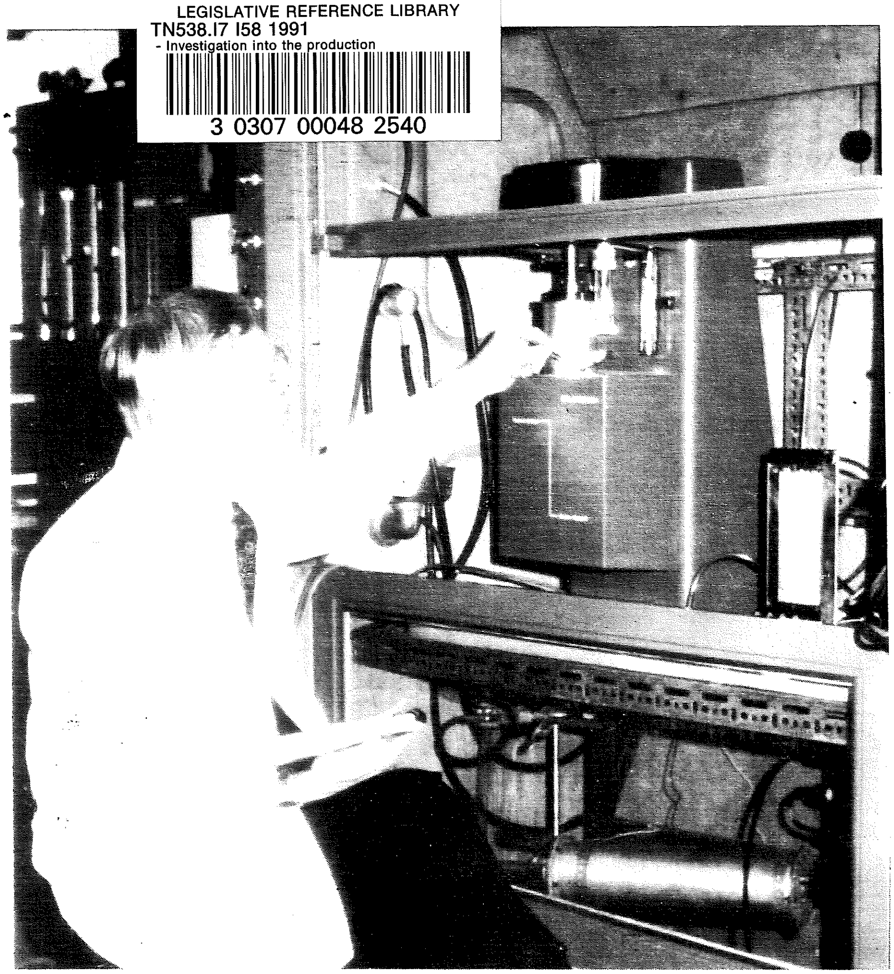

REPORT 1001
MINERAL RESEARCH SERIES

INVESTIGATION INTO THE PRODUCTION OF IRON ORE CONCENTRATES WITH LESS THAN 3 PERCENT SILICA FROM MINNESOTA TACONITES

EXECUTIVE SUMMARY OF BULK CONCENTRATE UPGRADING

A COLLABORATIVE PROJECT
AMERICAN IRON AND STEEL INSTITUTE
AND
STATE OF MINNESOTA

NOVEMBER 1991

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Minerals Research Series

November 1991

Principal Investigators:
R. L. Bleifuss, Natural Resources Research Institute
I. Iwasaki, Mineral Resources Research Center

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ABSTRACT

The investigation into the production of low-silica concentrates with an ultimate goal of producing ultra-low (less than 2 percent) silica concentrates from Minnesota taconite was stimulated by the current status of iron-making research world-wide. It was felt that many of the new processes currently being developed might require or be helped by the availability of a consistent quality ore resource that contained significantly less gangue material than could be produced with current technology. By collaborating, the sponsors were able to create a project that studied a variety of techniques on several typical Minnesota ores.

The major components of the overall project were the following:

- Detailed mineralogical analyses of Minnesota concentrates
- Bulk concentrate upgrading
- Magnetite depressant reagent development
- Intermediate stream processing

This executive summary covers the topic of bulk concentrate upgrading. Three plant concentrates were studied at both bench and pilot scales using various flotation and classification methods to achieve final concentrates containing less than 2 percent silica. In all three cases several flowsheets were developed which appear to give the desired result. The options which include matrix magnetic separation and/or column flotation appear particularly attractive.

INVESTIGATION INTO THE PRODUCTION OF IRON ORE CONCENTRATES WITH LESS THAN 3 PERCENT SILICA FROM MINNESOTA TACONITES

EXECUTIVE SUMMARY
BULK CONCENTRATE UPGRADING

INTRODUCTION

This document summarizes the research on bulk concentrate upgrading conducted at the Natural Resources Research Institute/ Coleraine Research Facility (NRRI/CMRL). Three taconite concentrates, representative of materials produced in Minnesota, were tested in laboratory and pilot scale equipment. The plants sampled were LTV Steel Mining (LTV), USS's Minnesota Ore Operations (Minntac), and Hibbing Taconite (Hibtac). The goal was to produce final concentrates containing less than 2 percent silica from each of the plant samples. Several technically feasible flowsheets were constructed for each plant. This report contains an overview of the results obtained for each concentrate and material relating to the project's origin and management. Detailed results are to be found in the individual plant reports.

