumentation:

	No	Yes	Type/Comments
New ore feed rate		X	
Sump water addition			
Mill water addition			
Feed density			
Feed velocity			
Feed mass flow			
Overflow density		X	by PSM
Underflow density	X		
Overflow particle size		X	
Sump level		X	
Others			

Control Instruments:
1. Sump water addition regulated by psm.
2. Sump level regulated by need feed rate.
3. Rod mill water ratio control.

Comments: This appears to be the simplest and most effective approach to date.

Reported Benefits: Improved throughput, stability, and other operating cost reductions as a result of plant wide computer control. Savings estimated $2.3 million/year. 3-5% increase is throughput was achieved with analog over manual control.

APPENDIX M

PRIMARY HYDROCYCLONE MODEL PARAMETERS FOR

KIDD CREEK SURVEY NO. 1

Primary Hydrocyclone Model Parameters for Kidd Creek Survey No. 1

The mass balance of solids (S.G. = 3.3) and water at the primary cyclones during survey no. 1 is given in Table B12-1. The calculated cyclone feed size distribution and classifier actual (Y) and corrected (Yc) recoveries to underflow are given in Table B12-2. The estimate of the solids bypass fraction (Rf) which gave the bst fit (by inspection) of the data to Plitt's cyclone performance curve was about 5/6 times the water bypass to the cyclone underflow. From a log-log plot of ln [1/(1-Yc)] versus d (the mean micron size of the material on each screen size), the parameters m (slope) and d50c corrected d50 cut size in the Plitt cyclone performance equation were estimated (Plitt, 1981). In this case, m=0.95, and d50c=95 um.

Table B12-3 shows the application of Plitt's cyclone separation curve, using the above calculated model parameters, to the cyclone feed size distribution. A standard electronic spreadsheet program (Visicalc) is used to calculate:

a) corrected recoveries (Yc) of each screen size;

b) corrected (i.e., no bypass) underflow and overflow (U_1 and O_1) masses for each screen size, based on 100 mass units in the feed;

Table M-1. Primary Hydrocyclone Solids and Water Mass Balance, Kidd Creek Survey No. 1

	Feed	Overflow	Underflow
MTPH Solids	681	127	554
MTPH Water	299	120	179
MTPH Slurry	980	247	733
% Solids w/w	65.5%	51.5%	75.5%

Table M-2. Primary Hydrocyclone Feed Size and Recovery Data, Kidd Creek Survey No. 1

A. Calculated Primary Cyclone Feed

Mesh	Ind. %	Cum. % Passing
6	0.57	99.43
8	1.39	98.04
10	2.60	95.44
14	4.24	91.20
20	4.58	86.62
28	4.80	81.82
35	6.32	75.50
48	7.76	67.74
65	9.62	58.12
100	11.23	46.89
150	9.14	37.75
200	7.85	29.90
270	5.06	24.84
400	5.56	19.28
-400	19.28	
	100.00	

B. Actual and Corrected Recoveries to Underflow

Mesh	Recovery (Y)	Corrected Recovery (Yc)
14	100	100
20	99.96	99.9
28	99.8	99.5
35	97.7	95.5
48	93.2	86.5
65	89.8	79.6
100	88.1	76.2
150	84.0	68.1
200	76.1	52.3
270	71.4	42.8
400	63.4	26.8
-400	54.2	8.3

c) bypass masses, based on the bypass fraction (Rf) and the mass in each screen size in the cyclone overflow;

d) actual masses recovered to underflow and overflow (U on O), by subtracting the bypass masses from overflow and adding them to the underflow; and

e) the underflow and overflow percent size distributions, both individual and cumulative.

The resulting overflow and underflow size distributions, and the overall mass split, show the goodness of fit of the model to the data from the sample survey. The size distribution data are given in Table M-4. The cyclone overflow displays the most deviation, but this is not important for estimation of the mill fines inventory. The circulating load from the fitted model is 436%, versus the measured value of 434%.

