A Simplified Economic Filter for Underground Mining of Massive Sulfide Deposits

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INTRODUCTION

In resource assessments of undiscovered mineral deposits and in the early stages of exploration, including planning, a need for prefeasibility cost models exists. In exploration, these models that separate economic from uneconomic deposits help to focus on targets that can benefit the exploration enterprise. In resource assessment, these models can be used to eliminate deposits that would probably be uneconomic even if discovered. The U. S. Bureau of Mines (USBM) previously developed simplified cost models for such problems (Camm, 1991). These cost models estimate operating and capital expenditures for a mineral deposit given its tonnage, grade, and depth. These cost models were also incorporated in USBM prefeasibility software (Smith, 1991).

Because the cost data used to estimate operating and capital costs in these models are now over ten years old, we decided that it was necessary to test these equations with more current data. We limited this study to underground mines in massive sulfide deposits.

In a previous study (Singer et al., 1998), we modified the simplified cost models for open-pit U.S. gold-silver deposit operations to reflect higher capacities observed in heap-leach processing with autoclave, carbon-in-leach (CIL), carbon-in-pulp (CIP), and Merrill Crowe mills. For heap-leach operations, we also modified equations for estimating operating cost and capital expenditure. Explanations of these various processing methods are available in Camm (1991).

For underground mining of massive sulfide deposits using each of five different mining methods, we compare capacity and cost estimates using the USBM models with observed mines. If significant differences exist between the observed costs and those predicted by the USBM model, we modify the equations appropriately.

NATURE OF DATA

Deposits used in this study include underground mines which were operated by some combination of room and pillar, cut and fill, crater retreat, shrinkage stope, or sublevel longhole mining methods. The material was processed through mills designed to handle one to three products. Twelve of the mines are in kuroko-type deposits; eight are in Mississippi Valley-type deposits; sedimentary exhalative, skarn copper, and polymetallic replacement-type deposits are represented by two mines each; one mine is in a polymetallic vein-type deposit; and one does not fit a recognized type. These deposits are located in the United States, Canada, Australia, Portugal, Spain, Chile, and Greenland. Data sources were files purchased from D. Briggs (Briggs, 1994).

A total of 28 economic deposits were used to estimate the parameters of the model equations. Determination of whether a known deposit was economic was based on having reported a profit in more than 70 percent of the years operated. In cases where only the first 2-3 years of an operation were reported, this rule was relaxed to allow for commonly reported first year losses. Even so, this scheme is not perfect in that a mine could have small losses in a third of its production years and yet have a large profit over the life of the mine. In this data one mine appeared to be profitable until it closed but left a 10 million dollar loan unpaid—it was treated as profitable here, but could only properly be classified after a careful financial analysis.

Operating costs are most commonly reported as total operating costs in dollars per troy ounce. In order to be compatible with estimates by the USBM method, the mine and mill operating costs made by the USBM method were added to represent total operating costs. Operating cost estimates from the observed mine data were adjusted to dollars per ton.

Capital expenditures used here represent the total reported over the life of a mine. Frequently mining operations are observed to spread out their capital expenditures by means of mine or mill expansions over a period of years. The simplified nature of the economic analysis used assumes that capital expenditures are made at the beginning of the first year and that mining/mill capacities remain constant until the deposit is depleted.

CAPACITY AND MINE LIFE

All cost estimates in the USBM method are derived from the estimated daily mining capacity or its estimated mine life. Because of this, unbiased estimates of daily mining capacities are critical. In our earlier study of open-pit gold-silver mining, we found that capacities were significantly different than predicted in the standard equation.

In Camm's (1991) report, daily mining capacities are calculated using Taylor's rule (Taylor, 1978) from the total amount of ore in the deposit as:

$$C = [T^{0.75}] / 70 \qquad (1)$$

where C is capacity in short tons per day, T is resource tonnage in short tons, and 350 operating days per year are assumed. Tonnage is modified from published reserves to account for recovery and dilution which vary from mining method to mining method and, to a lessor extent, mine to mine. The adjustment factors we use (table 1) are the same as Camm's (1991).

Table 1—Mine dilution and recovery factors.