BACKGROUND

The silica levels in Minnesota taconite concentrates, while acceptable for blast furnaces in the United States, are too high for other uses such as the production of direct reduced iron (DRI). Also, the desires of steel company customers for lower slag volumes has generated a need for higher-grade concentrates, and the new direct smelting technologies currently being researched may require very pure ores. These factors created a need for a research effort aimed at production of concentrates with the highest possible purity levels.

In 1987 the State of Minnesota approached the participants in the American Iron and Steel Institute (AISI) direct smelting project with a proposal to add iron ore to their list of research topics. The companies felt that the type of work proposed would be valuable but that it did not fit into the project as defined by the industry and the U. S. Department of Energy. With this encouragement, the Minnesota group sought funding from the state legislature with an expectation that several steel companies would participate in a project independent of the work on direct smelting.

In mid-1988 NRRI/CMRL submitted a proposal for research into the production of low-silica concentrate to the Legislative Commission on Minnesota Resources (LCMR). This proposal was supported by the Minnesota Department of Natural Resources (DNR) and the Minerals Resources Research Center at the University of Minnesota (MRRC). The LCMR agreed to fund the project contingent on receipt of matching funds from the Iron Range Resources and Rehabilitation Board (IRRRB). The IRRRB subsequently agreed to provide a match, so the total state funding was secure. With these funds in hand, the state approached the AISI with a proposal for a collaborative project.

The original proposal contained two phases. The work in Phase I would focus on three topics, i.e., developing a detailed characterization of the ores, magnetite depressant reagent development, and bulk concentrate upgrading. The Phase II work was to include pilot scale production of concentrates, using the results of Phase I, and metallurgical testing of pellets produced using ultra low-silica concentrate. After some discussion, it was decided the Phase I work would be conducted during the state's FY90-91 biennium, and any decision regarding continuation would await the Phase I results. Nine members of the AISI agreed to support this approach, and an agreement was executed between AISI and DNR. The state agreed to become the overall project manager and to work with the principal investigators at the university to complete the work plan as laid out in the contracts with AISI and the university.

The final work plan for Phase I consisted of the following items:

1. Detailed mineralogy of the Hibtac, Minntac and LTV concentrates using the QEM*SEM device located at MRRC.

2. Magnetite depressant reagent development

3. Concentrate fines, sand and slime treatment

4. Identification and upgrading of intermediate process streams

5. Bulk concentrate upgrading

    a. LTV concentrate
    b. Minntac concentrate
    c. Hibtac concentrate

This report summarizes the work included in item 5 above.

The documentation for this project will be contained in seven final reports. The final report on the detailed mineralogy was issued in September 1990. Draft final reports on bulk concentrate upgrading were distributed to project participants in August 1991, and the three remaining reports were delivered in September 1991, as individual research papers.

PROJECT WORK PLAN

Appendix A contains the detailed work plan which was incorporated into the agreement between AISI and DNR. The bulk concentrate portion of the work consisted of tasks outlined below. The work was repeated for each of the three concentrates with some variation based on prior test results. The final work plan as executed is outlined below:

1. Raw material characterization

    a. Collection of plant samples
    b. Size analysis
    c. Assay by size fraction

2. Assessment of pretreatment options at a laboratory scale

    a. Conventional magnetic separation
    b. Fine screening with a Derrick "K" screen
    c. Low intensity matrix magnetic separation
    d. Desliming and elutriation

3. Preliminary pilot plant program

    a. Laboratory flotation
    b. Pilot plant flotation in conventional cells

4. Pilot plant program

    a. Establish a baseline flowsheet
    b. Conventional rougher flotation
        i. Screening of cell product
            (1) Grinding of screen oversize
            (2) Separation of screen oversize
        ii. Grinding of rougher froths
        iii. Separation of rougher froths
    c. Column rougher flotation
        i. Screening of cell product
            (1) Grinding of screen oversize
            (2) Separation of screen oversize
        ii. Grinding of rougher froths
        iii. Separation of rougher froths

5. Derivation of alternative flowsheets

6. Estimation of major processing costs

BULK CONCENTRATE UPGRADING RESULTS

Raw Material Characterization

The results of the assays of the three bulk concentrates are shown in Table 1 below. In general, these results were anticipated based on prior knowledge of the plants, their production methods, and the ores used. The assays showed the overabundance of gangue material in the coarser fractions, i.e. +270 mesh, and the large amount of very pure (-500 mesh) material that the plants normally produce. One interesting aspect of the work performed is the indication of a lower limit on gangue at about 1.2 to 1.5 percent silica. Microscopic examination of finest fractions showed that small amounts of silica remain locked in particles that are essentially all magnetite. Therefore, achieving silica levels in the two percent range will bring the plants close to their ultimate purity levels.

Pretreatment Options

The baseline flowsheet as originally proposed started with a rougher flotation. However, the characterization work indicated that pretreatment might be used to achieve lower silica levels. The options tested were the following:

Conventional magnetic separation
Fine screening with a Derrick "K" screen
Low intensity matrix magnetic separation
Desliming and elutriation

The tests on all three concentrates indicated that pretreatment was not needed to achieve silica levels at or below 2 percent in the bulk concentrate. However, pretreatment might be an option if an extra 0.2 to 0.3 percent in silica grade became a critical objective.

