ER 1318

PILOT PLANT STUDIES OF THE FLOTATION OF AN ILMENITE ORE

By

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ER 1318

A thesis submitted to the Faculty and Board of Trustees of the Colorado School of Mines in partial fulfillment of the requirements for the degree of Master of Engineering.

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ABSTRACT

A study was made on the flotation of ilmenite in a three stage pilot plant at the Tahawus mill of the National Lead Company.

The results showed that concentrate grades of better than 45% TiO_2 with better than 90% recovery could be achieved in this three stage circuit.

The data from the flotation tests together with data from impulse tracer tests were used to determine the applicability of a kinetic model of flotation to this circuit. It was found the flotation is dependent on both mechanical and chemical cell conditions. A modification to the kinetic model is suggested to account for the mechanical conditions while still assuming the validity of a first order kinetic equation.

TABLE OF CONTENTS

	Page
ABSTRACT	iii
TABLE OF CONTENTS	iv
LIST OF FIGURES	viii
LIST OF TABLES	x
ACKNOWLEDGEMENTS	xii
INTRODUCTION	1
Statement of the Problem	1
Organization of the Problem	2
Method of Attacking Problem	2
Pulp Flow Rate	3
Percent Solids in the Feed	5
Air Flow Rate	5
Processing of Samples	5
ANALYTICAL METHODS AND PROCEDURES	7
Analytical Procedure for TiO_2 Determination	7

	Page
Solution of the Sample	7
Reduction of the Solution	8
Titration of the Solution	9
Analytical Procedure for Ilmenite Determination	9
Preparation of Standards	9
Preparation of Calibration Graph	10
Determination of Ilmenite in an Unknown Sample.	10
Analytical Procedure for NaCl Determination	12
Preparation of Standardized $AgNO_3$	14
Titration of Unknown Solution	14
Determination of Volumetric Flow by the Salt Method.	15
Calibration of the Flask	15
Preparation of Standardized Solution	16
Determination of Flow	16
Experimental Procedure for Tracer Tests	17
Experimental Procedure for Flotation Tests	18
REVIEW OF LITERATURE	25
Chemistry of Flotation	25
Mixing Theory	26
Flotation Kinetics	34

Page

Importance of a Kinetic Model..... 34

A Postulated Mathematical Model for Flotation... 35

Validity of Assumptions of Kinetic Model..... 40

Circuit Design..... 44

EXPERIMENTAL RESULTS..... 46

Impulse Tracer Tests..... 46

Flotation Tests..... 47

Flotation Kinetics..... 56

DISCUSSION OF EXPERIMENTAL RESULTS..... 57

Impulse Tracer Tests..... 57

Mixing Efficiency versus Flow Rate..... 59

Mixing Efficiency versus Impeller Speed..... 59

Mixing Efficiency versus Air Flow Rate..... 59

Mixing Efficiency versus Cell Number..... 60

Validity of Mixing Efficiency Parameters..... 60

Laboratory Cell Mixing versus Commercial Cell Mixing..... 63

Significance of Results..... 64

Ilmenite Flotation Tests..... 65

Preparation of Pulp..... 65

	Page
Reproducibility of Results	67
Effect of Variables on Rougher Bank Performance	68
Selection of Number of Cleaning Stages	75
Prediction of Recovery	77
Selection of Circulating Loads	78
Selection of an Optimum Circuit	80
Flotation Kinetics	82
Significance of Results	86
CONCLUSIONS	89
BIBLIOGRAPHY	91
APPENDIX	A 1
Section I, Impulse Tracer Tests	A 2
Section II, Ilmenite Flotation Tests	A 11
There is no page 46.	
Section III, Flotation Kinetics	A 47

LIST OF FIGURES

Figure		Page
1.	Schematic diagram of flotation pilot plant.....	4
2.	Calibration graph for X-ray determination of ilmenite	11
3.	Flowsheet of Tahawus mill.....	19
4.	Schematic diagram of constant feed device.....	22
5.	Calibration graph for orifices in constant feed device	23
6.	Cell mixing regimes as proposed by Cholette and Cloutier.....	28
7.	Bank mixing regimes as proposed by Bull and Spottiswood.....	29
8.	Impulse Tracer Test No. 1.....	48
9.	Impulse Tracer Test No. 2.....	49
10.	Impulse Tracer Test No. 3.....	50
11.	Impulse Tracer Test No. 4.....	51
12.	Impulse Tracer Test No. 5.....	52
13.	Impulse Tracer Test No. 6.....	53
14.	Impulse Tracer Test No. 7.....	54

Figure		Page
15.	Impulse Tracer Test No. 8.....	55
16.	Volumetric flow (l/min) versus upgrading.....	70
17.	Volumetric flow (l/min) versus recovery (%)....	71
18.	Pulp density (%) versus upgrading.....	72
19.	Pulp density (%) versus recovery.....	73
20.	Air flow (scfm) versus upgrading and recovery (%).....	74
21.	Concentrate grade versus number of stages of flotation.....	76
22.	Pulp density of cleaner tailing versus upgrading.	79
23.	Proposed flotation flowsheet.....	81
24.	Proposed model of flotation process.....	88

LIST OF TABLES

Table		Page
1.	Summary of flow conditions for impulse tracer tests	46
2.	Summary of mixing efficiencies in the sixth cell.	58
3.	Summary of mixing efficiencies in Test 6	60
4.	Summary of peak arrival times	61
5.	Comparison of peak and mean arrival times	62
6.	Comparison of peak arrival times in laboratory and commercial cells	63
7.	Summary of screen assay tests on rougher bank concentrates	66
8.	Summary of some rougher bank tests	68
9.	Comparison of cell concentrate grade using different air flow rates	75
10.	Summary of assays from Test Nos. 24 to 26	77
11.	Partial metallurgical balance on proposed flotation circuit	80
12.	Flotation rate constants for TiO_2 in Flotation Test No. 18	83

Table		Page
13.	Flotation rate constants for TiO_2 in Flotation Test No. 17.....	83
14.	Flotation rate constants for TiO_2 in Flotation Test No. 16.....	83
15.	Comparison of flotation rate constants for TiO_2 and ilmenite.....	85

ACKNOWLEDGEMENTS

The author wishes to express his appreciation to the National Lead Company for the provision of equipment and financial assistance to carry out this study.

The author's gratitude is also extended to his advisor, Dr. W.R. Bull, and to members of his committee--Dr. P. G. Herold, Dr. G.P. Martins, and Prof. T. Balberyszski. Thanks also go to Mr. B. C. Mariacher and members of the staff of the Colorado School of Mines Research Institute for their advice, suggestions, and assistance.

The author would also like to thank the Colorado School of Mines Foundation and the State of Colorado for provision of financial assistance during his studies at the Colorado School of Mines.

INTRODUCTION

Statement of the Problem

The Mining and Exploration Division of the National Lead Company operates an open-pit ilmenite mine at Tahawus in New York. The company ships its concentrates directly to its own smelters. National Lead is interested in enlarging the output of its present mine since recent work has increased the amount of proven ore reserves and recent mill expansion has increased mill capacity.

The Engineering and Mining Journal quotes ilmenite prices on the London Metal Exchange based on a TiO_2 content of 52%. Grades of the flotation concentrates produced at Tahawus fall beneath this value and if the concentrate were placed on the open market, the company could expect to pay large penalties. As the tonnage milled is increased, it is expected that a higher proportion of the feed will be treated in the flotation section of the plant.

The pilot plant consisted of a 6-cell rougher bank, a 4-cell cleaner bank, and a 2-cell receleaner bank. Initially tracer tests were carried out to determine if the mixing achieved was near ideal. At flow rates most closely approaching this ideal state, flotation tests were carried out using pulp from the mill's final conditioning tank. The pilot plant circuit is shown in Figure 1.

It was decided to investigate the effect of the following variables in each of the banks.

Pulp Flow Rate

Per Cent Solids in Feed

Air Flow Rates

Pulp Flow Rate: From the initial impulse tracer work, certain flow rates were chosen, within which departure from ideal mixing was not too severe. Flow rates to the rougher bank could be altered by changing the amount of pulp being drawn from the mill's final conditioning tank. Water additions to this pulp could be varied to maintain a constant solids per cent in the feed. Air flow rates were not altered.

In the cleaner bank, pulp flow rates to that bank could be altered by varying the air flow rate on the rougher bank. This varied the amount of material drawn off in the froth in each case. Water ad-

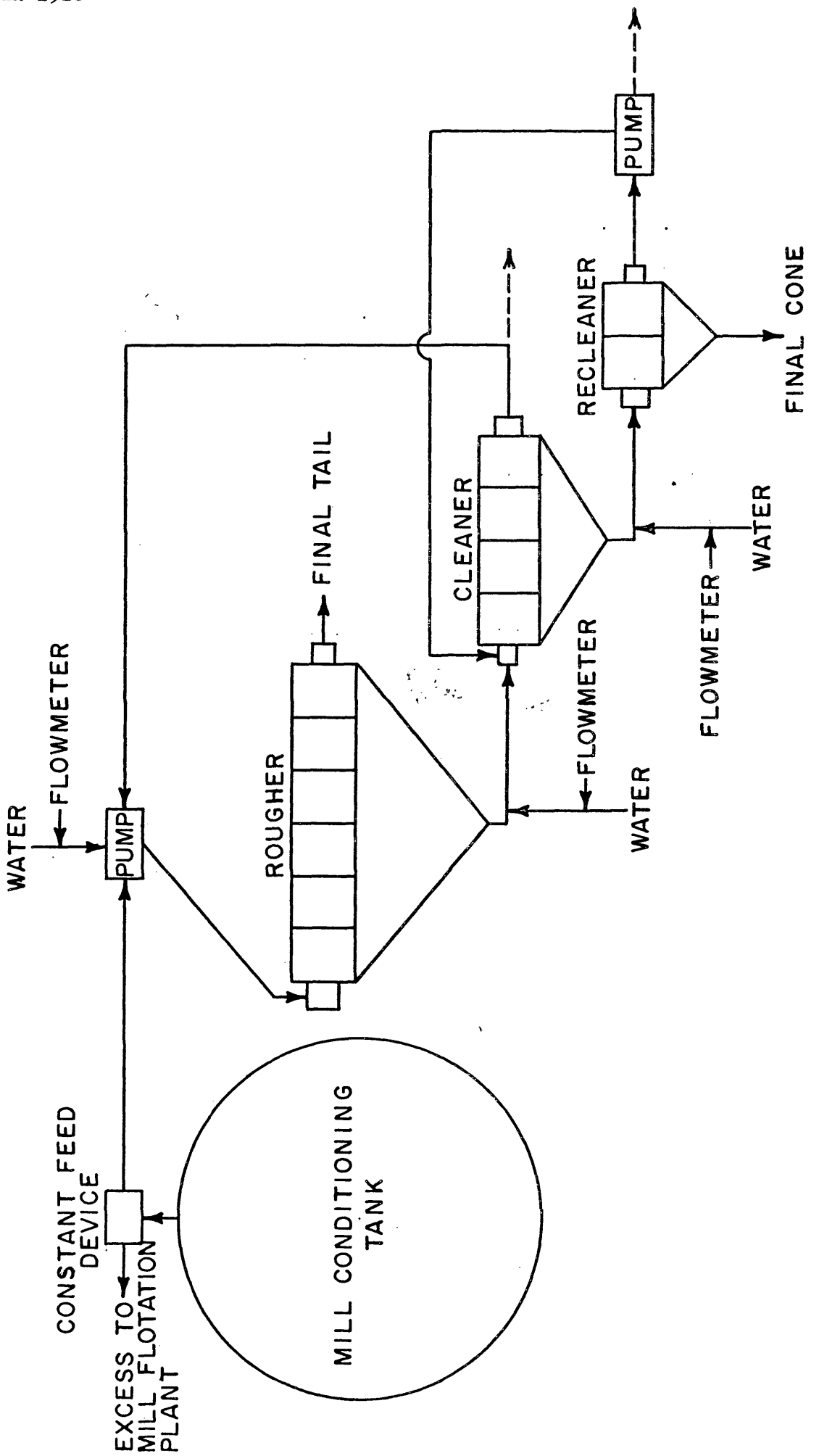

Figure 1. Schematic diagram of flotation pilot plant

ditions could be varied to maintain a constant per cent solids.

Per Cent Solids in the Feed: Changes in this variable could be achieved by increasing the volume of pulp being withdrawn from the mill's conditioning tank. The water additions to this pulp could be varied to maintain a constant volumetric flow rate.

Changes in per cent solids to the cleaner bank could be achieved by altering the air flow to the rougher banks and by a compensating change in the dilution water.

Air Flow Rate: Air additions to each cell were measured by air flowmeters. These meters could measure additions in the range from 0-14 cubic feet of air per minute. In the rougher bank flow rates from 0-8 cubic feet per minute were found to be most effective while in the cleaner bank rates from 0-2 cubic feet per minute gave best results.

Processing of Samples

A solution of common salt was used as an impulse tracer for the measurement of retention times. Samples were withdrawn from the cells and titrated against standardized silver nitrate to determine salt concentrations.

For the flotation test work, ilmenite's presence was measured by the TiO_2 assay of the sample. The assay procedure used a potassium

pyrosulphate fusion, an aluminum reduction, and a titration of the solution against standardized ferric ammonium sulphate.

For the study of flotation kinetics, two approaches were used. The first was to determine if a flotation rate constant for TiO_2 could be determined. This approach used the chemical assays described previously. The second approach was to determine a flotation rate constant for the mineral ilmenite. This required the determination of ilmenite concentration by X-ray diffraction methods.

ANALYTICAL METHODS AND PROCEDURES

Analytical Procedure for TiO_2 Determination

TiO_2 was determined by the aluminum reduction method. This involves the following steps.

Solution of the Sample

Reduction of the Solution

Titration of the Solution

Solution of the Sample: A 0.400 gram sample is weighed into a 50 ml high-form porcelain crucible containing 10 grams of potassium pyrosulphate and 4 drops of 36N H_2SO_4. The sample is then covered and fused in a muffle furnace at 1400°F for twenty-five minutes. The sample is then removed, cooled, and transferred to a 500-ml wide-mouth Erlenmeyer flask. Any adhering sulphate in the crucible is dissolved by 5 percent H_2SO_4 and added to the flasks. 30 ml of 50 percent H_2SO_4 are added to the flasks and the contents are allowed to stand for 10 minutes. 40 ml of 50% HCl are then added to the flask

and the contents are made up to about 200 ml. The flask is then heated short of boiling until all the fused sample is in solution.

This step dissolves all the acid soluble oxide in the sample. Some "glassy" silicates (e.g., garnets) may not dissolve in this procedure. Since garnets are often titanium-bearing, the TiO_2 assay may not be an assay of the total TiO_2 in the sample. Titanium goes into solution, in the presence of air, predominantly as the Ti^{4+} ion.

Reduction of the Solution: When the solution step is complete, the flasks are brought to boiling and aluminum foil is added. This reduces the Ti^{4+} in solution to Ti^{2+}. From this point on, it is essential that the solution be kept from the atmosphere to prevent reoxidation of the titanium ion. For this a rubber stopper containing a delivery tube is placed on the flask. The other end is placed below the surface of a saturated sodium bicarbonate solution in a beaker. When the aluminum foil has dissolved and the bicarbonate solution starts to rise in the tube, the flasks are moved back to the hot plate and boiled for two or three minutes.

After boiling the flasks are removed from the hot plate and bicarbonate is allowed to be sucked back into the flask. When violent sucking back has ended the flasks can be transferred to a cooling bath

and allowed to cool until all action has ceased. The delivery tube is now removed and the flask is immediately stoppered and set aside for titration.

Titration of the Solution: The stopper is removed from the flask, 2 ml of ammonium thiocyanate indicator added, and the solution is immediately titrated with standard ferric ammonium sulfate solution to a light orange endpoint. The reaction oxidizes the Ti^{2+} back to Ti^{4+}, hence

1 gram Fe = 0.4288 Ti

or 1 ml standard solution = 0.00475 grams TiO_2

Analytical Procedure for Ilmenite Determination

Ilmenite concentration was determined by the use of x-ray diffraction techniques. This involved the following steps:

Preparation of Standards

Preparation of Calibration Graph

Determination of Ilmenite in an Unknown Sample

Preparation of Standards: To prepare standard samples of known ilmenite concentration, an initial sample of high purity ilmenite had to be prepared. This was done by taking a sample of fine Wetherill concentrate (+200 mesh) and passing it repeatedly through a laboratory electro-

static separator. This would remove any nonmagnetic, non-conducting minerals (such as garnets, horneblends of feldspars). This sample was then ground and passed through a Davis tube to remove any magnetite which may have been liberated by the grinding. This material assayed 50.1 percent TiO_2. (cf 52.7 for stoichiometric $FeTiO_3$. Dana, 1944, cites examples of ilmenite ranging from 48.6 to 67.7 percent TiO_2.)