Mining method	Dilution factor %	Recovery factor %
Cut and fill	5	85
Room and pillar	5	85
Shrinkage stope	10	90
Sublevel longhole	15	85
Vertical crater retreat	10	90

Analysis of daily capacity and ore tonnages in 28 known economic deposits shows a relationship similar to equation 1 (fig. 1). Using a "t" test, the regression slope (0.704) is not significantly different than Taylor's (see Appendix). The regression intercept (0.0248) also is not significantly different than Taylor's (see Appendix). Therefore, we use Taylor's rule to estimate capacity.

The USBM estimated mine life in years (L) is based on Taylor's rule as:

$$L = 0.2 \bullet T^{0.25} \qquad (2)$$

The direct relationship between mine life and ore tonnage could not be determined from the present data because too few mines were depleted.

However, life can be estimated from daily capacity estimated in Equation 1 and ore tonnage as:

$$L = T / (C \cdot 350) \quad (3)$$

assuming 350 operating days per year.

CAPITAL EXPENDITURES

Available data on the known underground mines on massive sulfide deposits did not allow consistent separation of mine, shaft, and mill capital expenditures, so here we combine them. The total capital expenditures of underground mining operations of massive sulfide deposits are estimated as the sum of the estimated capital expenditures for the mine, the mill, and shaft(s), if required. The USBM equations for the capital cost of an underground mine (Camm, 1991) depend on the mining method:

$$\text{For room and pillar the capital expenditure} = 97{,}600 \cdot C^{0.644} \quad (4)$$

$$\text{For cut and fill the capital expenditure} = 1{,}250{,}000 \cdot C^{0.461} \quad (5)$$

$$\text{For vertical crater retreat the capital expenditure} = 42{,}200 \cdot C^{0.747} \quad (6)$$

$$\text{For shrinkage stope the capital expenditure} = 179{,}000 \cdot C^{0.620} \quad (7)$$

$$\text{For sublevel longhole the capital expenditure} = 115{,}000 \cdot C^{0.552} \quad (8)$$

where C is capacity in short tons per day (st/d) as above.

In about half of the mines, more than one mining method was used—in these cases, the approximate percent of ore mined by each method in the mine was used to weigh the estimated capital expenditure.

In each cost estimate of these mining methods, it was assumed that there is an adit entry. Mines having a shaft entry need to have the additional capital cost of the shaft added. The capital cost equation for a shaft from Camm (1991) is:

$$\text{Shaft cost} = 371 \bullet C + 180 \bullet D \bullet C^{0.404} \qquad (9)$$

where C is capacity of mine in st/d, and D is depth of the shaft to the bottom of ore in feet.

Each of the mines on massive sulfide deposits requires a flotation mill to produce one, two, or three concentrate products. In general, the more products, the greater the capital and operating expense. The capital cost of the mills according to Camm (1991) can be estimated as:

$$\text{One product mill} = 92{,}600 \bullet C^{0.667} \qquad (10)$$

$$\text{A two product mill} = 82{,}500 \bullet C^{0.702} \qquad (11)$$

$$\text{And, a three product mill} = 83{,}600 \bullet C^{0.708} \qquad (12)$$

where C is again capacity of mine in st/d.

The total capital expenditures of 16 of the known mines are available for analysis. Using the ore tonnages, mining methods, depth of shafts, and number of concentrate products reported for the 16 mines, the capital expenditures of mines, mills and shafts were estimated using the above equations and plotted against the observed capital expenditures (fig. 2). If the equations reported in Camm (1991) are no longer valid, the regression slope would be different than 1.0. The slope is not significantly difference than 1 when tested using a "t" test. Observed capital costs are not significantly different than the total capital expenditures estimated in the original USBM cost models (Camm, 1991). Observed capital expenditures are in current dollars.

OPERATING COSTS

Like capital expenditures, available data on the known underground mines on massive sulfide deposits did not allow consistent separation of mine, shaft, and mill operating costs, so here we combine them. The total operating costs of underground mining operations of massive sulfide deposits are estimated as the sum of the estimated operating costs for the mine, the mill, and shaft(s), if required. The USBM equations for the operating cost of an underground mine (Camm, 1991) depend on the mining method:

For room and pillar the operating cost = $35.5 \bullet C^{-0.171}$ (13)

For cut and fill the operating cost = $279 \bullet C^{-0.284}$ (14)

For vertical crater retreat the operating cost = $51.0 \bullet C^{-0.206}$ (15)

For shrinkage stope the operating cost = $74.9 \cdot C^{-0.100}$ (16)

For sublevel longhole the operating cost = $41.9 \cdot C^{-0.181}$ (17)

where C is capacity in short tons per day as above.