Pretreatment based on matrix magnetic separation generated the most interesting results, and it appears to be the most promising of the options tested. The tests indicated that matrix separation might substantially reduce froth volumes in subsequent flotation thereby reducing the cost of froth treatment. It also appears that matrix separation might nearly duplicate the results of conventional flotation for limited silica reduction by producing a final concentrate containing about 4 percent silica.

Preliminary Pilot Plant Runs

The preliminary pilot plant program was conducted to establish material and energy balances for the baseline flowsheet and to evaluate alternative treatment options. The work started with a series of laboratory flotation tests using a procedure which has been shown to be highly reproducible. This procedure is described in detail in the final reports. In general, it consists of using a small batch cell and collection of froth samples at 30 second intervals during the first two minutes of operation with a final sample collected after three minutes.

The laboratory studies generated important results for the Minntac and Hibtac samples. In the first case, the work showed that even though the sample had been stored in a protected environment, it had aged to the point where it no longer responded well to silica flotation. Therefore, a new bulk sample was collected from the plant. The Hibtac bulk sample, which was delivered as filter cake, responded quite poorly to silica flotation. The initial work indicated that the response could be improved by remagnetizing the sample, so the bulk sample was remagnetized for all subsequent tests.

The work then progressed to the pilot plant for rougher flotation. The equipment consists of three banks of cells (Fig. 1). The first bank (Cell A) used two 0.75 cubic foot Wemco machines, the second bank (Cell B) used three 1.25 cubic foot Wemco machines, and the third bank (Cell C) contained two 0.75 cubic foot Wemco machines. Reagents could be added to the feed boxes of Cells A and B or to the air intake of Cell A.

For the conventional rougher flotation tests, the feed material was slurried in a 250-gallon mixing sump to between 45 and 50 percent solids by weight. Agitation was provided by recirculation of the slurry and a mixer. A portion of the recirculating slurry was sent to a small mixing tank where water was added to dilute the slurry to about 30 percent solids. The overflow from the mixing tank was the feed to the first flotation machine. The feed rate was adjusted to maintain an overflow rate of 20 lb/minute of dry solids. With this feed rate, the average residence time in the flotation machines was 7 minutes. The froths from Cells A, B, and C and the final concentrate were collected separately in 55-gallon drums. Each rougher flotation test was about 45 minutes in length. After 30 minutes of running, timed samples of all processed streams were taken. These samples were used to develop material balances for each test.

The collector used was Sherex MG83A in amounts ranging from 0.05 to 0.20 lb./ton of concentrate, and the frother was Sherex 139. The tests on Minntac concentrates indicated that two stage flotation yielded a lower silica cell product with better weight and iron recovery than single stage flotation. Therefore, two stage flotation was added to the pilot plant program. The data for these tests is shown in Table 1 below. Test A-10 is the combined results of Test A-9 and the reflotation of the A-9 cell product.