It was assumed that this material was pure ilmenite. This was then diluted with amorphous quartz (silica gel) to give samples of known ilmenite concentration.

Preparation of Calibration Graph: The samples of known concentration were x-rayed using CuK_2 radiation with a scanning speed of 2Θ of 20 per minute and a magnification factor of 16. The 32.5° peak was chosen as a basis for determining concentration.

A plot was made of peak height against ilmenite concentration. A smooth curve was drawn through these points and this was used as a calibration graph. This graph is shown in Fig. 2.

Determination of Ilmenite in an Unknown Sample: Each time the diffractometer was used, one of the standards was x-rayed first. Adjustments to the settings were then made to ensure the peak height to this

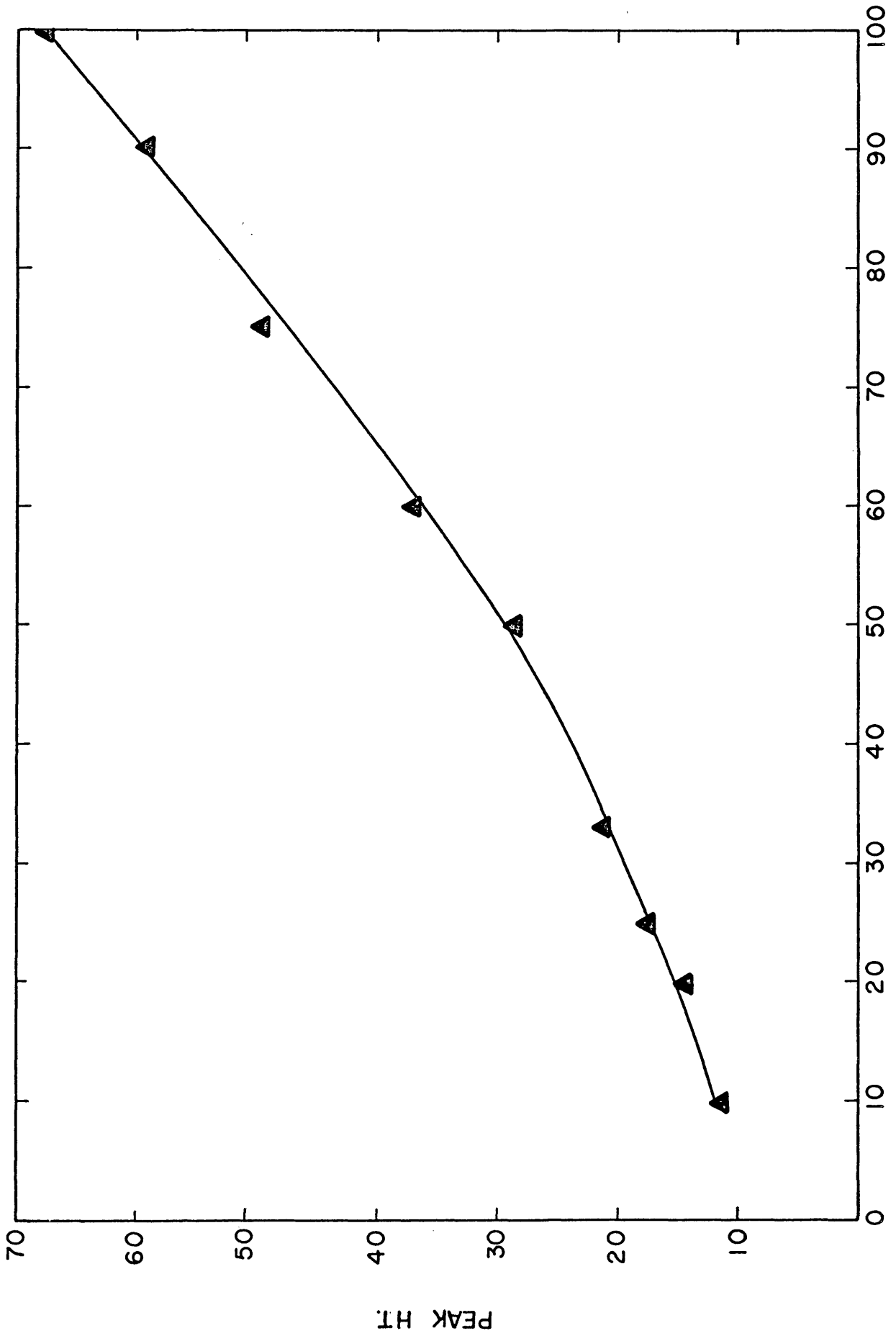

Figure 2. Calibration graph for x-ray determination of ilmenite

sample was the same as that on the calibration graph. If the machine was used for any extended length of time (say, longer than 3 hours) this check was made again.

After the adjustments were made the sample was mounted in the standard specimen holder and the peak height at 32.5° determined. The ilmenite concentration for this sample could then be found from the calibration graph.

Analytical Procedure for NaCl Determination

Sodium chloride was used as an impulse tracer to determine the mixing characteristics of the pilot plant rougher bank and the mill rougher banks. Departure of the mixing characteristics from ideality can be measured in two ways. The first is that proposed by Danckwerts (1953) in which mixing efficiency is determined from a normalized residence time distribution curve. This was originally determined for a step input but has been modified by Bull and Spottiswood (1970) to apply to a pulse input.

The second method is that described by Levenspiel (1962). This entails a mathematically rigorous definition of dead space. Liquid which stays in the cell for longer than three times the mean residence time is assumed to be stagnant. The volume occupied by this liquid is defined

as dead space. This, together with the material bypassing, will affect the position of the mean residence time. The total effect of all departures from perfect mixing on the mean can be used as a measure of mixing efficiency. Thus

$$\epsilon = 100 \frac{Ta}{T}$$

where

ϵ = mixing efficiency

Ta = actual dimensionless arrival time

T = theoretical dimensionless arrival time

Use of Danckwerts method involves the use of a normalized curve. This inolves a determination of the initial concentration (Co) in the cell. Bull and Spottiswood (1970) point out that the method is insensitive to changes in flow rate. The second method is the one used in this paper.

Since Levenspiel's method does not require a knowledge of the absolute tracer concentration at any time, but merely a knowledge of the concentration change with time, commercial reagents can be used as tracers. In view of this, sodium chloride in the form of common salt was used.

Determination of NaCl concentration in unknown solutions was achieved by titration against a standardized $AgNO_3$ solution using a potassium chromate indicator. This involved the following steps:

Preparation of Standardized $AgNO_3$

Titration of Unknown Solution

Preparation of Standardized $AgNO_3$: A standardizing solution of NaCl was prepared by dissolving 0.508 gram of common salt in 1 liter of distilled water. A stock solution of $AgNO_3$ was prepared by dissolving approximately 15 grams of $AgNO_3$ in 1 liter of distilled water.

Then 25 ml aliquots of the NaCl solution were titrated against the stock $AgNO_3$ solution. It was found that 23.8 ml of $AgNO_3$ were required for neutralization.

Hence, 1 ml salt solution contains 508 ppm of NaCl

and 1 ml $AgNO_3$ solution = 21.3 ppm of NaCl

Titration of Unknown Solution: A 25 ml aliquot of the unknown solution was placed in a 250 ml conical flask and 3 drops of K_2CrO_4 indicator were added. This was titrated to a pale yellow endpoint with the standardized $AgNO_3$.

Determination of Volumetric Flow by the Salt Method

Two types of cells are used in the flotation rougher banks in the Tahawus mill. The first bank is made up of 48 cu ft Denver cells and the second of 100 cu ft Denver cells. Neither of the banks is baffled. For the impulse tracer work on the pilot plant rougher banks, flowmeters were used to measure water flow rates. In the mill, however, no flowmeters could be used for such large volumes. In these cases, flow rates had to be determined by using dry granular common salt as a tracer whose dilution by the feed pulp could be used to calculate the feed pulp flow rate. This involved the following steps:

Calibration of the Flask

Preparation of Standard Solutions

Determination of Flow

Calibration of the Flask: A 2000 ml filter flask was fitted with an orifice of approximately 1/4 inch inside diameter. This was then filled with salt which had been dried in an oven and passed through a 20 mesh sieve to remove the lumps.

The flask was then inverted over a balance pan and the rate of salt flow was determined. Since solids do not form a static head, the

salt flows at a uniform rate regardless of the amount of salt in the flask.

Preparation of Standardized Solution: The salt solution was titrated against a standardized $AgNO_3$ solution. This solution was prepared by dissolving 7.27 grams of $AgNO_3$ in 1 liter of distilled water. A solution of this strength is convenient for flow rates up to 400 gpm.

The $AgNO_3$ solution was standardized against a standard salt solution containing 0.500 gram of salt per liter. This is equivalent to 1 pound of salt per ton of water. Under these conditions, 25 ml of the salt solution will require about 5 ml of the $AgNO_3$ solution to reach the endpoint.

In order to make certain that no chloride ion was measured except that from the added salt, a blank titration was run on a mill water sample. This mill water titre was subtracted from the total titre and the net was used in flow rate calculations.

Determination of Flow: Salt was added to the stream using the calibrated flask. Duplicate samples were taken downstream from the point of addition. If titrations on these duplicates do not match, the test should be repeated.

The volumetric flow is then determined in the following way:

Tons of water per minute

$$= \frac{\text{Salt flow from flask (lbs/min)} + AgNO_3 \text{ titre for 25 ml of standard solution}}{\text{Net } AgNO_3 \text{ titre for 25 ml of stream sample}}$$

Gallons of water per minute = tons of water x 239.7

Experimental Procedure for Tracer Tests

Before any tests using pulp were performed, it was important to know the mixing characteristics of the cells. To find this out a number of tests were carried out using common salt as an impulse tracer. Mixing characteristics could be deduced from the movement of this tracer through the bank. These tests were carried out using water only. A stock solution containing approximately 60 grams per liter of common salt was prepared and filtered to remove any insoluble material. An aliquot (approximately 200 ml) of this solution was used as the impulse tracer. This procedure did not allow calculation of Q_o (the initial concentration in Cell 1), but it has been pointed out that knowledge of this was not absolutely necessary.

The 200 ml sample was injected into the first cell of the bank. The tracer's movement in solution down the bank was followed as a function of time. This required three operators (one to 2 cells) to take six

samples (one from each cell) simultaneously. Tests were run at varying flow rates with air on or off.

Although this procedure used water only, it has been shown by Jowett (1961) that this gives an approximation to the mixing behaviour of a pulp. Large deviancies could occur in highly viscous pulps. No quantitative measurements were made, but visual observations of the pulp indicated it was not extremely viscous. Experimental work also showed that large changes in mixing behaviour occur with change in impeller speed. Significant change in impeller speed (e.g., due to belt slippage or overloading of motor) was not noted in measurements taken when pulp was added.

Experimental Procedure for Flotation Tests

X-ray analysis of the feed to the flotation plant in the Tahawus mill showed the following minerals were present--ilmenite, garnet, feldspar, and kaolinite. The flowsheet for this mill is shown in Fig. 3.

Flotation plant feed is approximately 15 percent +100 mesh. Conditioning of this feed follows the procedure outlined by Lapidot (1960) for flotation of a Swedish ilmenite. Current reagent consumption is as follows.

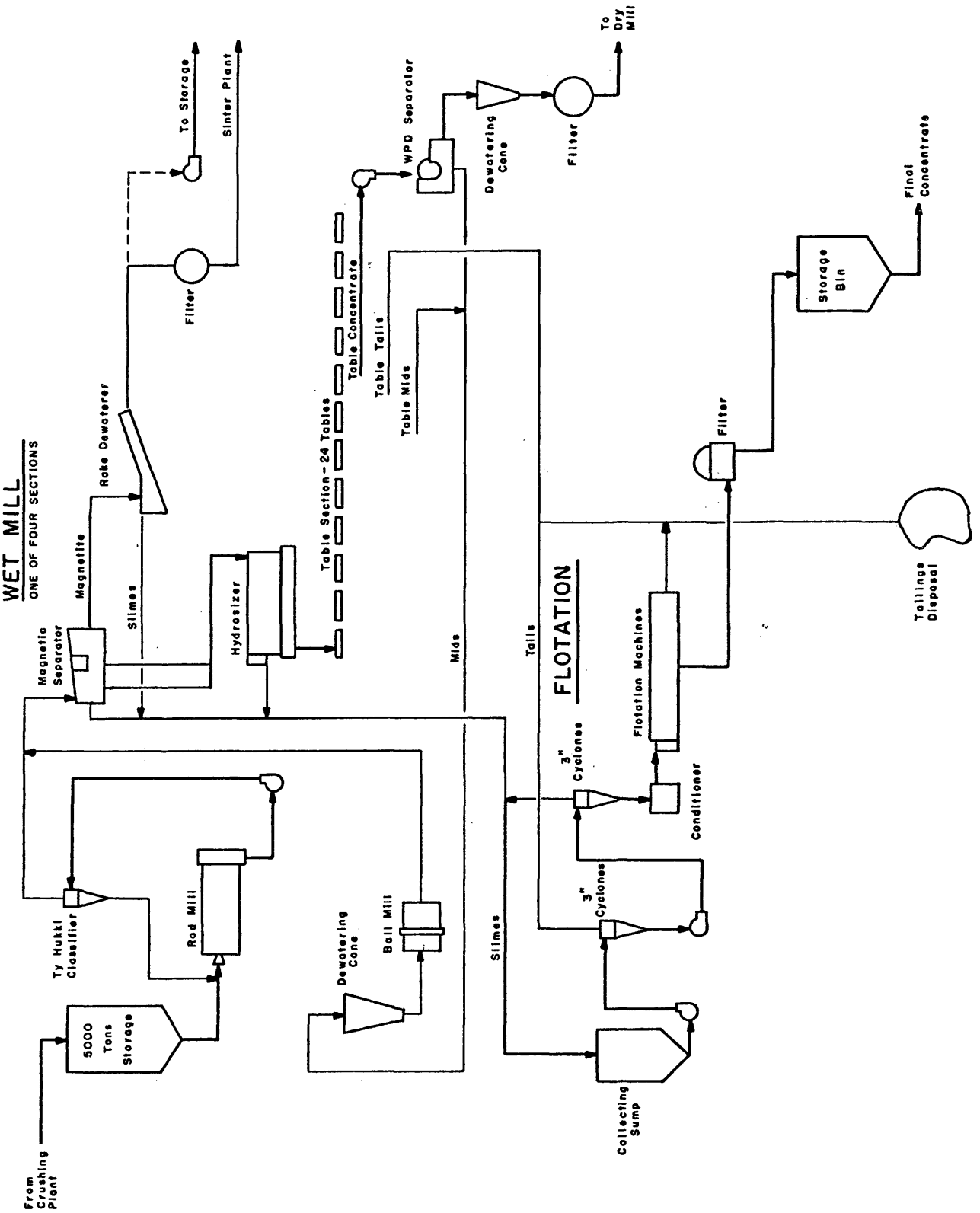

Figure 3. Flowsheet of Tahawus mill

Sodium fluoride:	1.0-1.3 lb/ton
Tall oil:	1.4-1.8 lb/ton
Fuel oil:	0.6-1.0 lb/ton
36N Sulphuric acid:	1.0-1.8 lb/ton
FR 70:	0.04 lb/ton

The mechanism by which tall oil and fuel oil act as collectors is not well understood. Aplan and Fuerstenau (1962) claim that metal oxides have flotation properties intermediate between two end members. At one end is a group of minerals (e.g., carbonates, phosphates) which have a crystal structure which is essentially ionic. They have a radicle as the anion and this is held together by strong covalent forces. The collector attaches itself to these by a chemisorption process. At the other end of this series are the covalent silicates which show only physical collector attachment. Ilmenite falls towards the silicate end of this spectrum.

The fatty acid in the tall oil functions as the collecting agent by reaction with the iron in any mineral. Hence garnet, ilmenite, and magnetite are activated and selective depression is achieved by the use of NaF and H_2SO_4.