Table M-3. **Plitt Cyclone Model Data, Primary Hydrocyclones, Kidd Creek Survey No. 1.**

D50C = 95

RF = 50.00

M = 0.95

MESH	DIAM.	GEOMEAN	FEED	YC	U'	O'
6	3360	3996	0.57	1.00	0.57	0.00
8	2360	2807	1.39	1.00	1.39	0.00
10	1680	1998	2.60	1.00	2.60	0.00
14	1180	1403	4.24	1.00	4.24	0.00
20	850	1011	4.58	1.00	4.57	0.01
28	600	714	4.80	0.99	4.76	0.04
35	425	505	6.32	0.97	6.11	0.21
48	300	357	7.76	0.91	7.08	0.68
65	212	252	9.62	0.83	7.95	1.67
100	150	178	11.23	0.72	8.05	3.18
150	106	126	9.14	0.60	5.45	3.69
200	75	89	7.85	0.48	3.76	4.09
270	53	63	5.06	0.37	1.90	3.16
400	38	45	5.56	0.29	1.61	3.95
−400	0	19	19.28	0.14	2.69	16.59
			100.00			

Table M-3. (Continued)

BYPASS	U	O	U%	O%	CUM U%	CUM O%
0.00	0.57	0.00	0.70	0.00	99.30	100.00
0.00	1.39	0.00	1.71	0.00	97.59	100.00
0.00	2.60	0.00	3.20	0.00	94.40	100.00
0.00	4.24	0.00	5.21	0.00	89.18	100.00
0.00	4.58	0.00	5.63	0.02	83.56	99.98
0.02	4.78	0.02	5.87	0.12	77.69	99.86
0.11	6.21	0.11	7.64	0.57	70.05	99.29
0.34	7.42	0.34	9.12	1.82	60.93	97.47
0.83	8.79	0.83	10.80	4.48	50.13	93.00
1.59	9.64	1.59	11.85	8.54	38.28	84.46
1.85	7.29	1.85	8.97	9.90	29.32	74.56
2.04	5.81	2.04	7.14	10.96	22.18	63.59
1.58	3.48	1.58	4.27	8.49	17.91	55.10
1.97	3.59	1.97	4.41	10.59	13.50	44.51
8.30	10.98	8.30	13.50	44.51	0.00	0.00
	81.36	18.64	100.00	100.00		

Table M-4. Model vs. Measured Cyclone Product Size Distributions, Kidd Creek Primary Cyclones, Survey no. 1 (Cumulative % Passing)

Screen Size (um)	Cyclone Overflow		Cyclone Underflow	
	Measured	Model	Measured	Model
3360			99.32	99.30
2360			97.64	97.59
1680			94.48	94.40
1180	99.97	100	89.23	89.18
850	99.95	99.98	83.58	83.56
600	99.87	99.86	77.66	77.69
425	99.05	99.29	70.06	70.05
300	96.23	97.47	61.17	60.93
212	91.09	93.00	50.55	50.13
150	84.03	84.46	38.38	38.28
106	76.18	74.56	28.94	29.32
75	66.19	63.59	21.59	22.18
53	58.45	55.10	17.15	17.91
38	47.37	44.51	12.84	13.50

APPENDIX N

ESTIMATED PRIMARY HYDROCYCLONE PERFORMANCE DATA FOR DIFFERENT CIRCULATING LOADS AT KIDD CREEK

Estimated Primary Hydrocyclone Performance Data For Different Circulating Loads at Kidd Creek

Table N-1. Kidd Creek Primary Hydrocyclone Solids and Water Estimated Mass Balances at Different Circulating Load Ratios.