Mines having a shaft entry need to have the additional operating cost of the shaft added. The operating cost equation for a shaft from Camm (1991) is:

Shaft cost = $2{,}343 / C + 0.440 \cdot D / C + 0.00163 \cdot D$ (18)

where C is capacity of mine in st/d, and D is depth of the shaft to the bottom of ore in feet.

The mills produce one, two, or three concentrate products with concomitant greater operating expense of each. The operating cost of the mills according to Camm (1991) is estimated as:

One product mill = $121 \cdot C^{-0.335}$ (19)

A two product mill = $149 \cdot C^{-0.336}$ (20)

And, a three product mill = $153 \cdot C^{-0.344}$ (21)

where C is again capacity of mine in st/d.

The total operating costs of 13 of the known mines are available for analysis. Using the ore tonnages, mining methods, depth of shafts, and number of concentrate products reported for the 13 mines, the operating cost of mines, mills and shafts were estimated using the above equations and plotted against the observed capital expenditures (fig. 3). If the equations reported in Camm (1991) are no longer valid, the regression slope would be different than 1.0 and the intercept would be different than 0.0 indicating a different equation is needed. The slope is not significantly different than 1 when tested using a "t" test and the intercept is not different than 0.0 (see Appendix). Therefore, observed operating costs are not significantly different than the total operating cost estimated in the original U.S. Bureau of Mines cost models (Camm, 1991). Observed operating costs are in current dollars.

ECONOMIC FILTER

Given an appropriate mining method and depth of a deposit, the deposit's tonnage is all that is needed to estimate various mining costs using the equations above. The deposit's grade(s) can, when combined with assumed copper, zinc, lead, gold and silver prices, be used to estimate the deposit's ore value per ton. Value of production per year can be calculated by multiplying the difference between value per ton and total cost per ton by capacity per day times number of operating day per year (350 days used here).

The life of the mine estimate is then used with the value of production per year and an acceptable rate of return (10 percent used here) in a standard present-value equation in a spreadsheet to estimate a deposit's present-value of production. The present-value of production minus the estimated capital expenditure for the deposit is the present-value of the deposit. If the deposit's present-value is positive, the filter is predicting that the mine is profitable. Negative present-values predict economic failure at the assumed metal prices and rate of return.

For a particular tonnage, the dividing (or break even) line between economic and uneconomic can be estimated by adding the estimated operating cost to the capital expenditure divided by capacity times operating days per year times the present-value of a dollar for the life of the mine. That is:

$$BE = TOC + MOC / (350 \bullet C \bullet PV) \qquad (22)$$

where BE is the break-even value ($/st), TOC is total operating cost ($/st), MOC is the mine operating cost ($/st), operating at 350 days per year, C is capacity (st/d), and PV is the present-value of one dollar at the selected rate of return and life of the mine in years. The break-even value could be viewed as the grade (expressed in $/ton) at which the specific deposit and mining method are just economic. To account for variabilities and uncertainties in most of the inputs to these estimates, we have taken 0.7 and 1.3 of this break-even value to estimate boundaries for uneconomic, marginal, and economic deposits (fig. 4).

In order to see how well these cost equations estimate economic viability, we have plotted on figure 4 the 28 deposits used to test the equations. The plot demonstrates that the equations are performing well. One of the mines plots just below the break-even line in the non-economic region—the same mine that was mentioned above in the Nature of Data section as being difficult to classify economically because it had a 10 million dollar unpaid loan.

The line connecting the break-even points (and associated lines) bounces up and down when plotted against tonnage (Fig. 4) principally because of the different mining methods used in the mines. Weaker effects are due to costs associated with shafts. Effects of the varying number of mill products are minor. In our study of open-pit gold-silver mines in the U.S. (Singer and others, 1998), the break-even lines were a smoothly decreasing function of tonnage because only one mining method was represented.