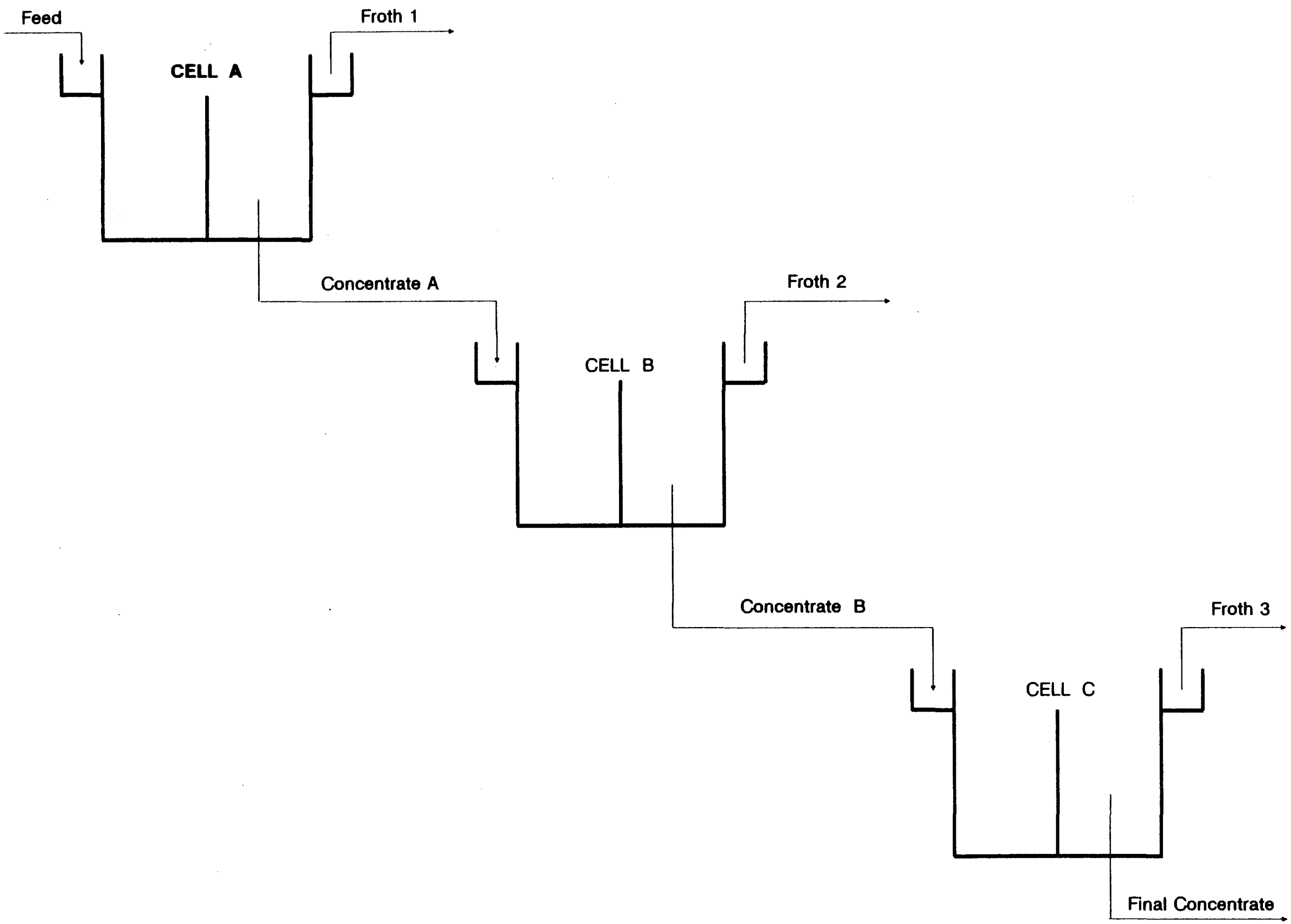


Figure 1. Schematic of Rougher Pilot Plant

Table 1: Preliminary Pilot Plant Flotation of Minntac Concentrate

Conditions	A-6	A-7	A-8	A-9	A-10
MG83A #/LT	0.206	0.205	0.194	0.150	0.213
Wt. Rec. %	59.7	51.8	51.2	77.8	56.3
Fe Rec. %	69.5	69.8	69.5	69.0	70.0
SiO2 %	2.78	2.42	2.49	3.05	2.27

The pilot plant work also showed that the LTV material was much more difficult to float. The data suggested that the froth material from the final cell was about the same grade as the cell product from Cell A. The results showed a very high iron loss in the -500 mesh fraction in the Cell C froth. This suggests that for the LTV samples the final froth should be retreated by column flotation to remove entrained fine iron.

The most unusual item noticed during the rougher flotation of the Hibtac concentrate was the impact of magnetization at the higher collector addition rates. When MG83A was added at a rate of 0.25 lb./LT, the filter cake responded with a 3.21 percent SiO2 grade and a 84.7 percent iron recovery. However, the magnetized material at the same addition rate yielded a silica grade of 2.40 percent with an 81.4 percent iron recovery.

None of the preliminary pilot plant runs yielded a rougher froth that could be thrown away. The assays of the cell products also produced strong evidence of the need for fine screening to achieve a 2 percent SiO2 final concentrate.

Pilot Plant Operation

The preliminary pilot plant runs provided sufficient information to develop baseline flowsheets for the three concentrates. The Minntac and Hibtac flowsheets are quite similar (Fig. 2). However, the baseline flowsheet for the LTV concentrate (Fig. 6) has several more steps. In all cases the baseline flowsheets yielded silica levels at or below 2 percent.

The various unit operations shown in the flowsheet were run independently, and in some of the alternatives generated, the whole flowsheet was not executed, as the results from prior runs could be projected into the current situation. In this way it was possible to generate several alternative flow sheets with a minimum of tests. The material and energy balances were created by combining the results of the unit operations tests.