	Feed	Overflow	Underflow
250% C.L. Ratio			
MTPH Solids	444.5	127.0	317.5
MTPH Water	222.6	119.6	103.0
MTPH Slurry	667.1	246.6	420.5
% Solids w/w	66.6%	51.5%	75.5%
350% C.L. Ratio			
MTPH Solids	571.5	127.0	444.5
MTPH Water	263.8	119.6	144.2
MTPH Slurry	835.3	246.6	588.7
% Solids w/w	68.4%	51.5%	75.5%
450% C.L. Ratio			
MTPH Solids	698.5	127.0	571.5
MTPH Water	305.1	119.6	185.5
MTPH Slurry	1003.6	246.6	757.0
% Solids w/w	69.6%	51.5%	75.5%
550% C.L. Ratio			
MTPH Solids	825.5	127.0	698.5
MTPH Water	346.3	119.6	226.7
MTPH Slurry	1171.8	246.6	925.2
% Solids w/w	70.4%	51.5%	75.5%

Table N-2. 250% Circulating Load Cyclone Performance

D50C = 136

RF = 39.00

M = 1.30

MESH	DIAM.	GEOMEAN	FEED	YC	U'	O'
6	3360	3996	0.50	1.00	0.50	0.00
8	2360	2807	1.00	1.00	1.00	0.00
10	1680	1998	1.50	1.00	1.50	0.00
14	1180	1403	2.00	1.00	2.00	0.00
20	850	1011	3.00	1.00	3.00	0.00
28	600	714	4.00	1.00	3.99	0.01
35	425	505	6.00	0.98	5.87	0.13
48	300	357	10.00	0.91	9.12	0.88
65	212	252	11.00	0.79	8.66	2.34
100	150	178	11.00	0.63	6.90	4.10
150	106	126	8.00	0.47	3.73	4.27
200	75	89	8.00	0.33	2.64	5.36
270	53	63	7.00	0.23	1.58	5.42
400	38	45	5.00	0.15	0.76	4.24
-400	0	19	22.00	0.05	1.15	20.85
			100.00			

Notes:

- Feed - cyclone feed size distribution, ind.%
- YC - corrected cyclone recovery
- U' - underflow weights, no bypass
- O' - overflow weights, no bypass
- U - underflow weights, with bypass
- O - overflow weights, with bypass
- U% - underflow size distribution, ind.%
- O% - overflow size distribution, ind.%

Table N-2. (Continued)

BYPASS	U	O	U%	O%	CUM U%	CUM O%
0.00	0.50	0.00	0.70	0.00	99.30	100.00
0.00	1.00	0.00	1.41	0.00	97.89	100.00
0.00	1.50	0.00	2.11	0.00	95.77	100.00
0.00	2.00	0.00	2.82	0.00	92.95	100.00
0.00	3.00	0.00	4.23	0.00	88.73	100.00
0.00	3.99	0.01	5.63	0.02	83.10	99.98
0.05	5.92	0.08	8.34	0.28	74.75	99.70
0.34	9.46	0.54	13.33	1.85	61.42	97.85
0.91	9.57	1.43	13.49	4.92	47.93	92.93
1.60	8.50	2.50	11.98	8.62	35.96	84.31
1.67	5.40	2.60	7.60	8.97	28.35	75.34
2.09	4.73	3.27	6.67	11.26	21.69	64.08
2.12	3.69	3.31	5.20	11.39	16.48	52.69
1.65	2.42	2.58	3.40	8.90	13.08	43.79
8.13	9.28	12.72	13.08	43.79	0.00	0.00
	70.96	29.04	100.00	100.00		

Table N-3. 350% Circulating Load Cyclone Performance

D50C = 116

RF = 46.00

M = 1.10

MESH	DIAM.	GEOMEAN	FEED	YC	U'	O'
6	3360	3996	0.50	1.00	0.50	0.00
8	2360	2807	1.50	1.00	1.50	0.00
10	1680	1998	3.00	1.00	3.00	0.00
14	1180	1403	3.00	1.00	3.00	0.00
20	850	1011	3.00	1.00	3.00	0.00
28	600	714	4.00	0.99	3.98	0.02
35	425	505	7.00	0.97	6.79	0.21
48	300	357	11.00	0.91	9.99	1.01
65	212	252	10.00	0.80	8.04	1.96
100	150	178	10.00	0.67	6.71	3.29
150	106	126	9.00	0.53	4.79	4.21
200	75	89	8.00	0.40	3.24	4.76
270	53	63	6.00	0.30	1.79	4.21
400	38	45	5.00	0.22	1.09	3.91
-400	0	19	19.00	0.09	1.72	17.28
			100.00			