Selection of mining method mainly depends on the attitude and shape of the deposit and wall rock strength. Of course, ore tonnage and grade also affect mining method selection in that very low grades can not support mining methods that have high operating expenses and low tonnage deposits can not be economically mined by methods requiring large capital expenditures. These factors affecting choice of mining method are related in part to type of mineral deposit. For example, 95 percent of the ore mined in the eight in Mississippi Valley-type deposits was mined on average by the room and pillar method—the remaining 5 percent was divided evenly between the cut and fill and the shrinkage stope methods. Multiple mining methods are much more common in kuroko massive sulfide deposits. On average, 35 percent of the ore mined from the 12 kuroko deposits is with shrinkage stope, 32 percent with sublevel longhole, 26 percent cut and fill, and 3 and 4 percent with room and pillar and crater retreat methods respectively. Based on only two sedimentary exhalative zinc-lead mines, 95 percent of the ore mined is mined by the room and pillar method and 5 percent by crater retreat methods. Also based on two mines, 75 percent of the ore mined in polymetallic replacement deposit is by room and pillar methods and 25 percent is by cut and fill. Fifty percent of the ore mined in two copper skarn mines is by the room and pillar method, 25 percent is by cut and fill, and 25 percent is by shrinkage stope methods.

SUMMARY

In a previous study (Singer et al., 1998), we modified the simplified cost models for open-pit U.S. gold-silver deposit operations to reflect higher capacities observed and modified equations for estimating operating cost and capital expenditure. Based on analysis of the economic relationships in mines on 28 massive sulfide deposits in this study, we found no reason to reject the simplified cost models for underground mining operations presented by Camm (1991). The deposits represent at least six different deposit types and are located in seven different countries. For cut and fill, room and pillar, crater retreat, shrinkage stope, and sublevel longhole mining, with or without shafts, the equations for estimating operating cost and capital expenditure are consistent with known operations. One marginally economic mine was estimated by these equations to be marginally economic. The resultant equations appear to provide reasonable estimates of costs, but all such estimates can be wrong because of factors such as poor metal recovery or errors in estimated future metal prices.

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APPENDIX:

Use of the "t" statistic to test differences between the regression slope and intercept predicted by Taylor and those observed in 28 mines in a regression of ore tonnage on daily mine capacity.

Taylor's equation (Eq. 1) is: $C= [T^{0.75}] / 70$

In log form the equation is: $\log_{10} C = 0.75 \cdot \log_{10} T - 1.845$

The observed regression equation is: $\log_{10} C = 0.704 \cdot \log_{10} T - 1.606$

The observed standard errors of the slope and intercept are 0.042 and 0.299 respectively.

For the test of slope differences, "t" = (0.75 – 0.704) / 0.042 = 1.1 with 26 degrees of freedom. The probability of obtaining a "t" value of 1.1 with 26 degrees of freedom by chance is less than 0.01. In other words, the hypothesis that the two slopes are significantly different is rejected.

For the test of intercept differences, "t" = (-1.845 - -1.606) / 0.299 = -0.8 with 26 degrees of freedom. The probability of obtaining a "t" value of 0.8 with 26 degrees of freedom by chance is less than 0.01. In other words, the hypothesis that the two intercepts are significantly different is rejected.

Use of the "t" statistic to test if the observed operating costs are significantly different than the total operating cost estimated in the original U.S. Bureau of Mines cost models (Camm, 1991). The regression equation obtained based on 13 mines is:

$\log_{10}$ OperatingCostobserved = 0.437 + 0.69 • $\log_{10}$ OperatingCostpredicted

where the standard errors of the intercept and slope are 0.396 and 0.262 respectively.

For the test of slope differences, "t" = (1.0– 0.69) / 0.262 = 1.1 with 11 degrees of freedom. The probability of obtaining a "t" value of 1.1 with 11 degrees of freedom by chance is less than 0.05. In other words, the hypothesis that the two slopes are significantly different is rejected.

For the test of intercept differences, "t" = (0.0 – 0.437) / 0.396 = -1.1 with 11 degrees of freedom. The probability of obtaining a "t" value of 1.1 with 11 degrees of freedom by chance is less than 0.05. In other words, the hypothesis that the two intercepts are significantly different is rejected.