Currently conditioning time is approximately 30 minutes at 70 percent solids. In addition to the above reagents further acid is added in the flotation circuit for pH control. This is necessary for garnet depression. pH on the rougher banks is approximately 6.5; 5.5 to 6.0 in the cleaner banks; and 5.0 to 5.5 on the recleaner banks.

The feed to the flotation pilot plant was drawn from the final conditioning tank. A schematic diagram of the constant feed device is shown in Fig. 4. The flow control in this device was the changeable bottom orifice. The overflow ensured a constant head was maintained about this plate at all times. A calibration graph of the orifices used is given in Fig. 5. Minor fluctuations in flow rates did occur because of viscosity changes in the pulp in the final conditioner. These viscosity changes were due to changes in the pulp density. Pulp density changes in the range of 68-72 percent solids were noted. The flow rate changes were not found to be severe, and in most cases it was quite reasonable to assume that feed flow rate was constant for that test.

All water additions to this pulp were measured by water flowmeters. These had a range of 0-25 gpm and most additions were in the range of 0-10 gpm. All air additions to the cells were measured by Fisher and Porter air flowmeters. These had a range of 0-14 scfm.

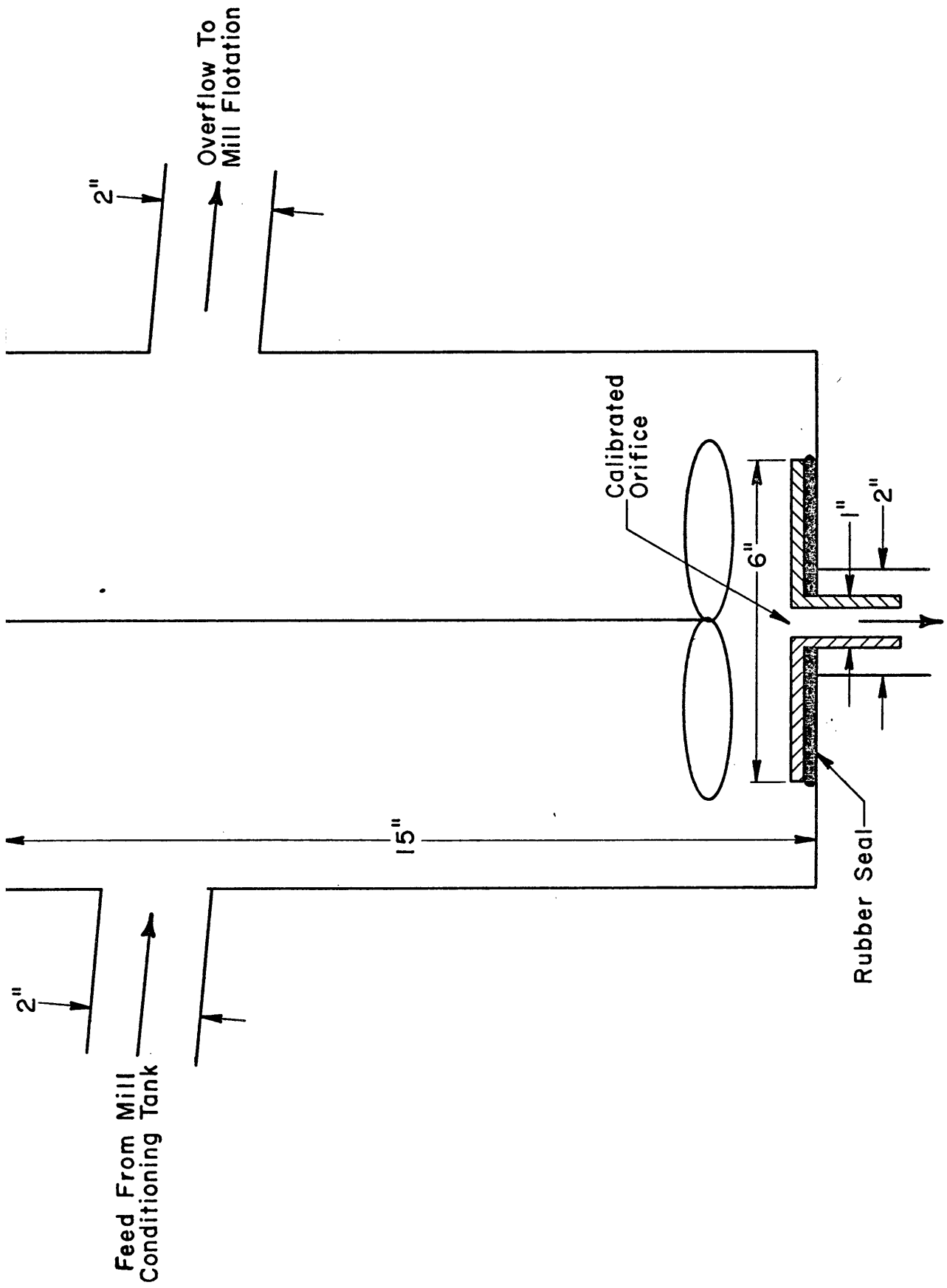

Figure 4. Schematic diagram of constant feed device

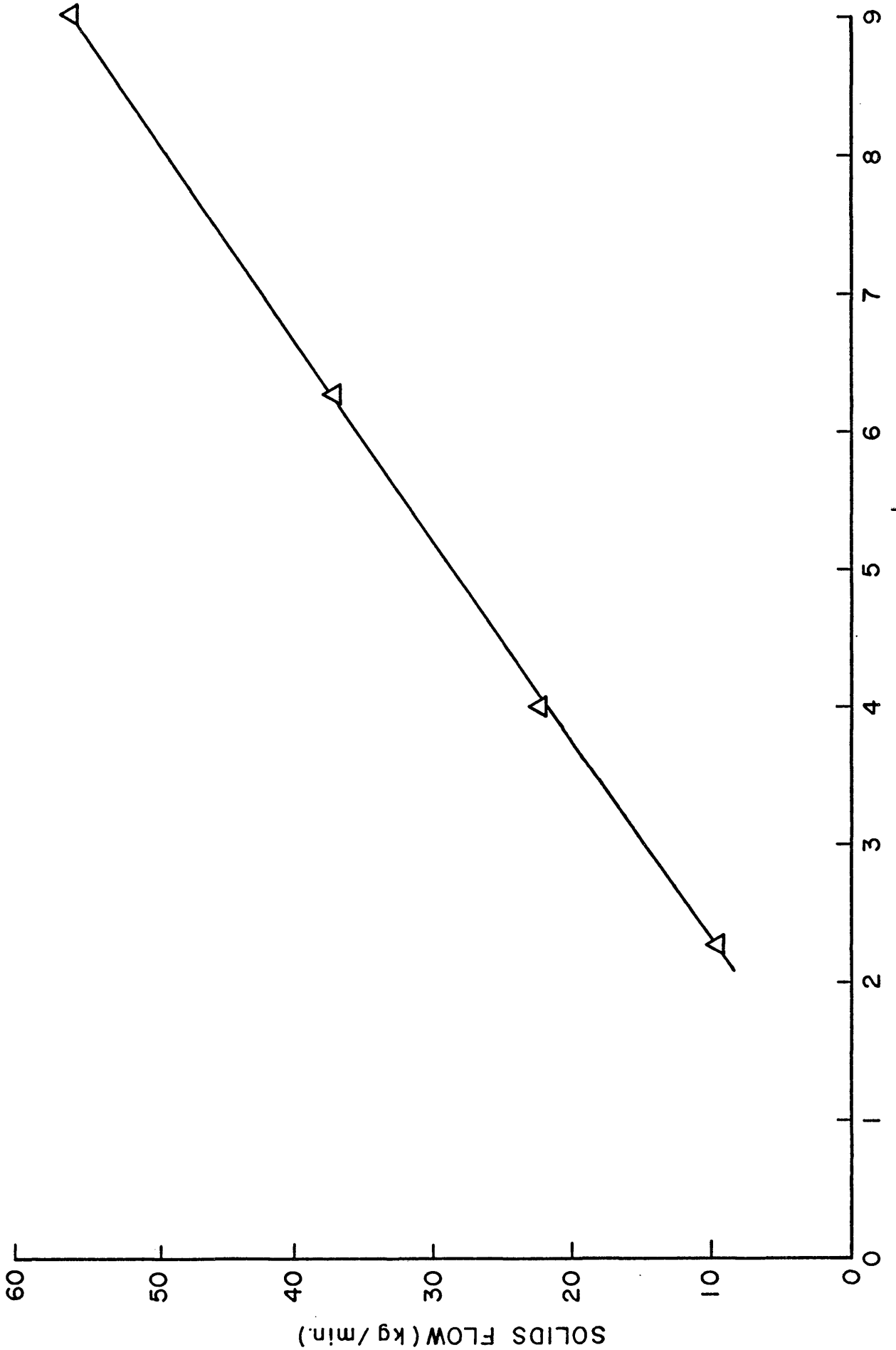

Figure 5. Calibration graph for orifices in constant feed device

Most additions were in the range 0-8 scfm on the rougher banks and 0-2 scfm on all other banks.

Collecting troughs on the banks were modified so that samples could be removed from each cell individually or from the whole bank. Cell-by-cell analysis was required for a study of the flotation kinetics of this system. One composite sample from the bank was sufficient for a bank performance study.

REVIEW OF LITERATURE

Chemistry of Flotation

Conditioning in the Tahawus mill follows the procedure outlined by Lapidot (1960) for the treatment of a Swedish ilmenite ore. X-ray analysis of the Tahawus feed showed the following minerals to be present: ilmenite, magnetite, garnet, horneblende, biotite, and feldspars. Analysis by microscope showed the magnetite to be present as exsolution lamellae on a microscopic scale. Garnet was present as both locked particles and free grains.

The collector used for the ilmenite flotation was tall oil. This is a non-selective collector for the flotation of iron-bearing oxides. It contains 45 percent fatty acids (mainly oleic and linoleic), 42 percent rosin acids, and 13 percent terpenes. Only the fatty acids act as collectors.

Of the minerals present, magnetite, garnet, and ilmenite contain iron and titanium. Aplan and Fuerstenau (1962) claim that Ti will not

take part in any chemisorption reaction. Hence the fatty acid collector functions through the Fe only. Use of a modifying agent such as NaF or H_2SO_4 will assist in the depression of iron-bearing silicates---notably garnet.

Tall oil is generally used in conjunction with fuel oil. This assists in the dispersion of the tall oil collector. Once the collector is in a dispersed condition, most efficient attachment is achieved at 70 percent solids (Baarsen and Parks (1954).

Mixing Theory

Kinetic theories of flotation have very often assumed that ideal mixing occurs in the cells of a flotation operation (e.g., Bull (1966)). In single cell studies this assumption is almost correct due to the unique baffling present. However, when a bank of cells is being considered, this assumption no longer holds. Jowett (1961) attempts to quantitatively determine the error this assumption leads to. If a first-order rate equation is assumed for a flotation process, then cell recovery can be predicted by the following equation

$$R = \frac{K\lambda}{1+K\lambda}$$

where R = fractional recovery

K = flotation rate constant

λ = nominal cell retention time

If ideal mixing occurs, then for a flow rate (V) through a cell of total volume (Vo)

$$\lambda = \frac{Vo}{V}$$

In a single cell, the available active volume (Va) is approximately equal to the total volume and hence the assumption of ideality holds.

In the case of a bank of cells there is imperfect baffling from cell to cell (with the possible exception of weir-type baffles) and mixing regimes which occur are said to be transitional between two extremes (Bull and Spottiswood (1970)). At one extreme is the case of ideal mixing. Here the tailing stream from a cell is identical to the pulp remaining in the cell. At the other extreme is the case of plug flow. Here lateral mixing only is permitted and all fluid elements have the same cell residence time. It will be shown that this picture of a transition-type cell is incomplete.

For the case of a single cell, Cholette and Cloutier (1959) proposed a subdivision of cell flow regimes with three distinct types as shown in Fig. 6. This model neglects the effect of recirculated material and is valid for a single cell, or a bank of weir-baffled cells only.

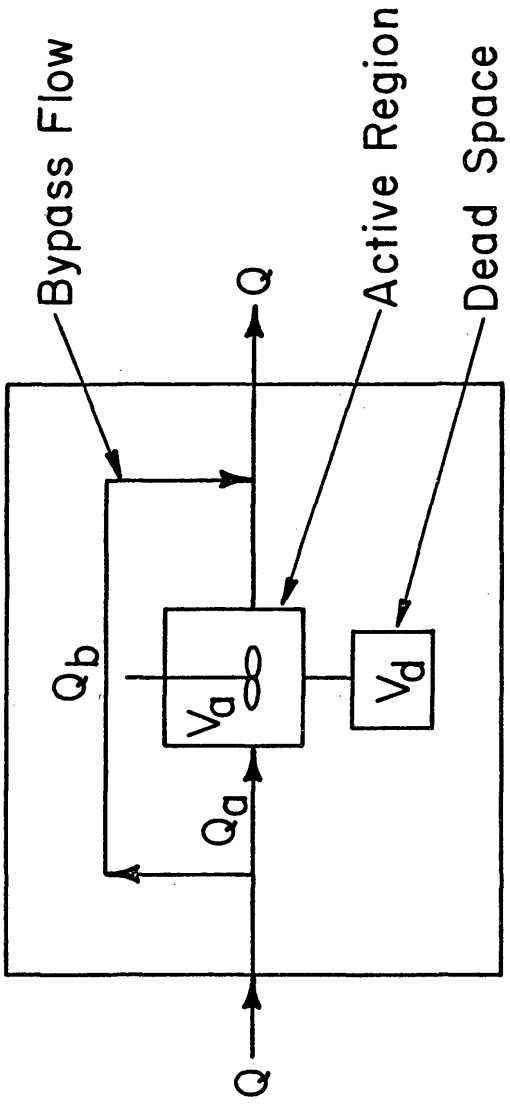

Figure 6. Cell mixing regimes as proposed by Cholette and Cloutier

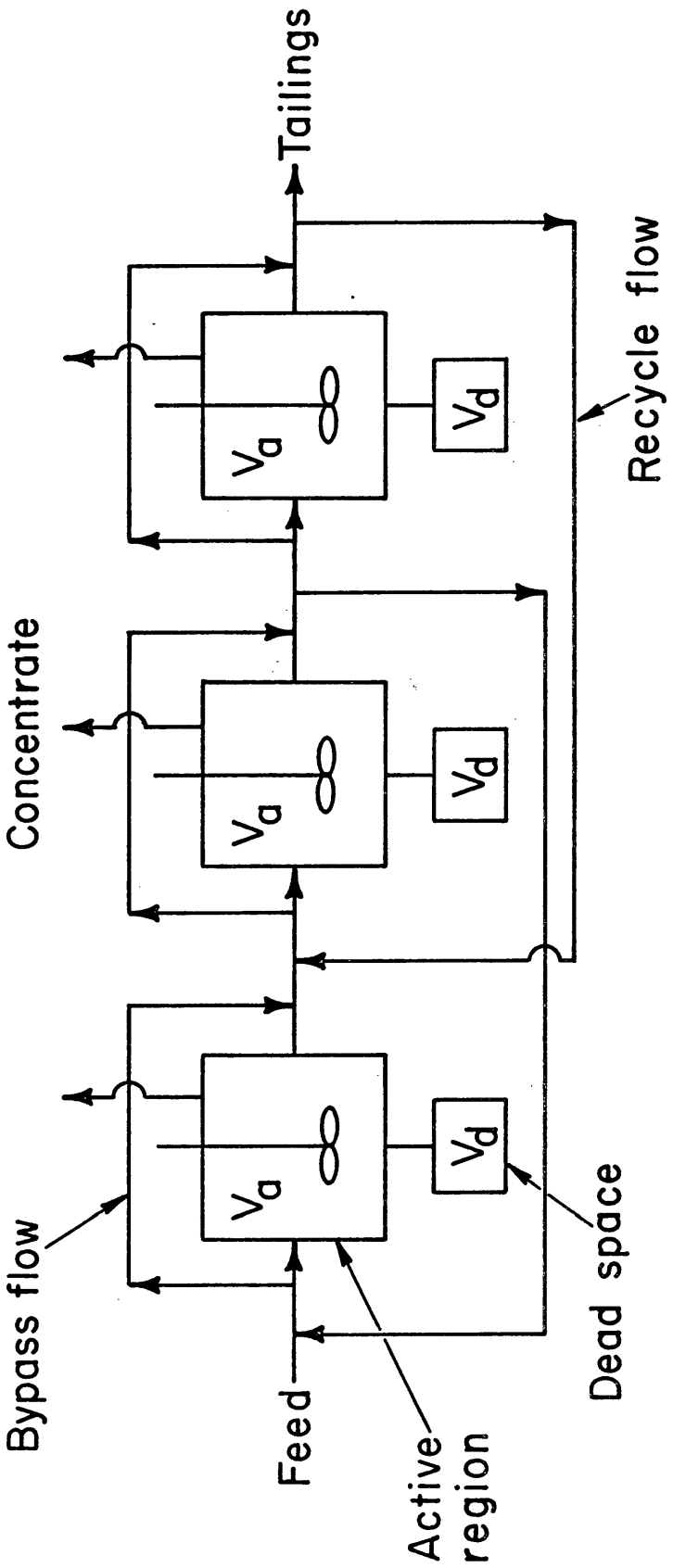

Figure 7. Bank mixing regimes as proposed by Bull and Spottiswood

A more realistic model of a "normal" flotation bank (i.e., one with less than perfect baffling) is proposed by Bull and Spottiswood (1970) and shown in Fig. 7. The presence of the recirculated material in this model complicates the simpler transitional type model proposed above. The model in Fig. 7 is too complicated for analysis using the data available, and in the following discussion the following simplifications are made:

(a) concentrate volume is zero

(b) all deviant flows can be lumped together as a single flow.