Table N-3. (Continued)

BYPASS	U	O	U%	O%	CUM U%	CUM O%
0.00	0.50	0.00	0.64	0.00	99.36	100.00
0.00	1.50	0.00	1.92	0.00	97.43	100.00
0.00	3.00	0.00	3.85	0.00	93.58	100.00
0.00	3.00	0.00	3.85	0.00	89.73	100.00
0.00	3.00	0.00	3.85	0.00	85.89	100.00
0.01	3.99	0.01	5.12	0.06	80.77	99.94
0.10	6.89	0.11	8.84	0.52	71.93	99.42
0.47	10.45	0.55	13.41	2.48	58.52	96.94
0.90	8.94	1.06	11.47	4.80	47.05	92.14
1.51	8.23	1.77	10.55	8.04	36.49	84.10
1.94	6.73	2.27	8.63	10.30	27.86	73.79
2.19	5.43	2.57	6.97	11.65	20.89	62.15
1.94	3.73	2.27	4.78	10.30	16.11	51.85
1.80	2.89	2.11	3.71	9.57	12.41	42.28
7.95	9.67	9.33	12.41	42.28	0.00	0.00
	77.93	22.07	100.00	100.00		

Table N-4. **450% Circulating Load Cyclone Performance**

D50C = 94

RF = 51.00

M = 0.95

MESH	DIAM.	GEOMEAN	FEED	YC	U'	O'
6	3360	3996	0.50	1.00	0.50	0.00
8	2360	2807	2.50	1.00	2.50	0.00
10	1680	1998	3.00	1.00	3.00	0.00
14	1180	1403	3.00	1.00	3.00	0.00
20	850	1011	4.00	1.00	3.99	0.01
28	600	714	5.00	0.99	4.96	0.04
35	425	505	5.50	0.97	5.32	0.18
48	300	357	10.00	0.91	9.15	0.85
65	212	252	11.00	0.83	9.13	1.87
100	150	178	10.00	0.72	7.20	2.80
150	106	126	8.00	0.60	4.80	3.20
200	75	89	7.00	0.48	3.38	3.62
270	53	63	6.00	0.38	2.27	3.73
400	38	45	5.00	0.29	1.46	3.54
-400	0	19	19.50	0.14	2.75	16.75
			100.00			

Talbe N-4. **(Continued)**

BYPASS	U	O	U%	O%	CUM U%	CUM O%
0.00	0.50	0.00	0.61	0.00	99.39	100.00
0.00	2.50	0.00	3.05	0.00	96.34	100.00
0.00	3.00	0.00	3.66	0.00	92.69	100.00
0.00	3.00	0.00	3.66	0.00	89.03	100.00
0.00	4.00	0.00	4.87	0.01	84.16	99.98
0.02	4.98	0.02	6.07	0.12	78.10	99.87
0.09	5.41	0.09	6.60	0.49	71.50	99.38
0.44	9.58	0.42	11.68	2.33	59.82	97.05
0.96	10.08	0.92	12.28	5.12	47.54	91.92
1.43	8.63	1.37	10.52	7.64	37.02	84.28
1.63	6.43	1.57	7.84	8.75	29.19	75.54
1.85	5.23	1.77	6.37	9.89	22.82	65.64
1.90	4.17	1.83	5.08	10.20	17.74	55.44
1.80	3.27	1.73	3.98	9.67	13.76	45.77
8.54	11.29	8.21	13.76	45.77	0.00	0.00
	82.06	17.94	100.00	100.00		