The slope is not significantly different than 1 when tested using a "t" test and the intercept is not different than 0.0. Therefore, observed operating costs are not significantly different than the total operating cost estimated in the original U.S. Bureau of Mines cost models (Camm, 1991).

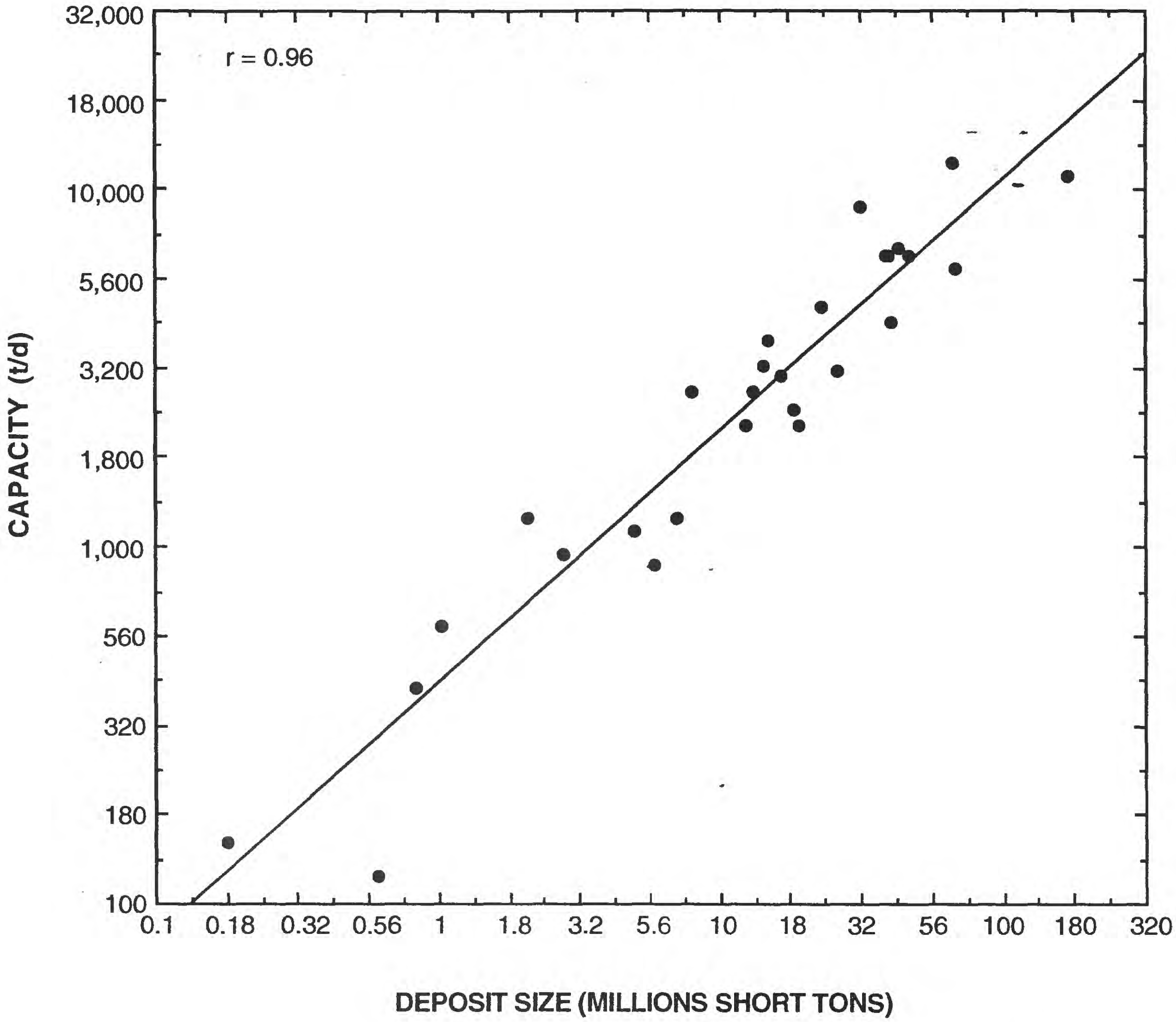

Figure 1--Relationship between mine capacity and deposit size (tons of ore) of underground massive sulfide deposits