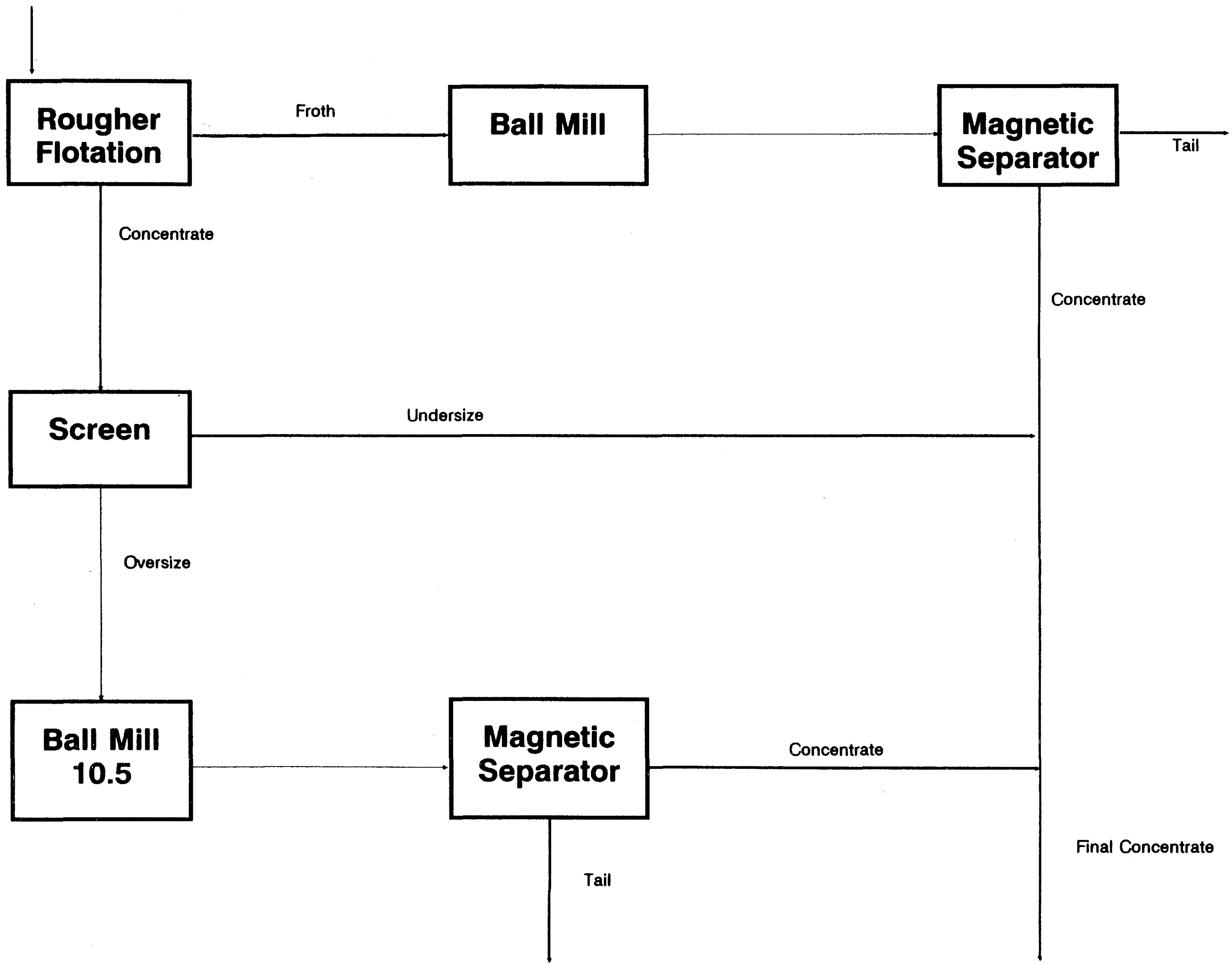
TABLE 2
COST ESTIMATION BASIS

ITEM	BASIS
Power	$0.04 /kwh
Collector	$1.20 /lb.
Frother	$0.60 /lb.
Grinding media	0.094 kg/kwh $0.44 /kg
Iron losses	$0.08 /pct

Cost Analyses

The major operating costs associated with the flowsheets are 1) grinding energy and media; 2) reagents (collector); and 3) lost iron units. The basis for estimating these costs are given in Table 2. The costs were calculated in dollars per long ton of flotation

Figure 2.
BASELINE FLOWSHEET FOR HIBTAC AND MINNTAC CONCENTRATE



feed. The power is the kwhr for grinding only and does not include pumps, agitators, blowers, etc. The cost for power was taken to be $0.04 per kwhr. Collector costs were estimated to be $1.10 per pound. The grinding media consumption was calculated from the Bond formula for grinding magnetite with an abrasion index of 0.222. This gives an average wear loss of 0.094 kg/kwhr. Media costs were estimated to be $0.44/kg. Iron losses were taken to be $0.08 per percent iron lost. The iron losses do not include any losses associated with dewatering or densification.

Seven flowsheets were developed for Minntac concentrate. The silica levels varied from 1.56 percent to 1.99 percent with iron recoveries varying from 88.6 percent to 98.3 percent. The major operating costs varied from $1.66 to $2.53 per long ton. The lowest cost flowsheet yielded 1.80 percent SiO_2 with a 98.3 percent iron recovery. This flowsheet is shown as Fig. 3.

The seven flowsheets developed for Hibtac concentrate did not parallel the ones created for Minntac, as more use was made of column flotation. In the Minntac cases only one flowsheet utilized column flotation, but in the Hibtac case four flowsheets utilized column flotation. The difference may be a result of the testing process, as the early work with columns uncovered several beneficial effects so that more use was made of column flotation in later runs. In the Hibtac cases silica levels ranged from 1.84 to 2.0 percent while iron recoveries ranged from 94.0 to 98.9 percent. Major operating costs ranged from $1.07 to $1.73 per long ton. In the Hibtac cases two flowsheets came within $0.02 of each other at the low end of the cost range. The second lowest cost flowsheet appears to give the better silica level and iron recovery, i.e., 1.87 percent and 97.2 percent respectively (Fig. 4).

Six flowsheets were constructed for LTV concentrate. In general these flowsheets showed higher silica levels and iron losses than the flowsheets for the other plants. In the LTV cases the silica levels ranged from 1.87 to 2.20 percent, and the iron recoveries ranged from 88.1 to 89.5 percent. Major operating costs ranged from $0.87 to $1.46 per long ton. The lowest cost flowsheet yielded a silica level of 2.2 percent. However the baseline flowsheet had a cost that was only one cent more with a silica level of 2.02 percent and an iron recovery of 88.3 percent (Fig 5).

While these comparisons are interesting as they show the technical feasibility of producing concentrates with less than 2 percent silica, an assessment of the practicality of production must await further analysis of capital costs and marketability. These items are to a large extent plant specific and so will be left to the owners and operators.