Table N-5. <u>550% Circulating Load Cyclone Performance</u>

D50C = 88

RF = 55.00

M = 0.90

MESH	DIAM.	GEOMEAN	FEED	YC	U'	O'
6	3360	3996	0.50	1.00	0.50	0.00
8	2360	2807	1.00	1.00	1.00	0.00
10	1680	1998	2.50	1.00	2.50	0.00
14	1180	1403	3.00	1.00	3.00	0.00
20	850	1011	4.00	1.00	3.99	0.01
28	600	714	6.00	0.99	5.94	0.06
35	425	505	10.00	0.96	9.65	0.35
48	300	357	10.00	0.91	9.13	0.87
65	212	252	10.00	0.83	8.33	1.67
100	150	178	9.00	0.73	6.57	2.43
150	106	126	8.00	0.62	4.93	3.07
200	75	89	7.00	0.50	3.53	3.47
270	53	63	5.00	0.40	2.01	2.99
400	38	45	4.00	0.32	1.27	2.73
-400	0	19	20.00	0.16	3.20	16.80
			100.00			

Table N-6. (Continued)

BYPASS	U	O	U%	O%	CUM U%	CUM O%
0.00	0.50	0.00	0.59	0.00	99.41	100.00
0.00	1.00	0.00	1.18	0.00	98.22	100.00
0.00	2.50	0.00	2.96	0.00	95.27	100.00
0.00	3.00	0.00	3.55	0.00	91.72	100.00
0.00	4.00	0.00	4.73	0.02	86.99	99.98
0.03	5.97	0.03	7.07	0.18	79.92	99.79
0.19	9.84	0.16	11.65	1.03	68.27	98.77
0.48	9.61	0.39	11.37	2.52	56.90	96.25
0.92	9.25	0.75	10.94	4.86	45.95	91.39
1.34	7.91	1.09	9.36	7.05	36.60	84.34
1.69	6.62	1.38	7.83	8.91	28.76	75.43
1.91	5.44	1.56	6.44	10.07	22.33	65.36
1.65	3.65	1.35	4.32	8.68	18.00	56.67
1.50	2.77	1.23	3.28	7.93	14.72	48.74
9.24	12.44	7.56	14.72	48.74	0.00	0.00
	84.49	15.51	100.00	100.00		

APPENDIX O

HYDROCYCLONE STABILITY TESTING (SURVEYS NO. 5 AND 6)

AT LES MINES SELBAIE

Hydrocyclone Stability Testing (Surveys no. 5 and 6) at Les Mines Selbaie

Test Descriptions

Two sets of samples were gathered on July 9, 1986, to determine the effect that cyclone surging has on the ball mill circuit, survey no. 5, while the cyclones were surging, and survey no. 6, while they were not surging. It was previously determined that cyclone surging is associated with a low level of slurry in the cyclone feed pump box.

Before taking the samples, the surging "cycle" from loss of cyclone overflow with pressure loss, followed by return of pressure, apparent normal operation, to loss of cyclone overflow once again, was timed with a stopwatch through a number of cycles, as follows.

1st cycle	42 seconds
2nd cycle	38 seconds
3rd cycle	44 seconds
4th cycle	47 seconds
5th cycle	36 seconds

Based on an average cycle time of approximately 40 seconds, a series of cuts were taken of each of the cyclone streams at equal time intervals using a hand cutter through one or more cycle periods. Following the first set of cuts, the surging cycle was retimed, and the second set of cuts taken. The process was repeated for the third set of cuts, as follows.

Table O-1. Sampling Sequence during Cyclone Surging (Survey No. 5)

Cut Series No.	Cycle Period Range (Average)	No. Cuts and Time: Feed	Overflow	Underflow
1	36-47 seconds (40)	20 cuts / 80 sec.	10 cuts / 80 sec.	24 cuts / 40 sec.
2	28-40 seconds (30)	20 cuts / 60 sec.	10 cuts / 60 sec.	20 cuts / 30 sec.
3	24-44 seconds (30)	20 cuts / 60 sec.	10 cuts / 60 sec.	20 cuts / 30 sec.