The data available for the study of mixing characteristics were the results of various impulse tracer tests. These are most conveniently plotted on a normalized curve for Danckwerts C-curve. This is a plot of dimensionless residence time (Θ) against dimensionless concentration ($C(\Theta)$) where

$$\Theta = UT/V$$

$$C(\Theta) = \frac{C}{Co}$$

and

U = flow rate

T = elapsed time

V = cell volume

C = cell concentration after some elapsed time

Co = initial tracer concentration in first cell

For the case of ideal mixing, the C-curve for any cell in a bank is given by the following equation

$$\frac{Ci}{Co} = \frac{UT}{V}^{(i-1)} \quad \frac{e^{-\frac{UT}{V}}}{(i-1)!}$$

where i = cell number

Mixing efficiency achieved in any system can be measured by the deviance of the actual C-curve from the theoretical curve.

Danckwerts (1953) shows how the C-curve for a step input can be used to determine efficiency of mixing. Bull and Spottiswood (1970) show how this can be modified to apply to a pulse input. This method uses a normalized residence time curve and in this study it was not possible to obtain dimensionless concentration values. The graphs of dimensionless time against actual concentration, which are included in the discussion section, can be converted to normalized curves by assuming the area under each curve is unity. Unit area would then vary from graph to graph. Bull and Spottiswood also point out the insensitivity of Danckwerts' area method. Because of these objections, this method of determining mixing efficiency is not followed in this thesis.

A second method for deriving the efficiency of mixing is that proposed by Levenspiel (1962). Here efficiency is derived from a study of the "mean" retention time of material in the active region of the cell. This requires a mathematical definition of dead space and active space in a cell. For the purpose of the following derivation, material which stays in the cell longer than three times the theoretical mean residence time is taken as stagnant. The volume occupied by it is said to be dead space.

The mean residence time in the active region of the cell is given by

$$\bar{\Theta}_a = \int_o^t \Theta \; C(\Theta) \; d\Theta$$

and the true mean residence time is given by

$$\bar{\Theta} = \int_o^t \Theta \; C(\Theta) \; d\Theta + \int_t^\infty \Theta \; C(\Theta) \; d\Theta$$

where t = 3x theoretical mean residence time

= 3x n

where n = cell number

In actual C-curves, a long tail exists and the second term is not zero (i.e., dead space exists).

By definition $\Theta = \frac{V}{U}$ and therefore $\Theta_a = Va/U$ where Va is that volume of the cell where perfect mixing occurs. Putting $\bar{\Theta} = 1$ and simplifying, the following relationship is obtained

$$\bar{\Theta}_a = \frac{Va}{U}$$

Since $\bar{\Theta}_a$ can be obtained from impulse tracer data, Va can be deduced.

No way has been deduced to quantitatively determine the values of Va, Vd, Qb, and Qr (see Fig. 7) simultaneously from a C-curve. If all these deviations from ideal mixing are lumped together as dead space, then efficiency of mixing (ϵ) can be expressed as follows

$$\epsilon = 100 \quad \bar{\Theta}_a$$

$$\text{or, } \epsilon = 100 \quad \frac{Va}{V}$$

The practical value of this mixing efficiency (or lack of it) will be discussed in this paper. It will be shown that each cell in a bank is unique and mixing achieved in it is quite unlike that achieved by its neighbour. Secondly it will be shown that mixing efficiency for the

same cell varies at different hydromechanical conditions. Thirdly it will be shown that the concentrate volume is not negligible. If mixing theory is going to be incorporated into a kinetic theory of flotation, these objections have to be taken into account.

Flotation Kinetics

Importance of a kinetic model: Mathematical models of steady state processes have been the basis of equipment design in such fields as chemical engineering for a long time. The models developed take into account such input variables as concentration of reagents or time in a reactor vessel. By use of chemical kinetics, thermodynamics or mass transport equations, this mathematical model will predict the products formed. To date, no one has proposed such a model for a flotation process despite the research done in the field.

Previously this was not a particularly great problem due to the small size of flotation cells and great flexibility of flotation circuits. As larger and larger cells are used this circuit flexibility will decrease. Operators will be unable to maintain circuit performance by visual examination only.

A Postulated Mathematical Model for Flotation

Grunder and Kadur (1940) were the first to make the comparison between a flotation operation and a chemical reaction. In a chemical equation of the following type

$$aA + bB \rightleftharpoons cC + dD$$

the following is true

$$\frac{C_c^{\,c} \cdot C_D^{\,d}}{C_A^{\,a} \cdot C_B^{\,b}} = \text{constant}$$

If the equation as written above contains the rate controlling step, then

$$-\frac{dC_A}{dt} = kC_A^{\,a} \cdot C_B^{\,b}$$

where C = concentration

k = rate constant

A chemical equation of this type is controlled by interactions between ions, atoms, or molecules. Generally the controlling mechanism for such reactions can be adequately described by mathematical equations, e.g., the Boltzmann distribution of kinetic energy for collision con-

trolled reactions. Flotation deals with collisions and chemical interactions of air bubbles and minerals.

Based upon such a generalized approach to flotation kinetics, Arbiter and Harris (1962) postulate the following kinetic equation

$$-dC/dt = K_i \prod_{1}^{m} C_i^{n_i} \qquad (1)$$

where C = concentration

i = control variable (e.g., air, reagent, ore particle)

m = number of variables

n = order of reaction with respect to that variable

K = flotation rate constant

This equation is too complex for practical use since too many input variables are involved. If chemical conditions are assumed to be constant, Equation (1) simplifies to

$$-\frac{dC_s}{dt} = KC_s^{n_g} \cdot C_a^{n_a} \qquad (2)$$

where C_s = concentration of floatable solids

C_a = concentration of air bubbles

If aeration is kept constant then Equation (2) simplifies to

$$-\frac{dC_s}{dt} = K C_s^{n_s} \qquad (3)$$

This equation has been investigated at length by a number of workers and the majority agree that $n = 1$.

Assuming $n = 1$, integration of Equation (3) gives

$$C_t = C_o\, e^{-Kt} \qquad (4)$$

where C_o = concentration of floatable material at time zero

C_t = concentration of floatable material at time t.

Taking the log of both sides of Equation (4)

$$\ln \frac{C_o}{C} = K \cdot t \qquad (5)$$

If $\ln C_o/C$ is plotted against t, then the graph should be a straight line of slope K if the assumption that $n = 1$ is correct. In many cases, it appears that some of the floatable material cannot be floated. If this concentration of unfloatable material is expressed as C_∞ then,

$$\ln \frac{C_o - C_\infty}{C - C_\infty} = K \cdot t \qquad (6)$$

These equations are applicable to "batch-type" flotation pro-

cesses and have to be modified for a continuous steady-state industrial process. In this case there is a continuous feed, concentrate, and tailing of constant quality and quantity. This can be considered analogous to a batch cell at some instant of time. For the continuous case

$$K = \frac{C_c \cdot Q_c}{Ct \cdot Vo} \tag{7}$$

where C_c = concentration of floatable species in the concentrate

Q_c = volume flow rate of concentrate

Ct = concentration of floatable species in tailing

Vo = volume of the cell

and fractional recovery is given by

$$R = \frac{Q_c\, C_c}{Q_c \cdot C_c + Q_t \cdot Ct} \tag{8}$$

where Q_t = volume flow rate of tailing.

From these equations it can be deduced that

$$R = \frac{K\lambda}{1 + K\lambda} \tag{9}$$

where λ = nominal residence time

$= \frac{Vo}{Qt}$ if the cell acts as an ideal mixer

If λ is constant, bank recovery can be predicted by

$$R_{(bank)} = 1 - (1+K\lambda)^{-n} \qquad (10)$$

If λ is not constant the individual cell recovery must be predicted separately by

$$R_n = \frac{K\lambda n}{1 + K\lambda n} \left(1 - \sum_{i-1}^{n-1} Ri \right) \qquad (11)$$

A number of objections have been raised to the above equations and some are listed by Roos (1969). The restriction of constant retention time can be overcome if we consider one cell alone under steady state conditions. Here Equation (7) is valid

$$\text{i.e.,} \quad K = \frac{C_c \cdot Q_c}{Ct \cdot Vo} \qquad (12)$$

$$\text{or} \quad C_c \cdot Q_c = K \cdot Ct \cdot Vo$$

where $C_c \cdot Q_c$ = flotation rate (FR)

and $Ct \cdot Vo$ = cell concentration of floatable species (CC)

or $FR = K \cdot CC$

If K is a constant and the kinetics are first order, then a plot of FR vs. CC will give a straight line of slope K. Since, in the case of ideal mixing, the tailing from a cell is a measure of the cell pulp, then CC can be determined from the tailing sample. Thus, one test on a bank can provide a number of values for the flotation rate at varying flow rates and pulp densities.

Validity of assumptions of kinetic model: Kinetic rate equations have been used for a long time to describe chemical equations and their validity, in general, is not in doubt. The basic analogy of a chemical equation to a flotation cell appears to be valid from first principles. Hence the initial equation $-\frac{dC}{dt} = K \prod_{i=1}^{m} C_i^{n_i}$ appears to be correct. As pointed out, it is too complex to handle in this form and simplifying assumptions have to be made. It is the validity of these assumptions which is in doubt.

One major assumption is that reagent concentration is constant through a flotation process, or, if it changes, it does so in a fashion that gives predictable results. Since most flotation reagents are surface active agents, the removal of froth will change the reagent concentration in the remaining pulp. There is evidence to suggest that collector attachment is an irreversible or difficult to reverse process (Gaudin 1957).

This appears to be true in only certain circumstances (Aplan and Fuerstenau 1962). Those cases of physical collector attachment, especially, are sensitive to reagent changes and quickly re-establish equilibrium. Faulkner (1966) shows the interdependence of reagent concentration with such variables as rate of flotation. Bull (1966) also points out the change in flotation rate when a collector is added in the middle of a flotation bank.

A second major assumption is that residence time is constant from cell to cell. This would require, among other things, that the concentrate volume be negligible. While this is true for flotation of low grade sulphides such as molybdenite, it is not true in all cases. The equation can be modified, as shown, to correct for the variable λ. It is pointed out in the section on Mixing Theory that each cell in a bank is unique. Even if mechanical conditions (such as motor speed, cell baffling, and impeller wear) are identical from cell to cell, the variance in the flow regimes will vary the active volume from cell to cell. If the kinetic equation (in some modified form) is to be of use in predicting products to be produced, λ must be constant or vary in a predictable fashion. Neither alternative appears to be correct.

A third assumption is that the flotation rate coefficient is independent of particle size. The kinetic model draws an analogy between the collision mechanism of chemical reactions and bubble-mineral collisions in flotation. The statistical probability of this collision is given by Klassen and Makrousov (1963) as

$$\frac{dX}{dt} = Z \cdot \phi \cdot N_t \ (X_o - X)^n \qquad (13)$$

where Z = coefficient of proportionality

ϕ = coefficient of collision efficiency (the proportion of collisions which result in attachment)

$(X_o - X)$ = number of particles per unit volume capable of attachment

N_t = quantity of bubbles per unit volume capable of mineralization

Equation (13) can be expressed in terms of mineral surface area as

$$\frac{dX_i}{dt} = Z^1 \cdot \phi \cdot \sum_1^n r_i^2 \ N_t \ (Xo_i - Xi)^n \qquad (14)$$

where r_i = radius

$(Xo_i - Xi)$ = number of particles per unit volume of radius (ri) capable of attachment.

Equation (14) could be expanded further to account for bubble surface area. If the rate coefficient for various fractions is assumed, then

$$\frac{dx}{dt} = \sum_{i=1}^{n} \frac{dx_i}{dt}$$

where n = number of size fractions

The coefficient of proportionality (Z) would be a function of the number of bubbles per mineral particle required for flotation. Pictures published by Smith (1968) show in excess of 200 bubbles on one coal particle. Bubble counts done by Smith show that the number of bubbles required for flotation varies from size range to size range. One would expect, therefore, that flotation rates would vary with size. This is confirmed by Roos (1969). If certain fractions are removed in preference to others, then one would expect the bulk flotation rate $\left(= \sum_{i=1}^{n} \frac{dx_i}{dt} \right)$ would vary also as the bulk composition changed.

A fourth problem is that concerned with relative rates of association and dissociation of the bubble-mineral pair. Smith (1968) claims that turbulence in a conventional cell is self-defeating since it dislodges large particles which might otherwise have floated. Arbiter

and Harris (1962) give this as the reason why conventional flotation cells fail entirely to float particles coarser than the 35 to 48 mesh range.

The objections, among others, place the kinetic model on questionable grounds. As yet no commercial operation uses it as a control device or as a planning tool.

Circuit Design

Flotation plant operation is often considered more of an "art" than a science. Procedures used in a mill often depend on such limiting factors as mineral type or size of liberation but they often depend, also, on the personal experiences (and often biases) of the mill's metallurgists and operators. An article by Draper and Bush (1966) giving the history of the changes in the Pb-Zn circuit at Mount Isa Mines is an excellent example of the way in which circuit design changes with advances in technology, and withchanges in opinion.

Some generalized rules exist and some of these are summarized by Dorenfeld (1962). Aspects to be considered in any design include such factors as size of liberation, pulp density, flotation time, type of water and uniformity of ore. When all such factors are taken into consideration, the designed circuit is unique.

In this study of the Tahawus mill, factors of size and chemical control of flotation feed pulp were not investigated. Main consideration was given to the relative effect of pulp dilution and the effect of varying the circulating load. Some generalized guidelines exist for these areas. Gaudin (1957) suggests that for maximum collector attachment and minimum reagent consumption, conditioning should be done at very high pulp densities. The upper limit would be set by the appearance of plastic properties where proper mixing is impossible. In the Tahawus mill, conditioning takes place at 70 percent solids which approaches this upper limit.

Baarsen and Parks (1954) suggest that for maximum recovery using fatty acid collectors, pulps should be in the order of 30 to 40 percent solids. In cleaning operations where selectivity, not recovery, is important, pulps of 15 percent solids or less should be used. In the Tahawus mill the cleaning operations are carried out at pulp densities of 30 to 40 percent solids. Thus a method of obtaining selectivity in this operation is not being utilized. Cleaner bank tailings were less than 10 percent solids in the Tahawus mill. This gives a calculated circulating load of less than 20 percent. By judicious use of pulp dilution and larger circulating loads, greater selectivity should have been possible.

EXPERIMENTAL RESULTS

Three different aspects of the flotation process were investigated in this program. Each of these is reported under a separate heading--impulse tracer tests; flotation tests; and flotation kinetics.

Impulse Tracer Tests

Eight separate impulser tracer tests were conducted and a summary of the flow conditions for each is given in the following table.