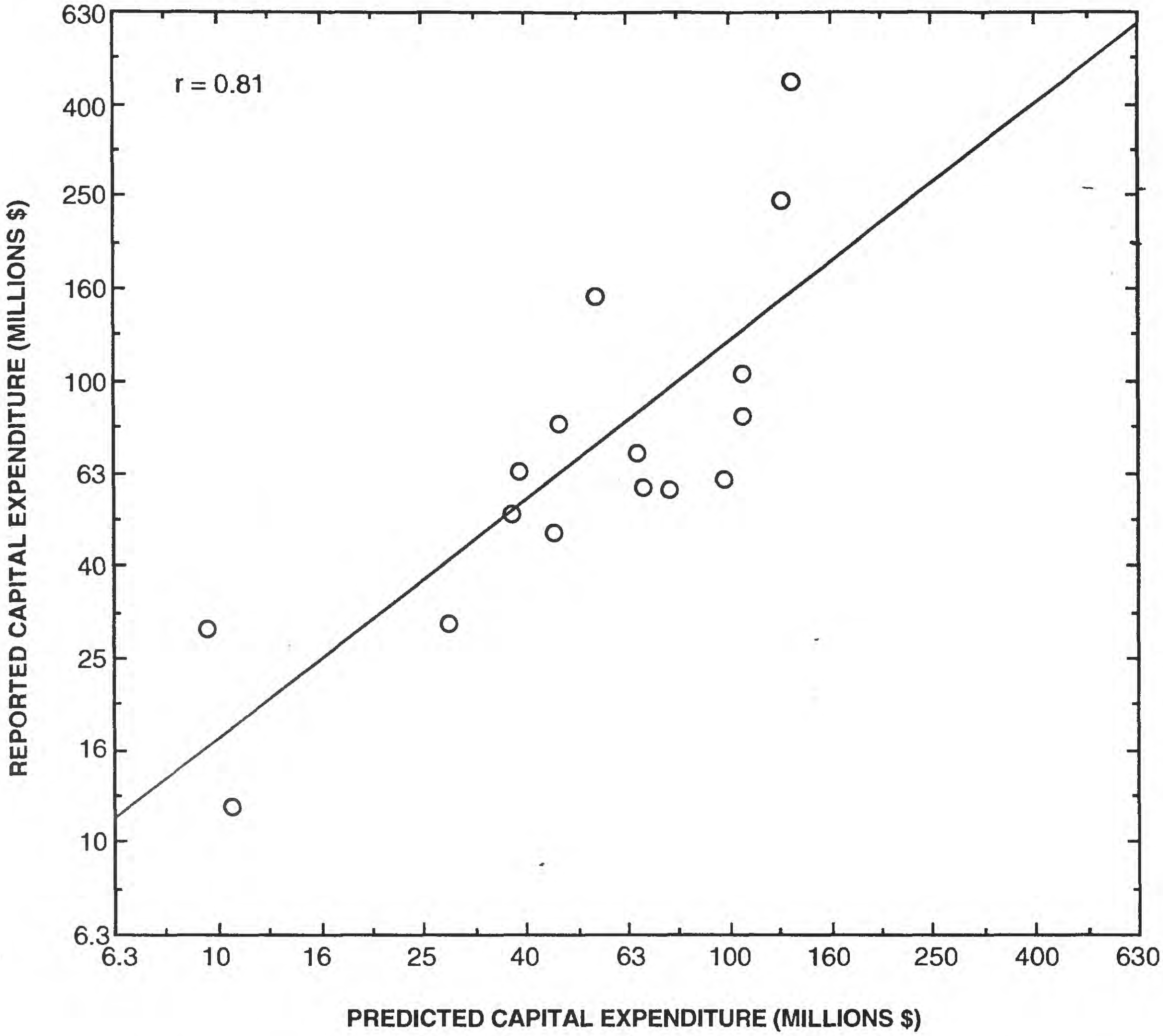

Figure 2-- Relationship between total capital expenditure reported and total capital expenditure predicted for underground massive sulfide deposits.