Cyclone feed samples were obtained from the bypass line that had been set up for rod mill fines addition tests. The combined underflow was taken at the ball mill feed box (cyclone underflow water was turned off), and separate overflow samples were collected at each of the operating cyclone overflow discharge pipes (units no. 2 and 3). Note that to obtain a suitable "weighted mean" sample, the cutter was passed at even time intervals, even when the flow had stopped.

At the end of the above sample collection period, a series of 8 separate underflow samples (one cut each) were taken in sequence through a 62 second period.

Cyclone surging was then eliminated by adjustment (slowing) of the speed of the cyclone feed pump, and the circuit was allowed approximately one hour to stabilize. Cyclone feed pressure was constant at approximately 55 kpa (8 psi). Combined samples of 20 cuts at the cyclone feed and 10 cuts each of the underflow and overflows were taken during survey no. 6.

Note that the crushing plant was off, and the rod mill feed rate setting was unchanged throughout the shift. The time and weightometer readings taken throughout the tests were as follows.

Test	Time Start / Finish	Weightometer Start / Finish
Survey No. 5	1:09:00/2:38:00	24385.70/24481.90
Survey No. 6	3:31:00/4:19:00	24539.77/24593.17

Sample Analyses

Moisture analyses performed on all the samples collected during the tests are reported in Table B14-2. Screen analyses were also performed on all the primary samples from the two test periods, as well as the samples which represented the extremes in percent solids for the cyclone underflow series (no. 5 and 8), as shown in Tables B14-3, 4, and 5.

Table O-2. Moisture Analyses, Surveys No. 5 and 6

Sample		% Solids by Weight
Survey No. 5	Cyclone feed	62.9%
	Cyclone underflow	72.4
	#2 Cyclone overflow	46.6
	#3 Cyclone overflow	43.0
	Underflow series cut no. 1	76.8
	2	73.9
	3	63.8
	4	75.8
	5	77.6
	6	77.1
	7	68.2
	8	63.5
Survey No. 6	Cyclone feed	65.6
	Cyclone underflow	75.7
	Cyclone overflow (combined)	42.7

Table O-3. General Size Distribution Data, Survey No. 5

Screen Size (um)	Cyclone Feed		Cyclone Underflow		#2 Cyclone Overflow		#3 Cyclone Overflow	
	Ind.%	Cum.% Pass.	Ind.%	Cum.% Pass.	Ind.%	Cum.% Pass.	Ind.%	Cum.% Pass.
3350	0	100	0	100				
2360	0.99	99.01	1.17	98.82				
1700	1.16	98.85	2.03	96.79				
1180	2.93	94.92	3.84	92.96	0	100	0	100
850	3.92	91.01	5.15	87.81	0.05	99.95	0.05	99.95
600	5.25	85.76	6.91	80.90	0.10	99.85	0.10	99.85
425	6.93	78.83	8.58	72.33	0.45	99.40	0.30	99.55
300	8.30	70.53	9.98	62.35	1.24	98.16	0.76	98.79
212	8.30	62.22	10.02	52.33	3.08	95.08	1.92	96.87
150	11.49	50.73	12.19	40.14	7.99	87.09	5.60	91.28
106	9.42	41.31	9.21	30.93	10.03	77.06	8.32	82.96
75	7.87	33.43	7.04	23.88	10.23	66.83	9.13	73.83
53	5.34	28.10	4.51	19.37	7.80	59.04	7.82	66.01
38	6.11	21.99	4.24	15.12	10.82	48.21	11.35	54.66
- 38	21.99		15.12		48.21		54.66	
Total	100.0		100.0		100.0		100.0	