Table 1. Summary of Flow Conditions for Impulse Tracer Tests

Test No.	Cell Type	Flow Rate (liters/minute)	Nominal Retention Time min	Aeration Rate (scfm)	Impeller Speed (rpm)
1	Agitair No.15	86.7	0.5	-	1050
2	Agitair No.15	43.3	1.0	-	1050
3	Agitair No.15	39.7	1.0	6	1050
4	Agitair No.15	21.7	2.0	-	550
5	Agitair No.15	43.3	1.0	-	550
6	Agitair No.15	86.7	0.5	-	550
7	Denver No.24	32.2 cu ft/min	1.5	-	Not measured
8	Denver No.30	62.2 cu ft/min	1.61	-	Not measured

The results for these tests are included as Section I of the Appendix. The results are plotted on Figures 8 to 15 as actual con-concentration (= C) versus dimensionless time (= $\frac{UT}{Vo}$ or Θ). These graphs were then compared to a theoretical curve plotted on a dimensionless time ordinate with the same scale. The shift in the mean arrival time was then used as a measure of mixing efficiency in the manner described by Bull and Spottiswood (1970).

Flotation Tests

The aim of this work was to provide answers to the following questions:

a) What is the effect of the operating variables?

b) What is the "maximum" grade possible?

c) How many stages are required to maintain a final grade of 45% TiO_2?

d) What recovery can be achieved at this grade?

e) What is the optimum circuit required to achieve this final grade?

In order to obtain sufficient data to answer these questions, the following types of tests were undertaken.