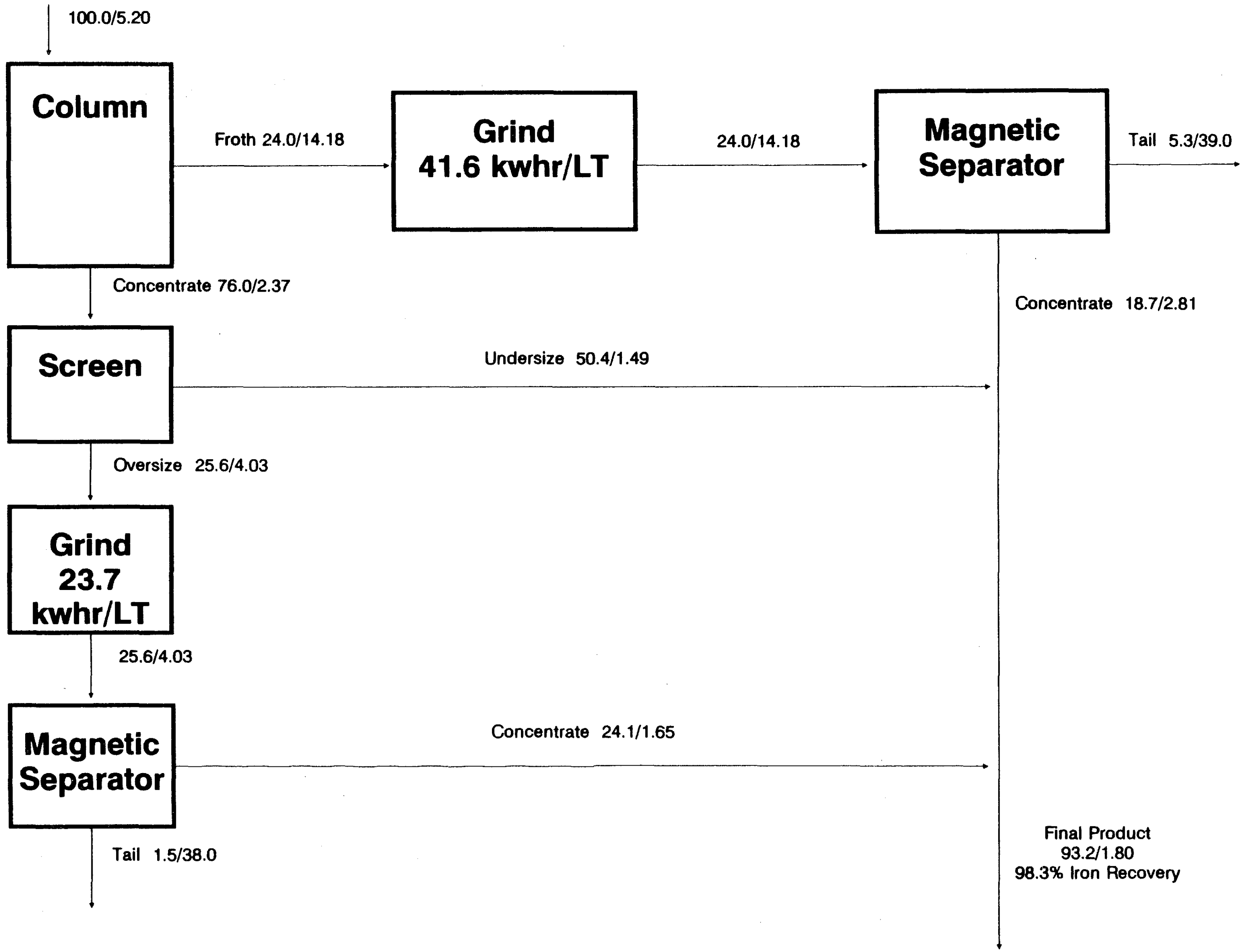


Figure 3. Material Balance for Flowsheet Using Rougher Column Flotation MINNTAC Concentrate