Table O-4. Underflow Size Distribution Variations

Survey No. 5

Screen Size (um)	Underflow Series Cut: No. 5 (77.6% solids) Ind.%	Cum.% Pass.	No. 8 (63.5% solids) Ind.%	Cum.% Pass.
3350	0	100	0	100
2360	2.04	97.96	0.33	99.67
1700	3.14	94.82	0.81	98.86
1180	6.13	88.69	3.04	95.82
850	7.03	81.65	5.03	90.79
600	8.27	73.38	6.36	84.43
425	9.65	63.74	7.22	77.21
300	10.17	53.56	7.83	69.37
212	9.17	44.39	8.07	61.30
150	11.83	32.56	12.01	49.29
106	8.84	23.72	9.50	39.79
75	6.32	17.40	7.88	31.91
53	3.90	13.50	5.13	26.78
38	3.61	9.89	6.03	20.75
- 38	9.89		20.75	
Total	100.00		100.00	

Table O-5. General Size Distribution Data
Survey No. 6

Screen Size(um)	Cyclone Feed Ind. %	Cum.% Pass.	Cyclone Underflow Ind. %	Cum.% Pass.	Cyclone Overflow Ind. %	Cum.% Pass.
3350	0	100	0	100		
2360	1.45	98.55	1.35	98.65		
1700	2.32	96.23	2.34	96.31	0	100
1180	4.29	91.94	4.52	91.79	0.05	99.95
850	4.75	87.19	5.82	85.97	0.05	99.90
600	5.84	81.35	7.38	78.60	0.05	99.84
425	6.87	74.48	8.88	69.71	0.11	99.73
300	8.01	66.47	10.18	59.53	0.32	99.41
212	8.11	58.37	10.18	49.35	1.17	98.24
150	11.31	47.06	13.35	36.00	5.54	92.70
106	8.83	38.22	9.19	26.81	8.96	83.74
75	7.02	31.20	6.65	20.16	9.33	74.41
53	4.80	26.39	4.31	15.84	7.68	66.74
38	5.22	21.18	4.05	11.79	11.35	55.38
- 38	21.18		11.79		55.38	
Total	100.00		100.00		100.00	

APPENDIX P

PRIMARY HYDROCYCLONE UNDERFLOW SAMPLING AT KIDD CREEK

Primary Hydrocyclone Underflow Sampling at Kidd Creek

A series of underflow samples were taken on August 12, 1987, on the "C" circuit primary hydrocyclones to check for consistency of the densities and size distributions from different units on the cluster arrangement. This circuit was chosen because the "B" circuit cyclones were badly surging at the time. The samples were taken using approximately 4 cuts each with a hand sampler. The densities and size distributions obtained on the five circuits that were operating at the time are given in Tables P-1 and 2. Samples were split, wet screened at 75 um, and then dry screened 20 minutes to 75 um. A specific gravity test was also performed on the sample from cyclone no. 8, giving a value of 3.40.

Table P-1. Hydrocyclone Underflow Dry Sample Weights and Percent Solids

Cyclone No.	Sample Dry Weight (gms)	% Solids by Weight
1	1469.1	81.9
2	1191.2	80.3
4	1088.1	81.3
6	838.3	80.0
8	1403.2	80.9

Table P-2. Hydrocyclone Underflow Size Distributions (Cummulative Percent Passing)

Screen Size (um)	Unit No.1	Unit No.2	Unit No.3	Unit No.4	Unit No.5
3,350	98.8	98.8	99.2	99.7	99.2
2,360	96.5	96.3	97.6	98.9	97.7
1,700	92.9	93.0	95.0	96.6	95.3
1,180	87.5	87.8	90.1	92.4	90.7
850	83.0	83.4	85.7	88.2	86.4
600	77.3	77.7	79.8	82.3	80.7
425	69.8	70.2	71.9	74.1	72.8
300	61.9	62.2	63.6	65.3	64.3
212	52.5	53.0	53.8	55.1	54.2
150	42.4	43.2	43.3	44.4	43.6
100	32.1	32.9	32.4	33.4	32.9
75	22.1	23.1	22.2	22.9	22.7