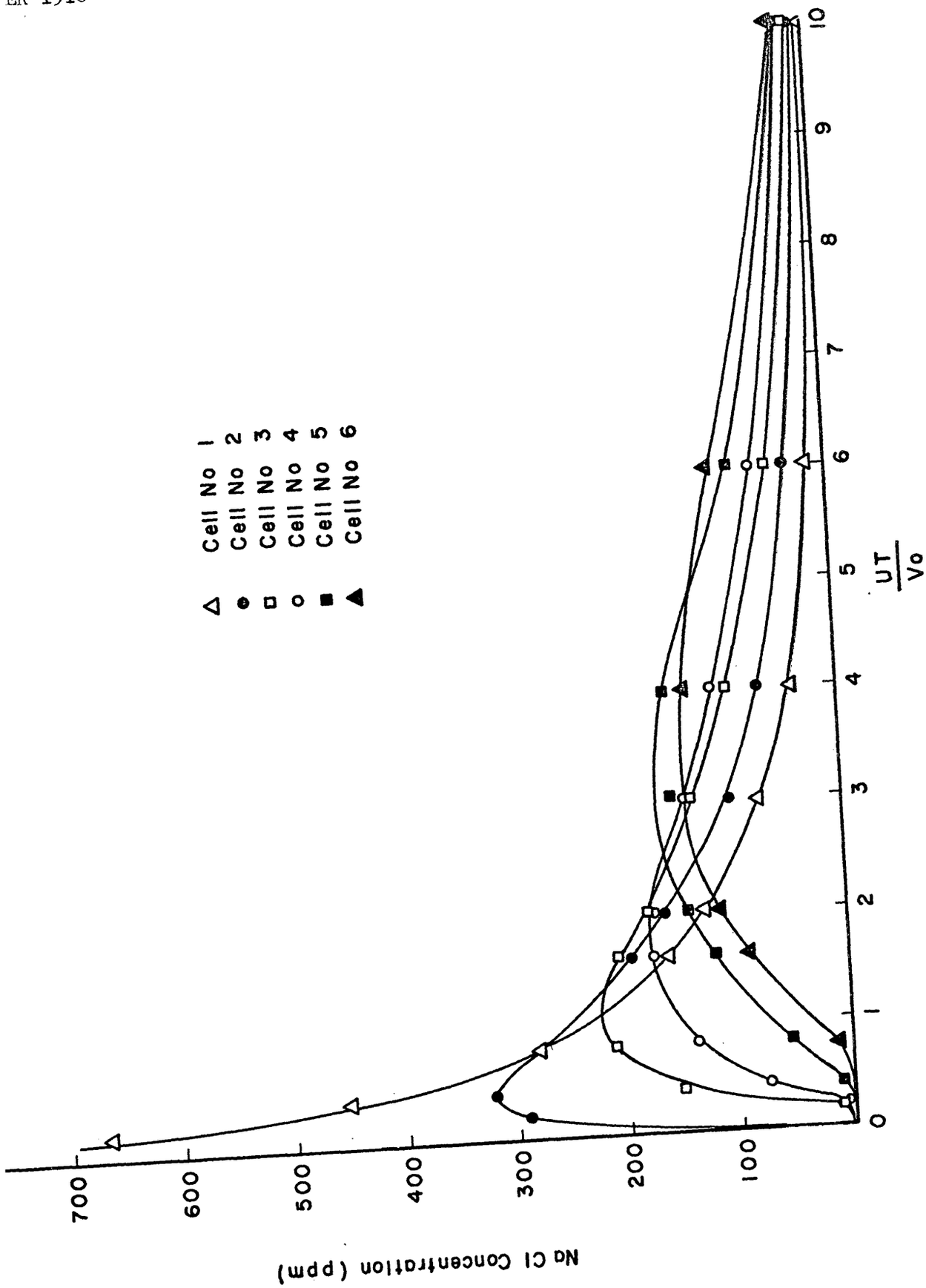

Figure 8. Impulse Tracer Test No. 1

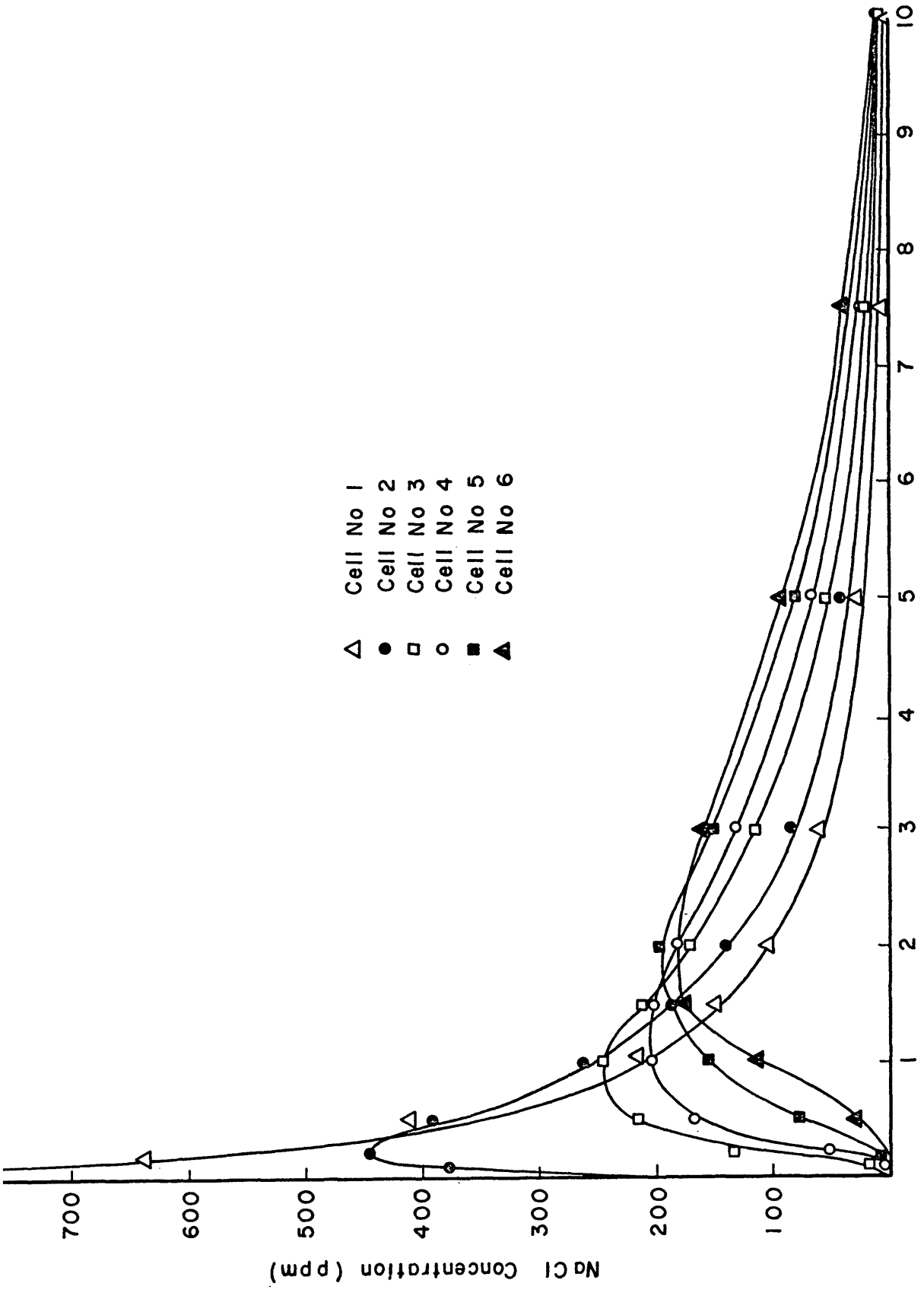

Figure 9. Impulse Tracer Test No. 2

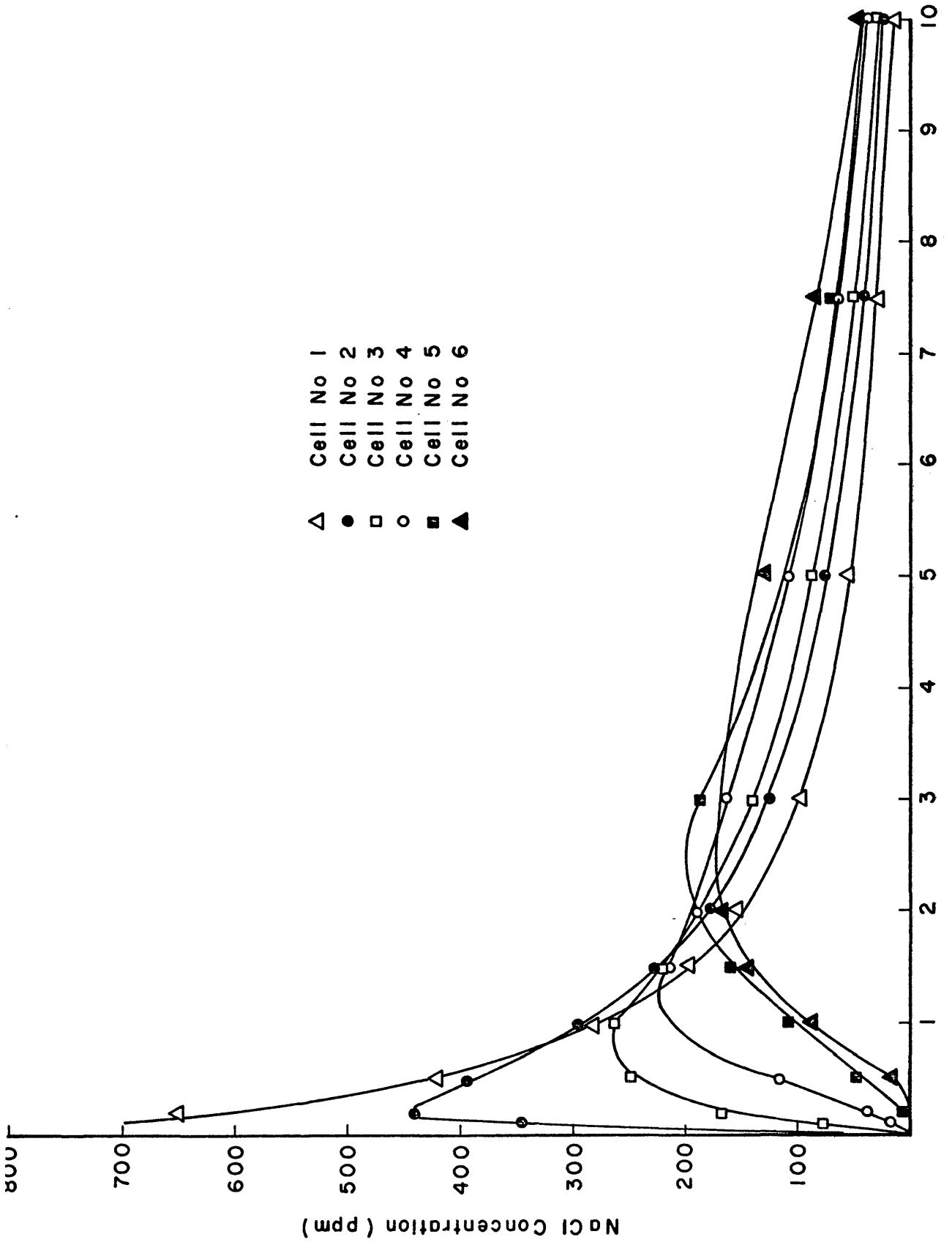

Figure 10. Impulse Tracer Test No. 3

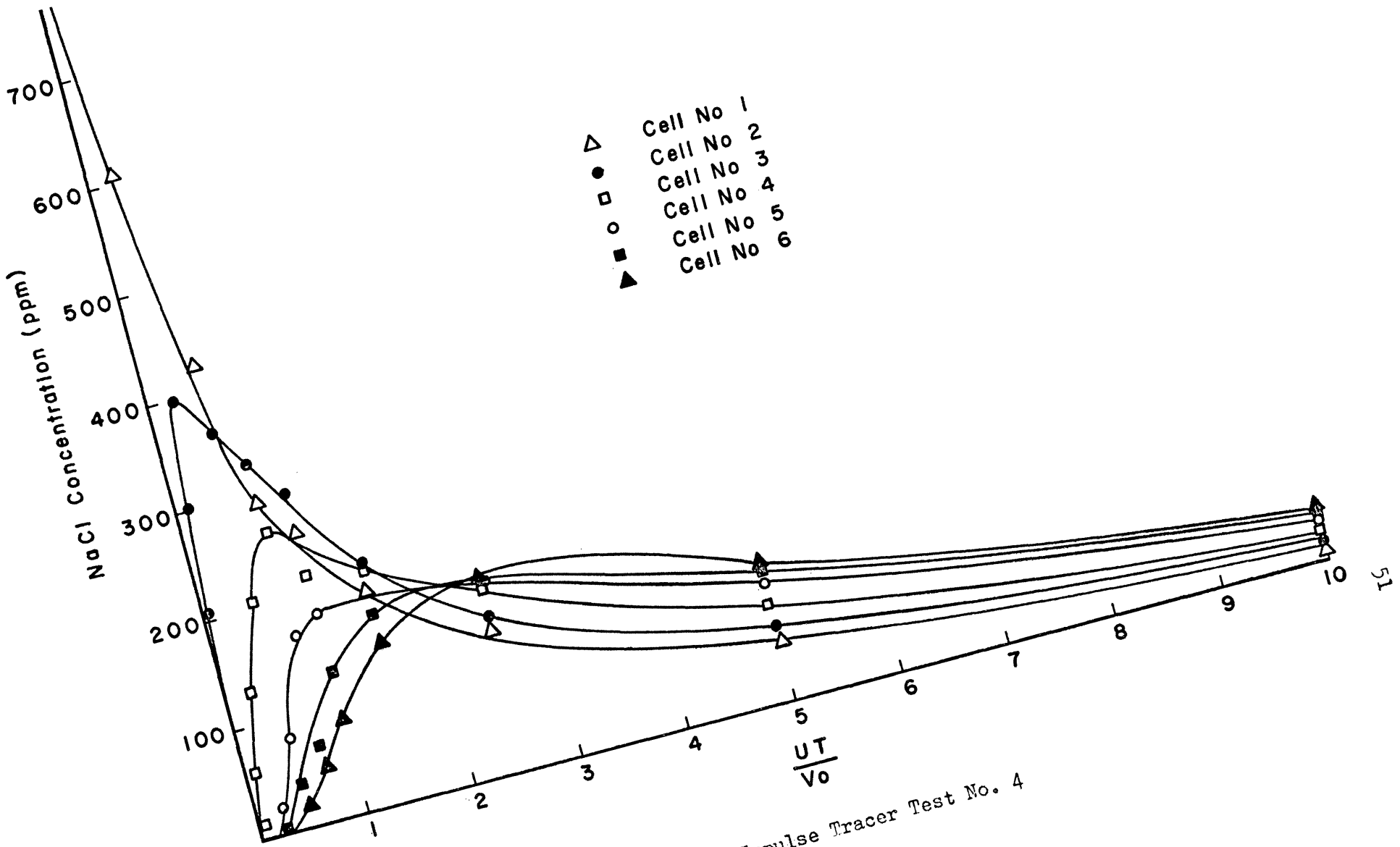

Figure 11. Impulse Tracer Test No. 4

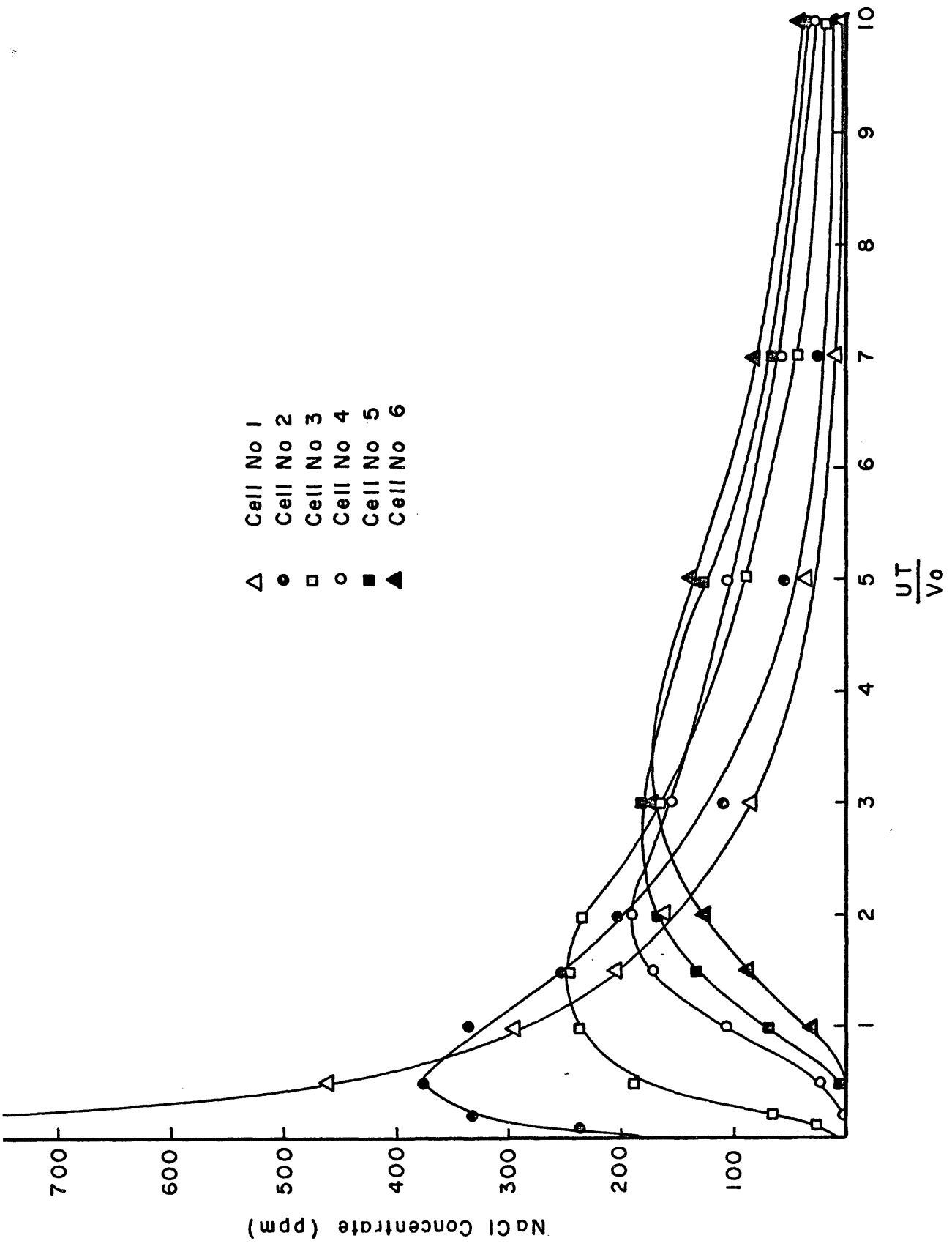

Figure 12. Impulse Tracer Test No. 5

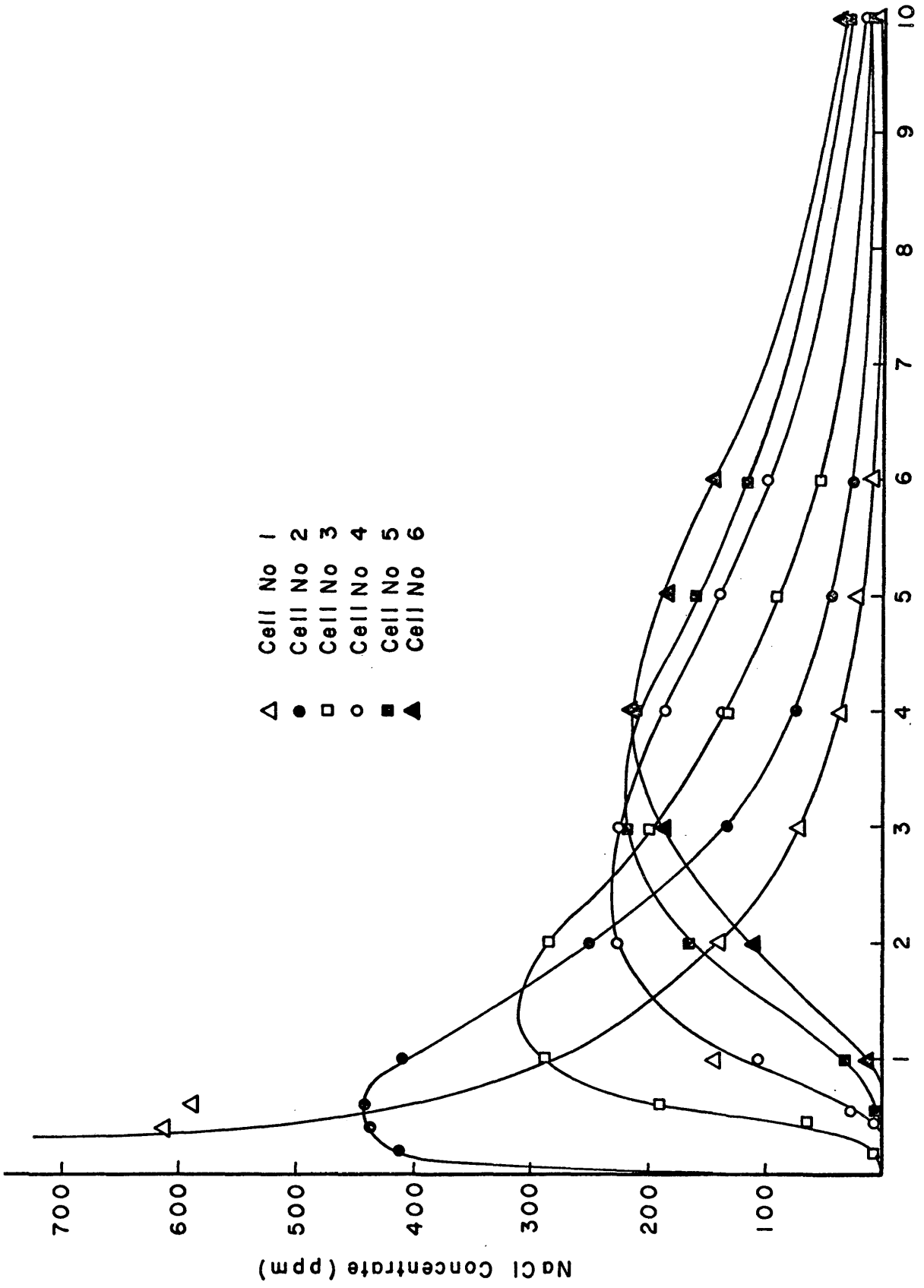

Figure 13. Impulse Tracer Test No. 6

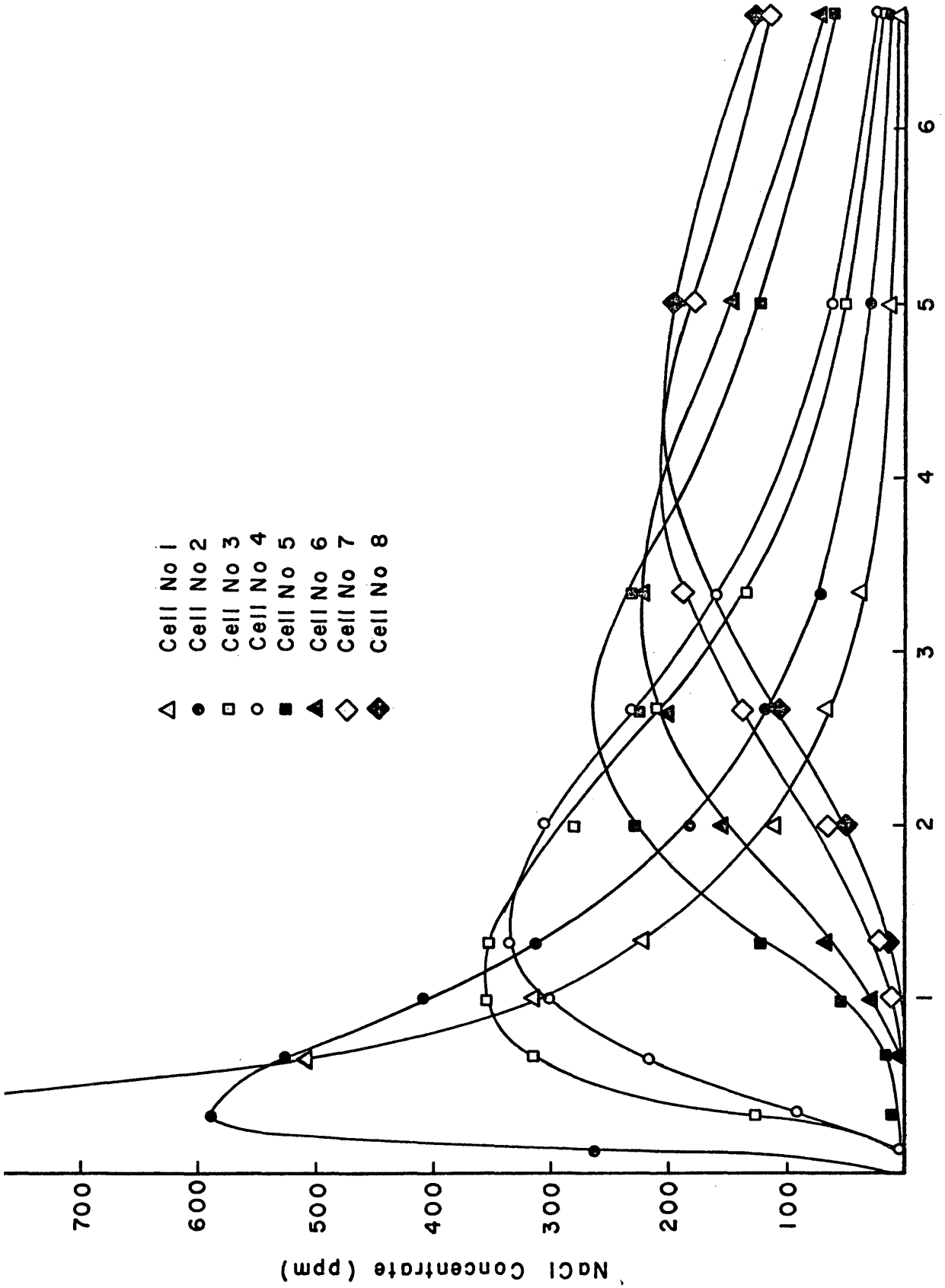

Figure 14. Impulse Tracer Test No. 7

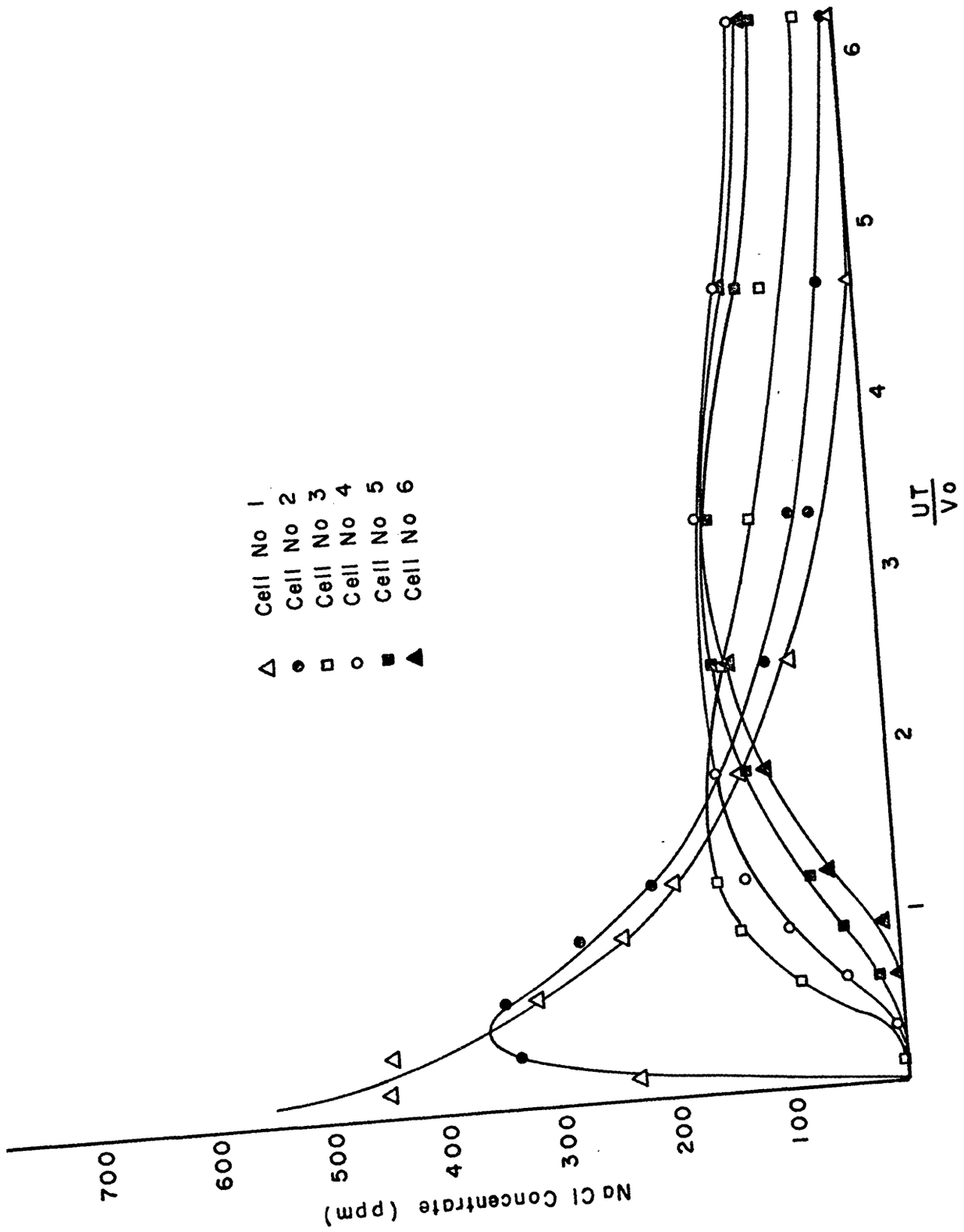

Figure 15. Impulse Tracer Test No. 8

a) Tests on the rougher bank in open circuit.

b) (i) Tests on the cleaner bank in open circuit.
(ii) Tests on the cleaner bank in closed circuit.

c) Tests on the recleaner bank in closed circuit.

Results of all these tests are included in Section II of the Appendix.

Flotation Kinetics

Three tests were carried out in which physical operating conditions from cell to cell were identical. In each of these tests the flotation rate constant of TiO_2 was calculated. For one of the tests (Test 18) the concentrates and tailing were screened and the flotation rate constant for TiO_2 for each screen fraction in each cell was calculated.

Since TiO_2 can occur in ilmenite as well as some of the gangue minerals present, a TiO_2 analysis is not a mineralogical analysis. These mineralogical concentrations were determined by x-ray analysis of the samples used for TiO_2 analysis. Flotation rate constants of ilmenite in the unsized samples were then calculated.