Figure 4.
MATERIAL BALANCE FOR FLOWSHEET USING ROUGHER COLUMNFLOTATION -- HIBTAC CONCENTRATE

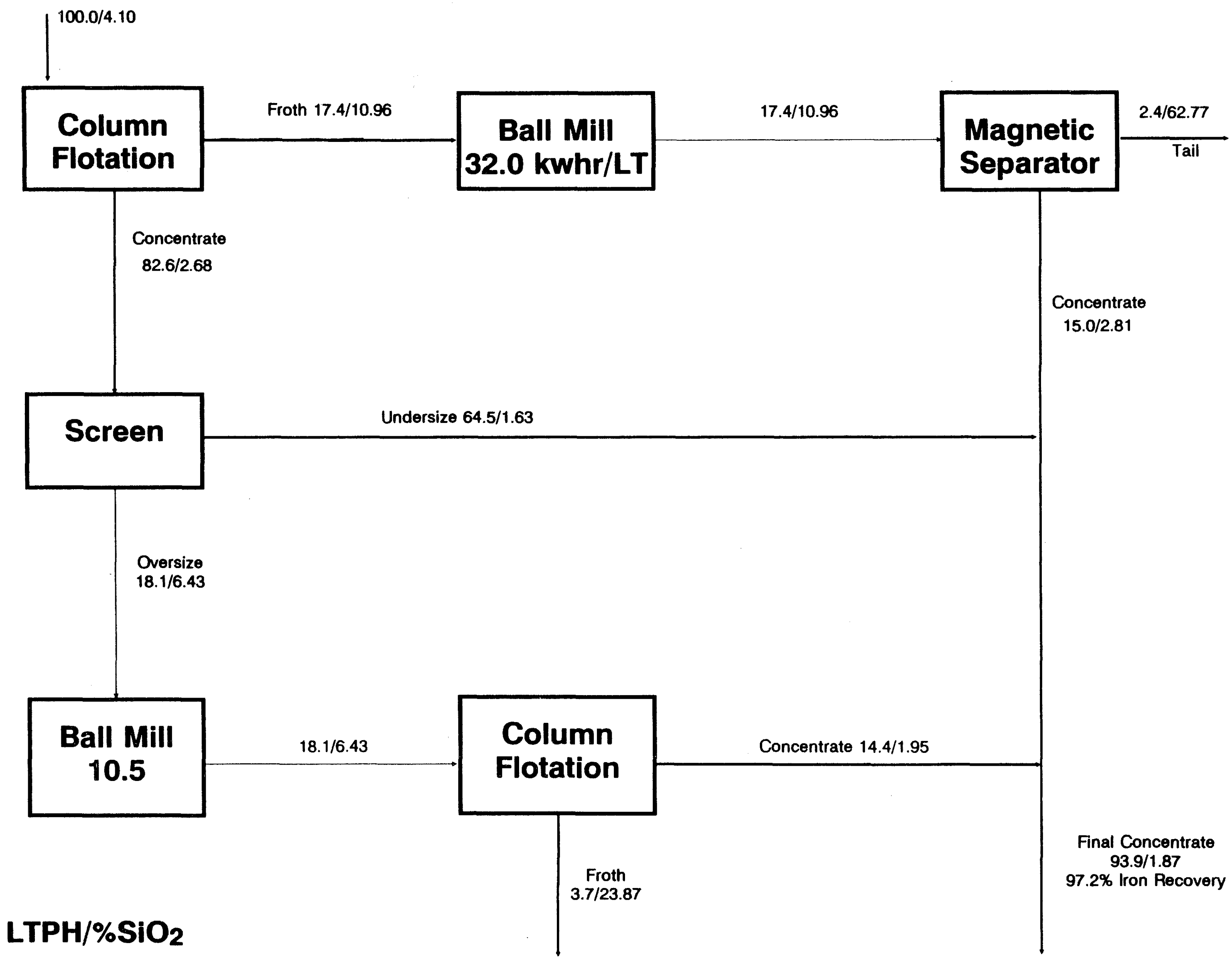
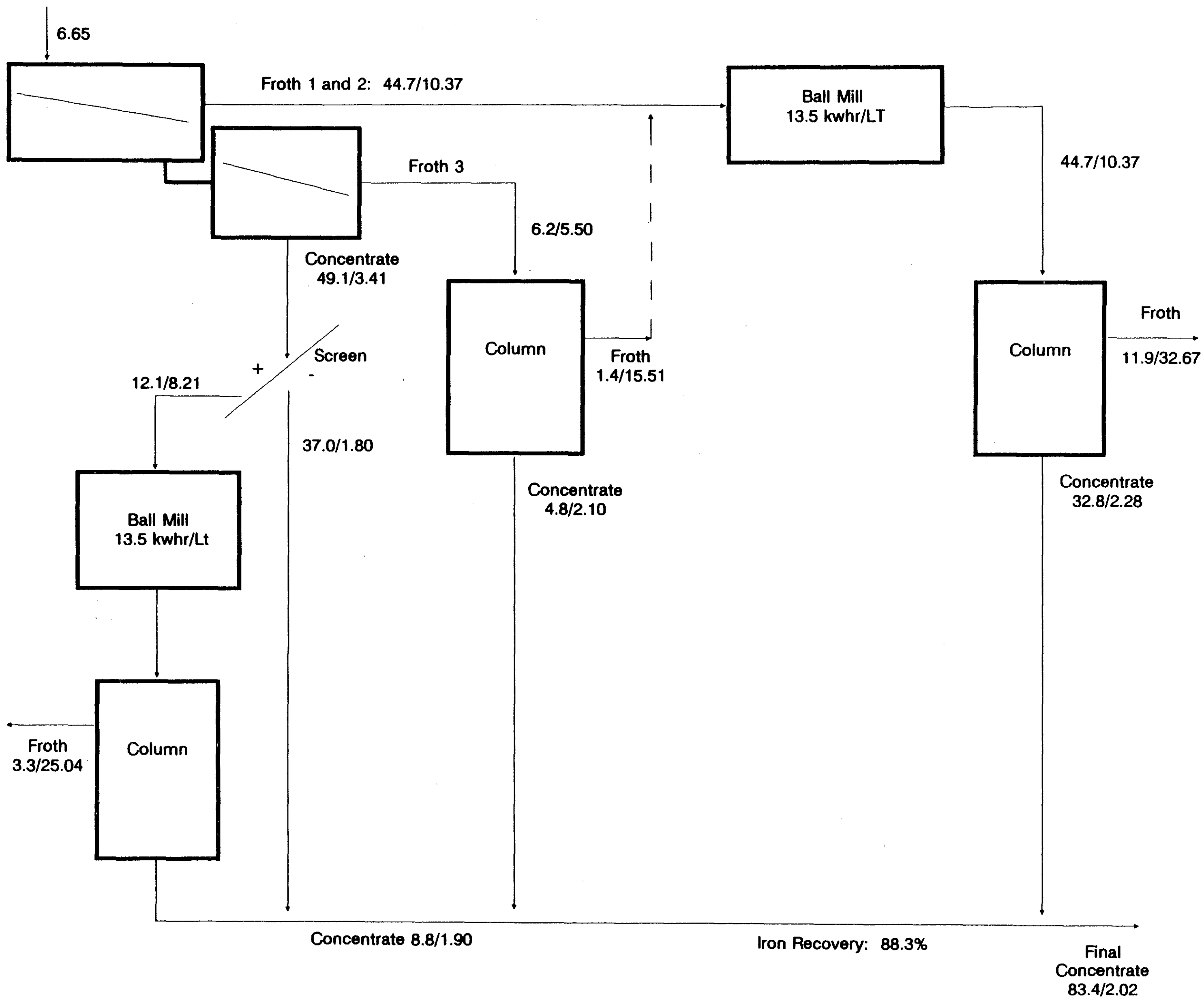