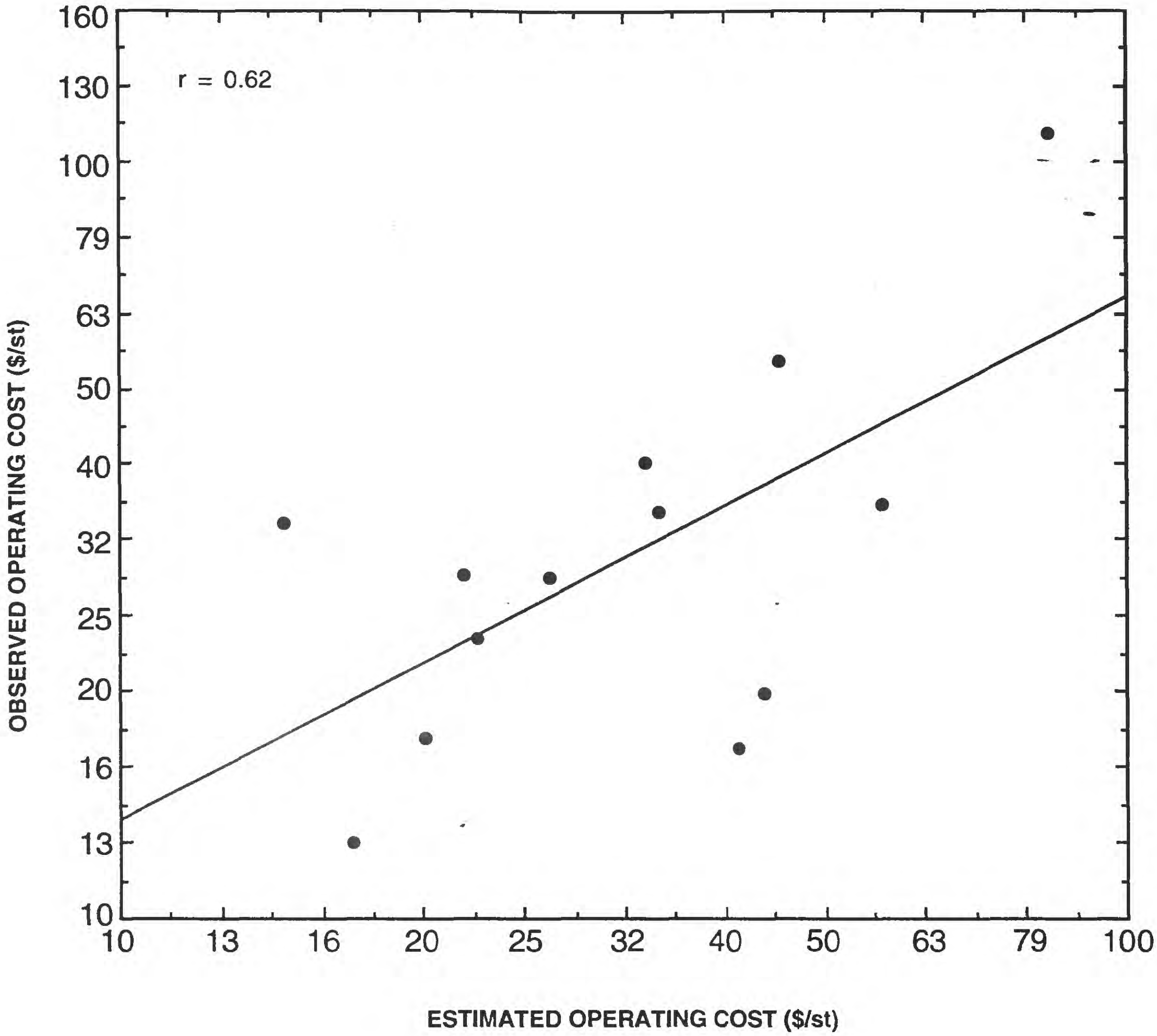

Figure 3-- Relationship between operating expenditure reported and operating expenditure predicted for underground massive sulfide deposits.