All the data for these calculations is included in Section III of the Appendix and some results are plotted on Figure 24.

DISCUSSION OF EXPERIMENTAL RESULTS

Impulse Tracer Tests

For quantitative discussion of mixing efficiency Levenspiel (1962) defines dead space in a mixing vessel as the space occupied by liquid which stays in the vessel longer than three times the theoretical mean residence time. Thus actual mean residence time ($= \bar{\Theta}a$) for the actual region of one vessel is given by

$$\bar{\Theta}a = \int_0^3 \Theta \; C(\Theta) \, d\Theta \tag{15}$$

and the true mean residence time is given by

$$\bar{\Theta} = \int_0^3 \Theta \; C(\Theta) \, d\Theta + \int_3^\infty \Theta \; C(\Theta) \, d\Theta \tag{16}$$

but $$\Theta a = \frac{Va}{U} \tag{17}$$

and $$\Theta = V/U \tag{18}$$

Hence $$\frac{\Theta a}{\Theta} = \frac{Va}{V} \tag{19}$$

or $$Va = V \cdot \frac{\Theta a}{\Theta} \tag{20}$$

Mixing efficiency is then defined by

$$\epsilon = 100 \frac{Va}{V} \qquad (21)$$

For the impulse tracer tests Θa for the sixth cell was found by counting the squares under the curve between the limits of 0 and 18 on the dimensionless time ordinate. Va was calculated by substituting a value of $\Theta = 6$ into Equation(20). The following table is a summary of the results.

Table 2. Summary of Mixing Efficiencies in the Sixth Cell.

Test No.	Theoretical Mean Residence Time	Actual Mean Residence Time	Va (liters)
1	6	4.88	35.2
2	6	3.60	26.0
3	6	4.85	32.1
4	6	4.47	32.2
5	6	4.85	35.0
6	6	4.84	34.9

The conditions for each of these tests is given in Table 1 and the data sheets are included in Section I of the Appendix. The curves from which they were computed are given as Figures 8 to 13. The significance of these results is discussed in the following sections.

Mixing Efficiency versus Flow Rate: The change in mixing from Test No. 1 to Test No. 2 is caused by a change in flow rate from 86.7 liters per minute to 43.3 liters per minute. The change from Test No. 4 to Test No. 6 occurred when the flow rate changed from 86.7 liters per minute to 21.7 liters per minute. These results show mixing efficiencies are greatest at highest flow rates and support the findings of Roos (1969) and Spottiswood (1970).

Mixing Efficiency versus Impeller Speed: Test Nos. 1 and 6 have identical retention times (=0.5 minute) and Test Nos. 2 and 5 have identical retention times (=1.0 minute). At the higher flow rate there is no difference in mixing efficiency. At the lower flow rate there is a wide variance.

Mixing Efficiency versus Air Flow Rate: Test Nos. 2 and 3 have identical water flow rates but in No. 3 the air was turned on. The volume of the cell occupied by air was measured and the curves have been compensated for this effect. The variance is due only to the air.

The results show much higher mixing efficiencies when the air was turned on. Spottiswood (1970) also reports this change but the

variance was not as large.

Mixing Efficiency versus Cell Number: Visual examination of the graphs shows Tracer Test No. 6 (Figure 13) has the smoothest curves. For this graph the mean arrival times were computed with the following results.

Table 3. Summary of Mixing Efficiencies in Test 6.

Cell No.	Theoretical Mean Residence Time (Θ)	Actual Mean Residence Time (Θa)	Va
2	2	1.88	40.7
3	3	2.45	35.3
4	4	3.58	38.8
5	5	4.23	36.6
6	6	4.84	34.9

These values show that mixing efficiencies vary from cell to cell with virtually no systematic variation. This suggests that mixing regimes in each cell of the bank are unique.

Validity of Mixing Efficiency Parameters: Mixing efficiencies, as defined, are virtually identical for Tests 1, 3, 5, and 6. Visual inspection of their C-curves shows that this does not appear to be the case. Departure of the peak arrival time (rather than the mean)

from the theoretical can be used as a semi-quantitative measure of mixing efficiency. A comparison of peak arrival times is made in Table 4.

Table 4. Summary of Peak Arrival Times

Cell No.	Theoretical Peak Arrival Time	Actual Peak Arrival Time			
		Test 1	Test 3	Test 5	Test 6
2	1	0.4	0.2	0.5	0.6
3	2	1.1	0.9	1.5	1.4
4	3	1.8	1.3	2.0	2.4
5	4	3.1	2.5	2.7	3.3
6	5	3.5	2.6	3.4	3.9

These figures show that there is a wide variation in curve shapes despite the apparent similarity in mixing efficiencies.

In a Gaussian distribution curve the mode (or peak height) and mean should be identical. A difference between the peak and mean arrival times would indicate an asymmetric curve. In these impulse tracer test curves the asymmetry is a function of the material trapped in the cell. The following table gives a comparison of peak and mean arrival times for Cell No. 6 in Test Nos. 1, 3, 5, and 6.

Table 5. Comparison of Peak and Mean Arrival Times

Test No.	Peak Arrival Time	Mean Arrival Time
1	3.5	4.88
3	2.6	4.85
5	3.4	4.85
6	3.9	4.84

These figures show there is a wide variance in the asymmetry of these curves despite the apparent similarity in mixing efficiencies.

The approach to the determination of mixing efficiency, as used in this thesis, is a very simplified one. All deviant flows are lumped together and no attempt has been made to quantitatively determine the component flows shown in Fig. 7. It was impossible to do this with the available data. The Reynolds Number (=Re) for this cell is of the order of 5×10^7. This is in the region of extremely turbulent and unstable flow. Hence if it were possible to quantitatively determine the values of these deviant flows, the results would be suspect. The flow regimes would be quite unstable and would probably tend to change with time.

Laboratory Cell Mixing versus Commercial Cell Mixing: Insufficient samples were taken to allow calculation of mean residence times in later cells of the mill bank. Hence comparison of mixing characteristics is based on peak arrival times only. These are shown in Table 6.

Table 6. Comparison of Peak Arrival Times in Laboratory and Commercial Cells

Test No.	Cell Type	Actual Peak Arrival Times Cell No. 2	Cell No. 3	Cell No. 4	Cell No. 5	Cell No. 6
6	Agitair No.15	0.6	1.35	2.1	3.3	3.9
7	Denver No.24	0.3	1.2	1.4	2.7	3.3
8	Denver No.30	0.4	1.5	2.5	2.8	3.3

These data show that mixing in the Denver No. 30 (or 100 cu ft cells) is comparable to the mixing in the Agitair units. The nominal cell retention time in Test No. 6 was 0.5 minute and in Test No. 8 it was 1.61 minutes. Presumably mixing efficiency in these large commercial units would increase as the flow rate increased. These data also show that mixing in the smaller commercial cells was poorer than in the larger cells.

Spottiswood (1970) showed that mixing efficiency varied with baffle type used with unbaffled cells giving the poorest mixing. The commercial cells used at Tahawus had no baffles. Presumably mixing efficiency would improve in these large cells if baffles were introduced. However since commercial cell mixing was comparable to the best of the pilot plant results, baffling would be superfluous.

Significance of results: These results have significance for engineering design studies and for the development of a mathematical model based on a kinetic approach. The latter is discussed in the section on flotation kinetics.

The results of this work indicate that mixing occurring in a bank of cells is a highly complex process and data obtained from impulse tracer test studies is a net effect of all these interacting flows. Each cell in the bank is unique in the matter of mixing regimes. Mixing efficiency, as defined for a single cell in a bank has virtually no meaning. This parameter does not take into account all these interacting flows.

A flotation bank then should be thought of as an integrated unit and not as a number of individual cells. Since this short cir-

cuiting is of such magnitude, there is a minimum number of cells required in a bank to prevent material exiting before being mixed. The Denver Equipment Company suggests at least 4 cells per bank. The data from the Agitair No. 15 cells suggest at least 6 are required.

Ilmenite Flotation Tests

Preparation of pulp: X-ray analysis of the flotation feed showed the following minerals to be present -- plagioclase, biotite, ilmenite, garnet, a clay, and magnetite. This material assayed approximately 20 percent TiO_2 or 38 percent ilmenite. As shown in Figure 4, the ore was ground by ball milling in closed circuit with Ty-Hukki classifiers to give a flotation feed of minus 65 mesh. Details of chemical preparation of the pulp are given in "Analytical Methods and Procedures".

A cell by cell analysis of the pilot plant rougher bank was made and each of the cell concentrates was screened. Each screen fraction was then sent for assay. The recoveries in the individual size fractions were then computed and are given in Table 7. Complete data sheets are given on A 12 to A 18 in Section II of the Appendix.

Table 7. Summary of Screen Assay Tests on Rougher Bank Concentrates

Size Fraction		Feed Weight %	Feed TiO_2 %	Concentrate Weight %	Concentrate TiO_2 %	Tailing Weight %	Tailing TiO_2 %	Recovery %(1)
	+48	0.7	1.3	--	--	1.4	1.3	0
-48	+65	3.8	4.2	0.8	25.5	7.0	1.7	63
-65	+100	9.2	8.5	3.4	34.3	15.3	2.5	76
-100	+150	14.1	16.2	10.0	39.3	18.3	3.1	88
-150	+200	16.9	22.7	17.5	40.3	16.2	3.1	93
-200	+270	10.2	24.7	11.4	39.8	9.0	2.6	95
-270	+325	9.4	21.7	9.9	38.1	8.7	2.4	95
-325		35.7	23.6	47.0	33.6	24.1	3.1	93

1/ Recovery is the percentage of TiO_2 in that screen fraction which reports to the concentrate.

Table 7 shows that coarser fractions of the feed tend to have lower TiO_2 assays and lower TiO_2 recoveries. Microscope analysis of these fractions shows a large proportion of locked gangue-ilmenite particles and only a small proportion of free ilmenite grains. The assays of the feed material also show a lack of TiO_2 enrichment in the -325 fraction. This shows that there is no preferential sliming of ilmenite and hence size of grinding is determined by the flotation recovery achieved and the subsequent handling problems.

The data in Table 7 indicate that best recoveries occur in the 150 mesh range. If the ore were ground to -150 mesh better recoveries would be achieved. However the Tahawus concentrates must be shipped by railroad to the National Lead refineries. If the ore were ground to -150 mesh, a much higher proporiton of the concentrate would be -325 mesh. This would cause a considerable materials handling problem.

It was concluded that there should be no change in the grinding circuit.

Reproducibility of results: It has been found from previous plant experience at Tahawus that mill performance is an extremely variable quantity. Flotation test results are very dependent on such factors as mixing time, pulp density during mixing, temperature of mixing and reagent strengths. It is also dependent on such outside factors as the position in the orebody that the ore comes from or length of time the ore has been exposed to the atmosphere on the coarse ore stockpile. It was found, as a result, that pilot plant results were almost impossible to reproduce at a later date even though the tests were conducted under apparently similar conditions. Therefore it was impossible to completely isolate a single variable and

study its effect. Table 8 shows some results from tests on the rougher bank in open flow. These tests were conducted over a period of one month during which there was a stockpile change.

Table 8. Summary of Some Rougher Bank Tests

Test No.	Feed Pulp Density	Volumetric Flow (l/min)	Concentrate Assay (TiO_2 %)	TiO_2 Recovery %
1	56.7	32.7	36.3	95.7
3	42.0	39.4	36.7	93.4
2	34.2	43.0	39.4	85.1
4	33.1	39.2	33.6	96.5
5	28.4	39.4	40.3	89.2
6	24.6	38.5	33.4	92.9

It was found that the only way to obtain consistent results was to complete a series of tests very close to each other in time. The samples had to be stockpiled and processed when the series of tests was complete.

Effect of Variables on Rougher Bank Performance

The aim of this work was to determine the effect of changing the physical operating variables. No change in the chemical conditions was made. It was found that there are three main operating

variables which can be used to control flotation performance. They are

(a) volumetric flow rate

(b) pulp density

(c) air flow rate

The effect of volumetric flow on rougher bank performance is shown in Figures 16 and 17. Complete results are included as Flotation Test Nos. 7 to 15 on A 25 to A 33 in Section II of the Appendix.

The effect of pulp density on rougher bank performance is shown in Figures 18 and 19.

The effect of air flow rate on rougher bank performance is shown in Figure 20. Complete results are included as Flotation Test Nos. 16 to 18 on A 34 to A 36 in Section II of the Appendix. A feature of these results was the constancy of the cell concentrate grade. A comparison of concentrate grades from Test 16 with cell grades from Test 3 is made in Table 9.

"Upgrade Ratio" in the following discussion is defined as $\frac{\text{concentrate grade}}{\text{feed grade (computed)}}$.