Figure 5. Material Balance Flowsheet for LTV Concentrate



CONCLUSIONS AND RECOMMENDATIONS

The test program has achieved its principal objectives which were to demonstrate the technical feasibility of producing taconite concentrates that would contain less than -3 percent silica. In the Minntac and Hibtac cases it was possible to obtain less than 20 percent silica concentrates with over 95 percent iron recovery from plant flotation feed. However, in the LTV sample iron recoveries were generally in the 88 to 89 percent range.

The primary concentration step in the flowsheet is rougher flotation with either conventional cells or column flotation to produce a low silica concentrate and a high silica froth.

Fine screening of the rougher flotation concentrate using Derrick screens with a 280 deck will produce an undersize product that will contain less than 2.0 percent silica containing 60 to 75 percent of the iron units for Minntac and Hibtac concentrates.

Retreatment of both the rougher concentrate fine screen oversize and the rougher froth are required to achieve satisfactory iron unit recoveries.

Retreatment of the rougher concentrate fine screen oversize by regrinding and magnetic separation or column flotation is effective and will produce a low silica stream that can be added to the final concentrate.

Retreatment of the rougher froth is more difficult and several options were tested:

1) Regrind and magnetic separation
2) Regrind and column flotation
3) Scavenger column flotation with froth regrind

All three options are technically feasible; the principal difference between them is the trade-off among regrind energy, grade, and recovery.

If conventional cells are used for rougher flotation, collector addition is an important variable. Two stage collector addition resulted in lower costs but higher silica concentrates than single stage collector addition. More test work is needed to determine the optimum way to add the collector.

Use of column flotation in the rougher stage produces less froth product because of more effective recovery of 500-mesh magnetite. This reduces the cost of secondary treatment.

The pretreatment options tested included the following:

1. Conventional magnetic separation
2. Fine screening with a Derrick "K" screen
3. Low intensity matrix magnetic separation
4. Desliming and elutriation.

Pretreatment was not essential for production of concentrates containing less than 2.0 percent silica. However, it does allow production of lower silica concentrates from the fines fraction. Pretreatment would then allow operators to reduce silica levels by another 0.2 to 0.4 percent in critical situations.

The most promising pretreatment method was low intensity matrix magnetic separation, since it allowed production of lower silica concentrates and reduced the flotation froth volumes. The latter, of course, reduces the amount of expensive treatment required. Low intensity matrix magnetic separation also appears to nearly duplicate the results of conventional rougher flotation for limited silica reduction. This means that the technology has the potential to supplement or replace some of the existing taconite flotation circuits.

The major operating cost items associated with the production of low silica concentrates will be power, grinding media, flotation reagents, and iron losses. Over 50 percent of the incremental operating costs is typically in grinding energy and media consumption. The cost estimates generated in this study do not seem unreasonable considering the amount of silica reduction achieved.

Several of the study areas could benefit from further study. For example:

1. Low intensity matrix magnetic separation because of its potential to supplement or replace flotation and reduce costs.

2. Column flotation as the rougher stage.

3. Expanded work on stage addition of collector in the rougher stage for conventional cells.

4. Fine grinding of froth or screen oversize using the Vertimill system.

5. Scavenger flotation of conventional rougher froths.

6. Further work using column flotation to upgrade reground froth.

All of the project participants believe the work should continue into a second phase, as originally proposed. Most of the items listed above will be built into the next phase of the project, which is primarily targeted toward production of enough low silica concentrate to meet the requirements of metallurgical testing. In Phase II the pilot plant will be run in a semi-continuous fashion to produce about 4 tons of low silica concentrate. The pilot plant run will be followed by an extensive series of pot-grate tests to explore the metallurgical properties of the new concentrates. In addition, new magnetite depressants and other techniques to reduce the amount of fine ore carry over will be tested on a pilot scale. A proposal for the second phase has been drafted by the principal investigators and the project manager. This document is now out for review by the AISI collaborators.