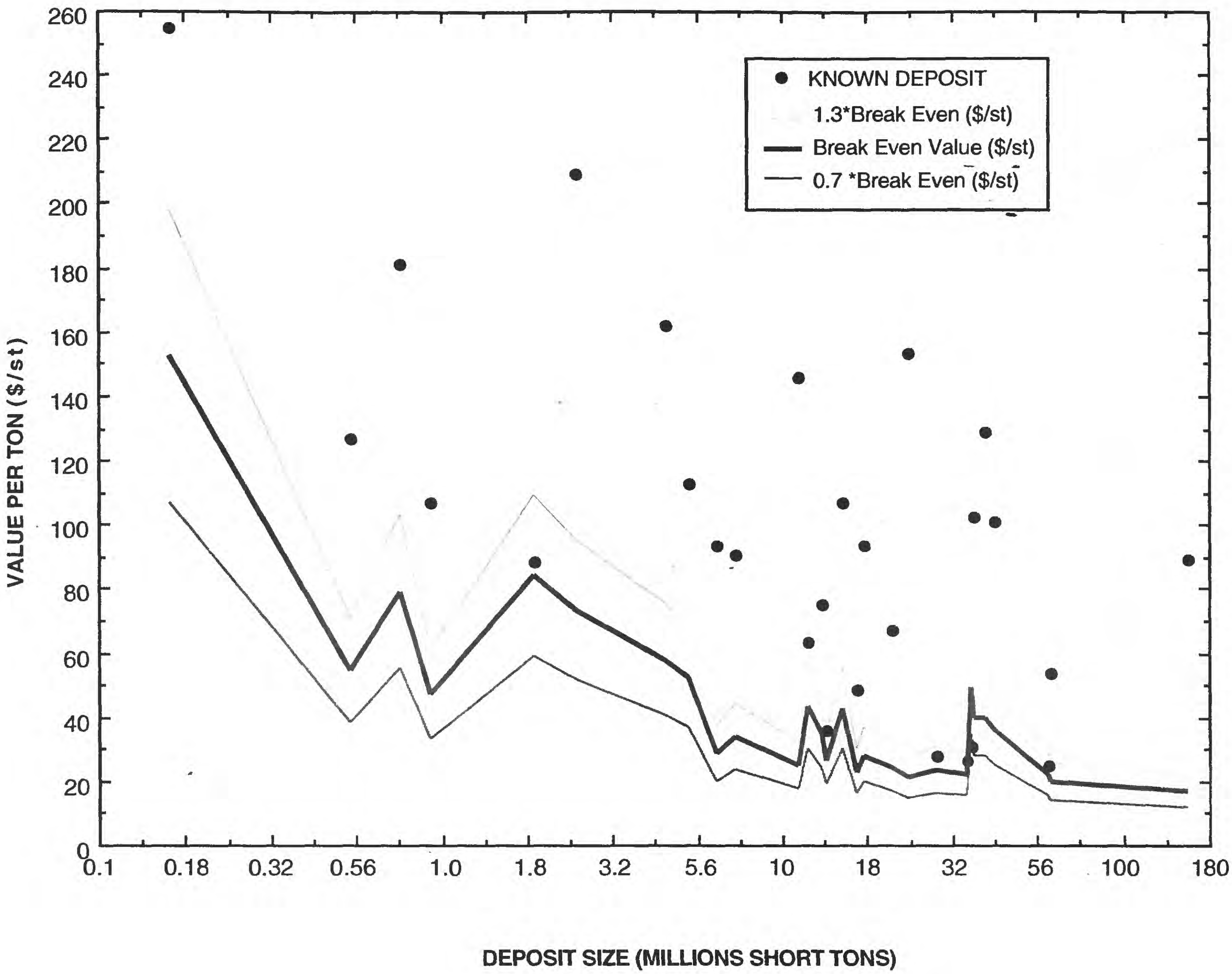

Figure 4-- Relationship between deposit size (tons of ore) and value per short ton for underground massive sulfide deposits. Also shown are associated economic filters of the break-even value and 0.7 and 1.3 of the break-even value based on a rate of return of 10% and a copper price of $0.75/lb., zinc price of $0.5/lb., lead price of $0.25/lb., silver price of $5/oz, and gold price of $260/oz .