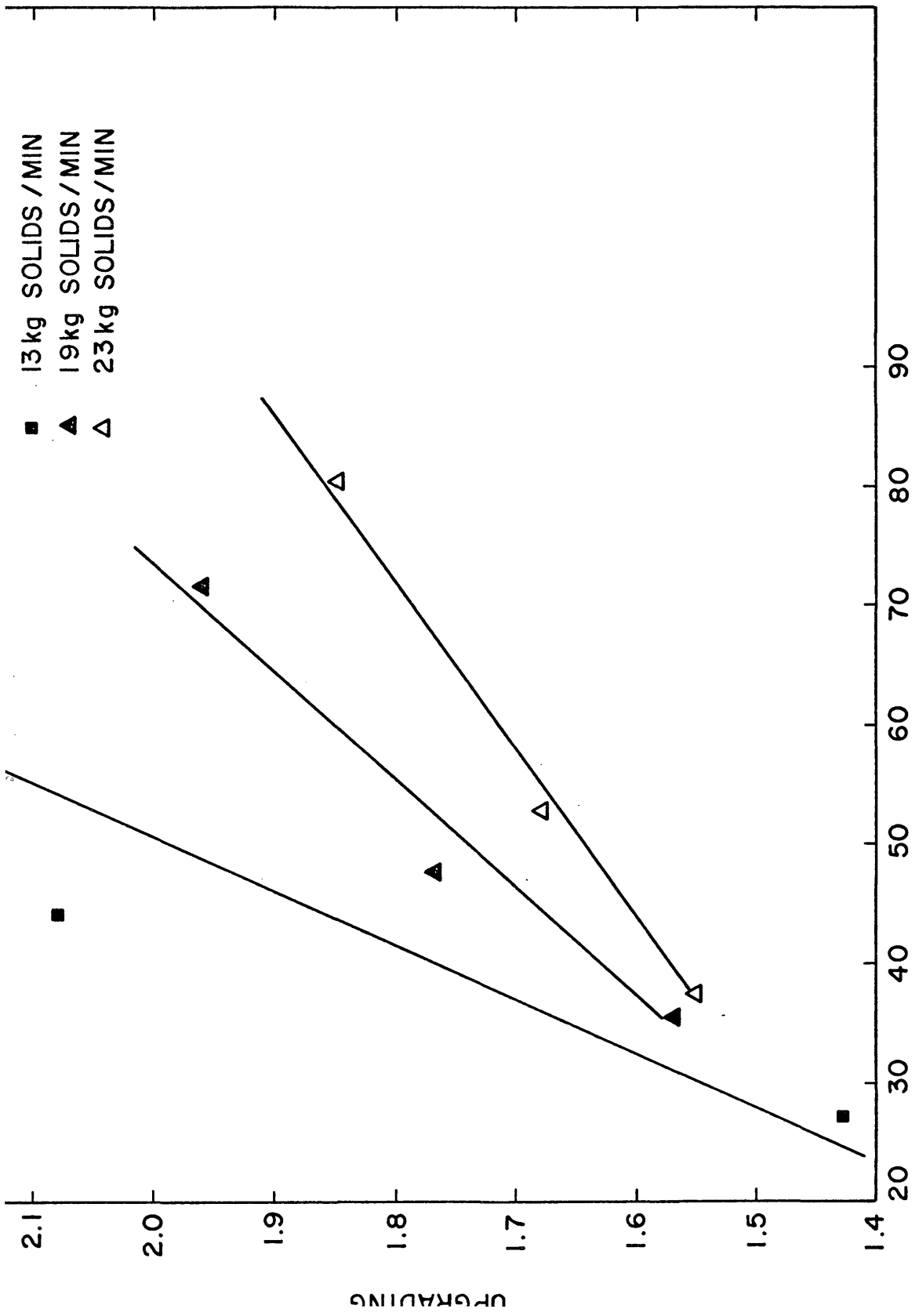

Figure 16. Volumetric flow (l/min) versus upgrading

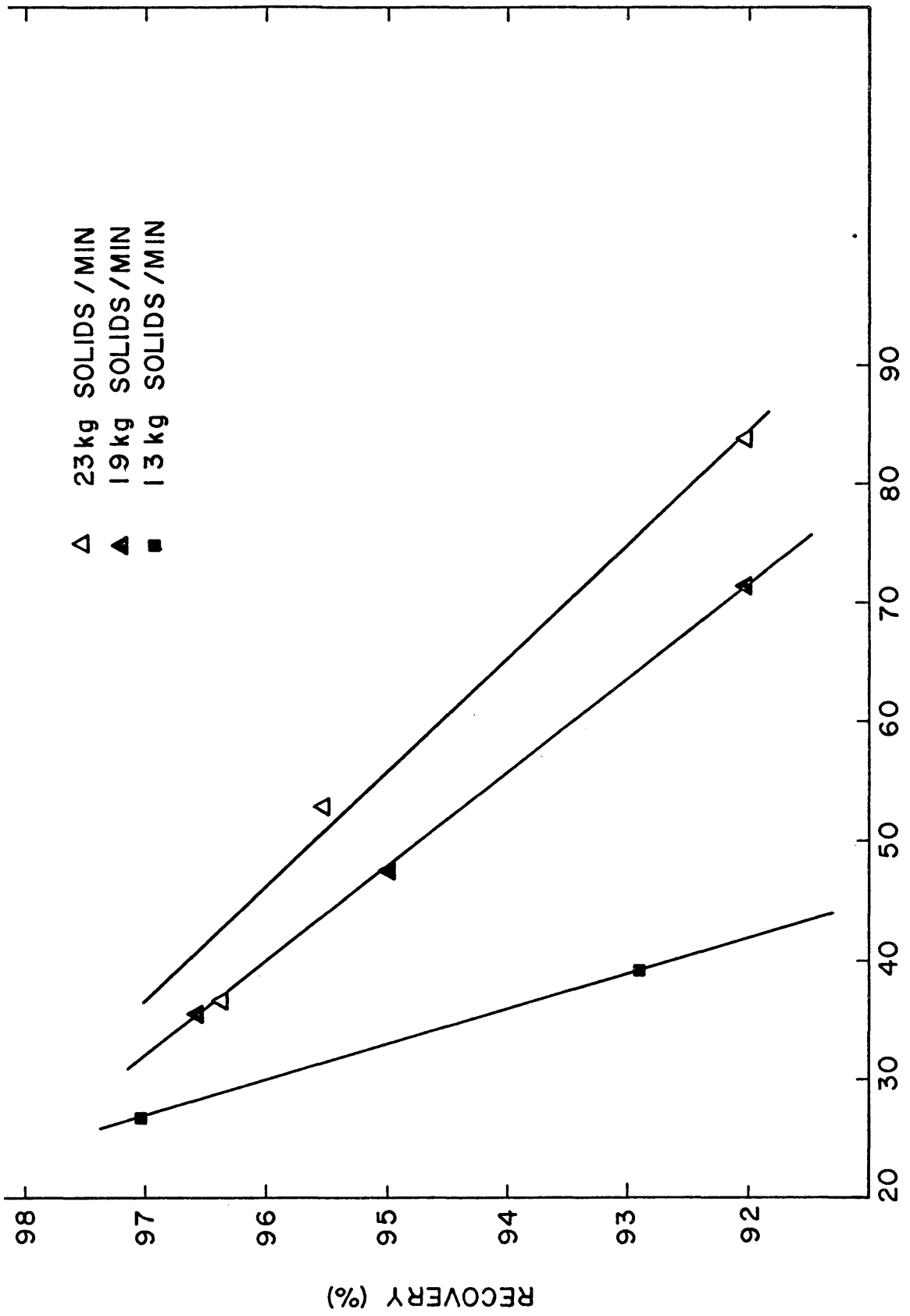

Figure 17. Volumetric flow (l/min) versus recovery (%)

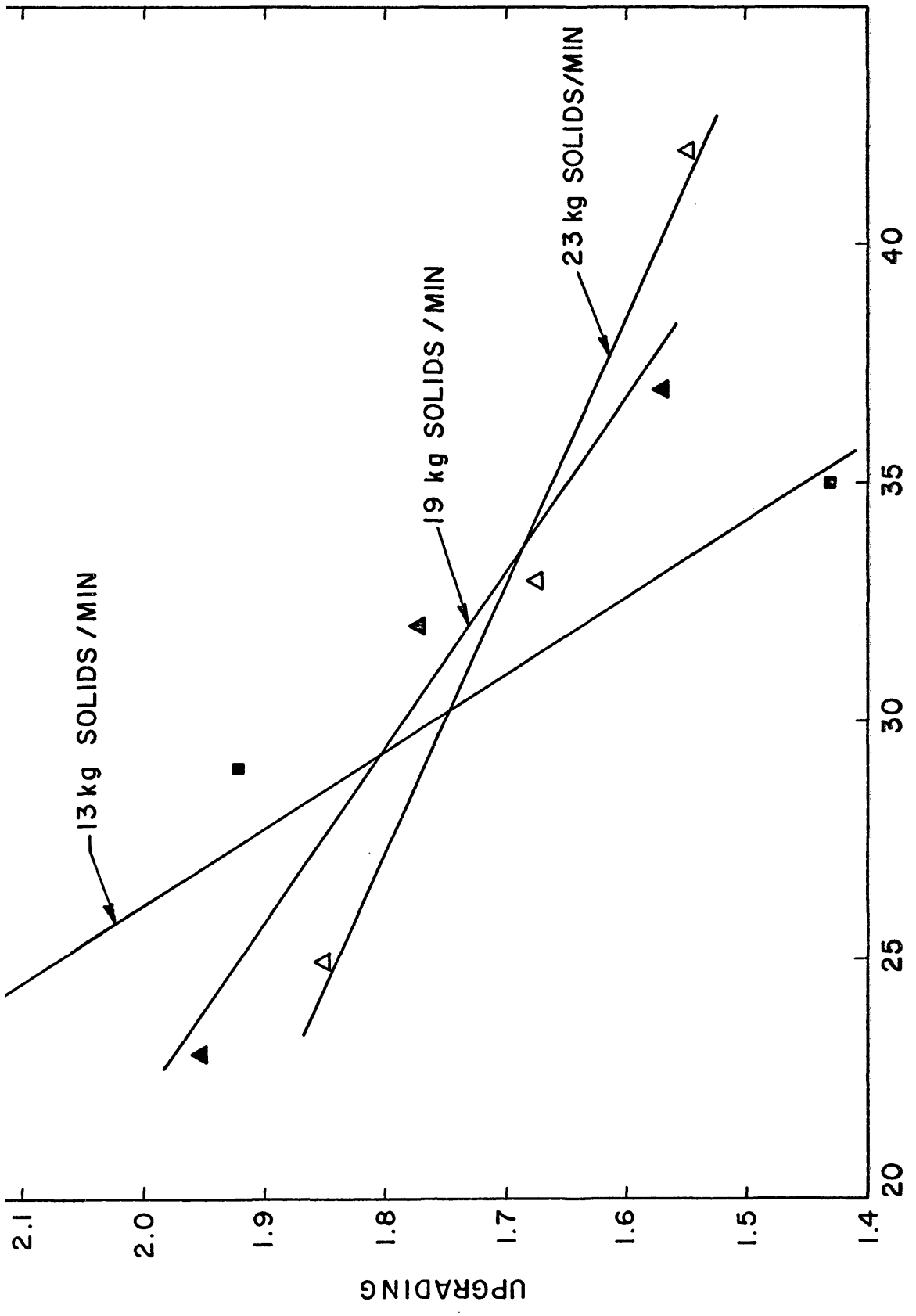

Figure 18. Pulp density (%) versus upgrading

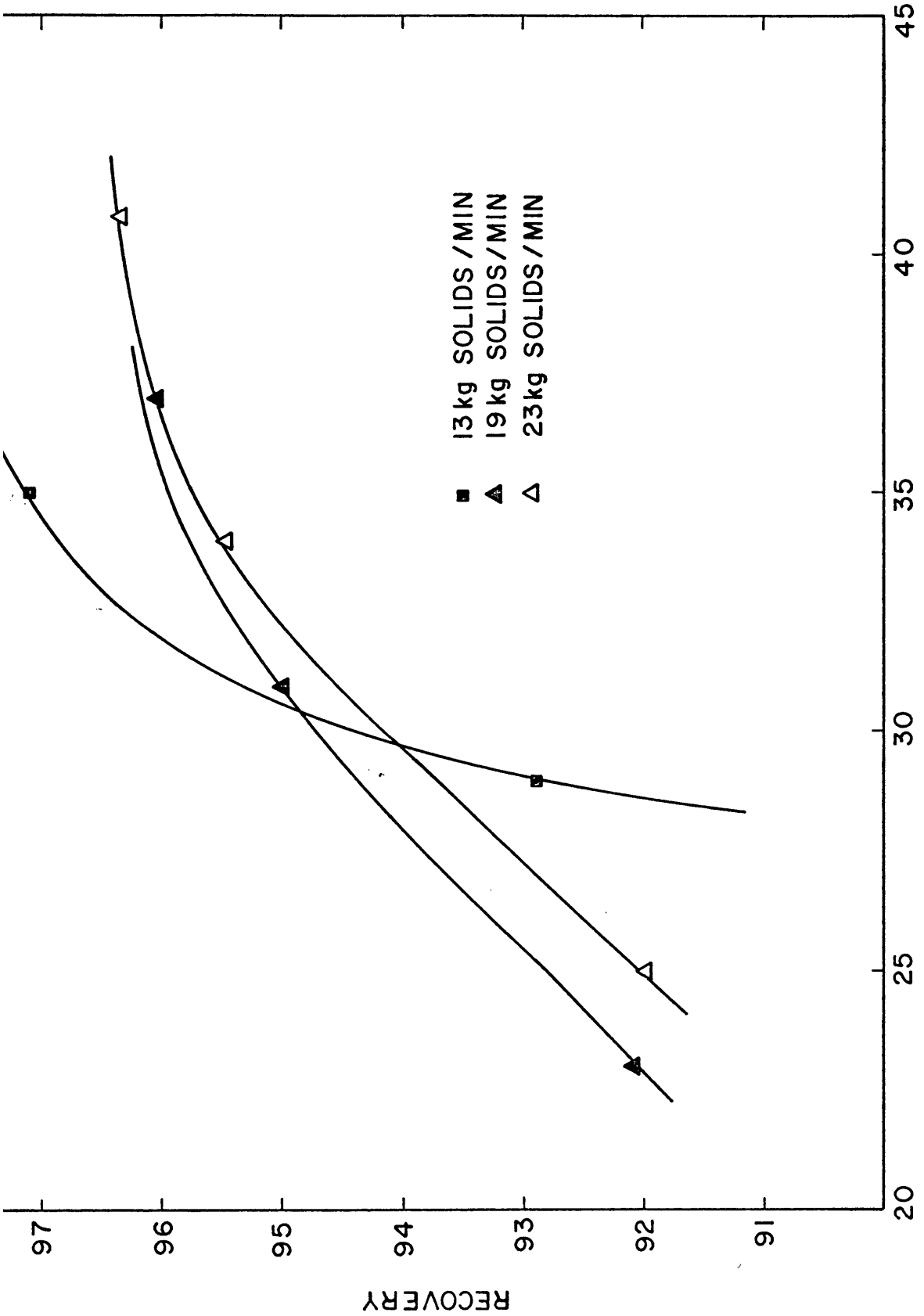

Figure 19. Pulp density (%) versus recovery

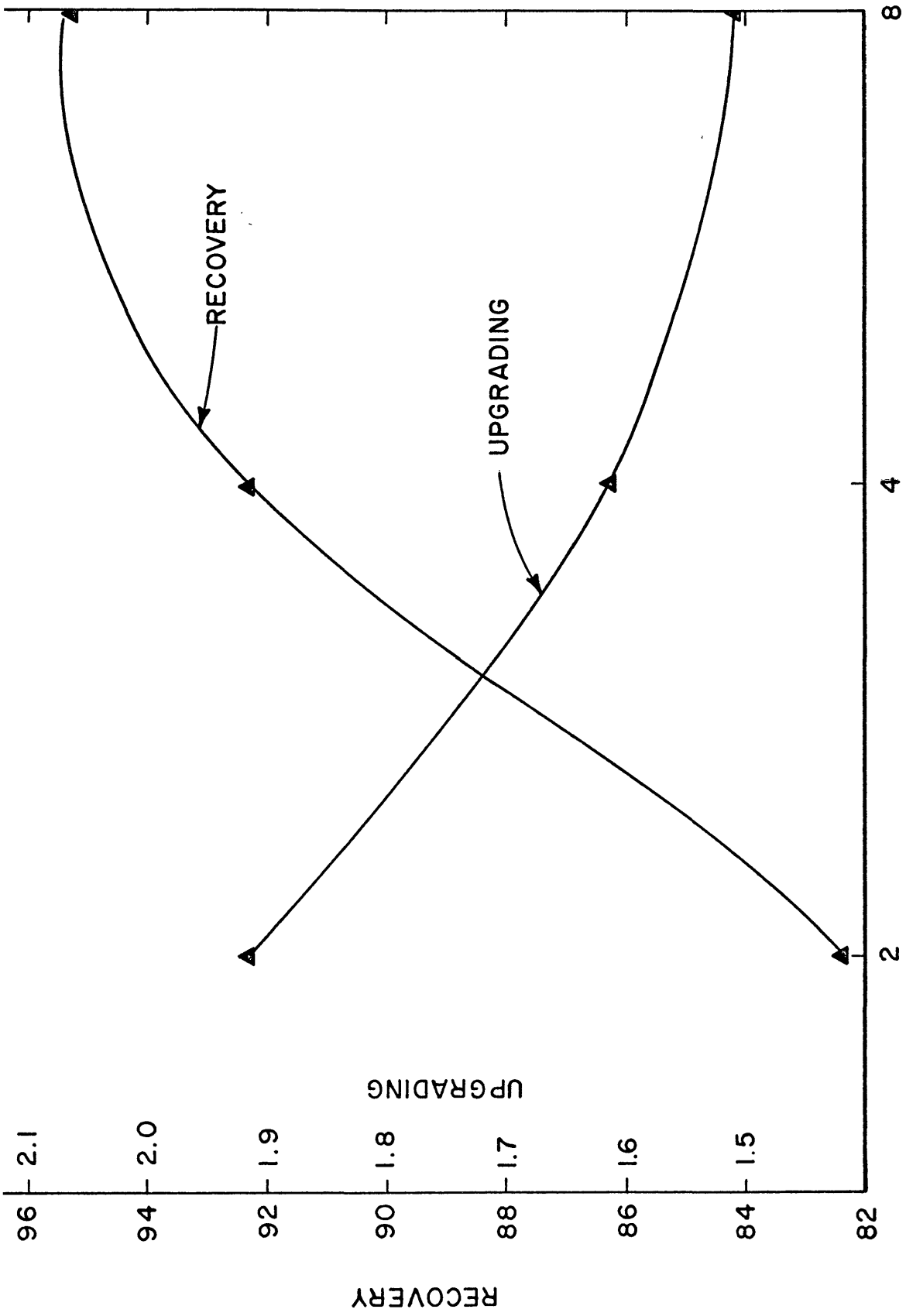

Figure 20. Air flow (scfm) versus upgrading and recovery (%)

Table 9. Comparison of Cell Concentrate Grade using Different Air Flow Rates

Cell No.	Cell Concentrate Assay (TiO_2 %) Test No. 3	Test No. 16
1	39.9	34.3
2	37.1	34.2
3	35.2	36.7
4	26.8	37.6
5	22.8	35.9
6	20.3	--
Average	36.7	34.3

In order to select conditions for the operation of a flotation rougher bank, preset performance criteria had to be used. These were set at 95% or better recovery and an upgrade ratio of 2.0 or better. To achieve these, the following operating conditions were needed.

Nominal retention time - 5 minutes
Feed pulp density - denser than 30%
Air flow - greater than 1.8 cu ft/kg dry feed

Selection of Number of Cleaning Stages

A number of tests were conducted with one or two stages of cleaning in open or closed circuit. Some of these results are plotted on Figure 21.

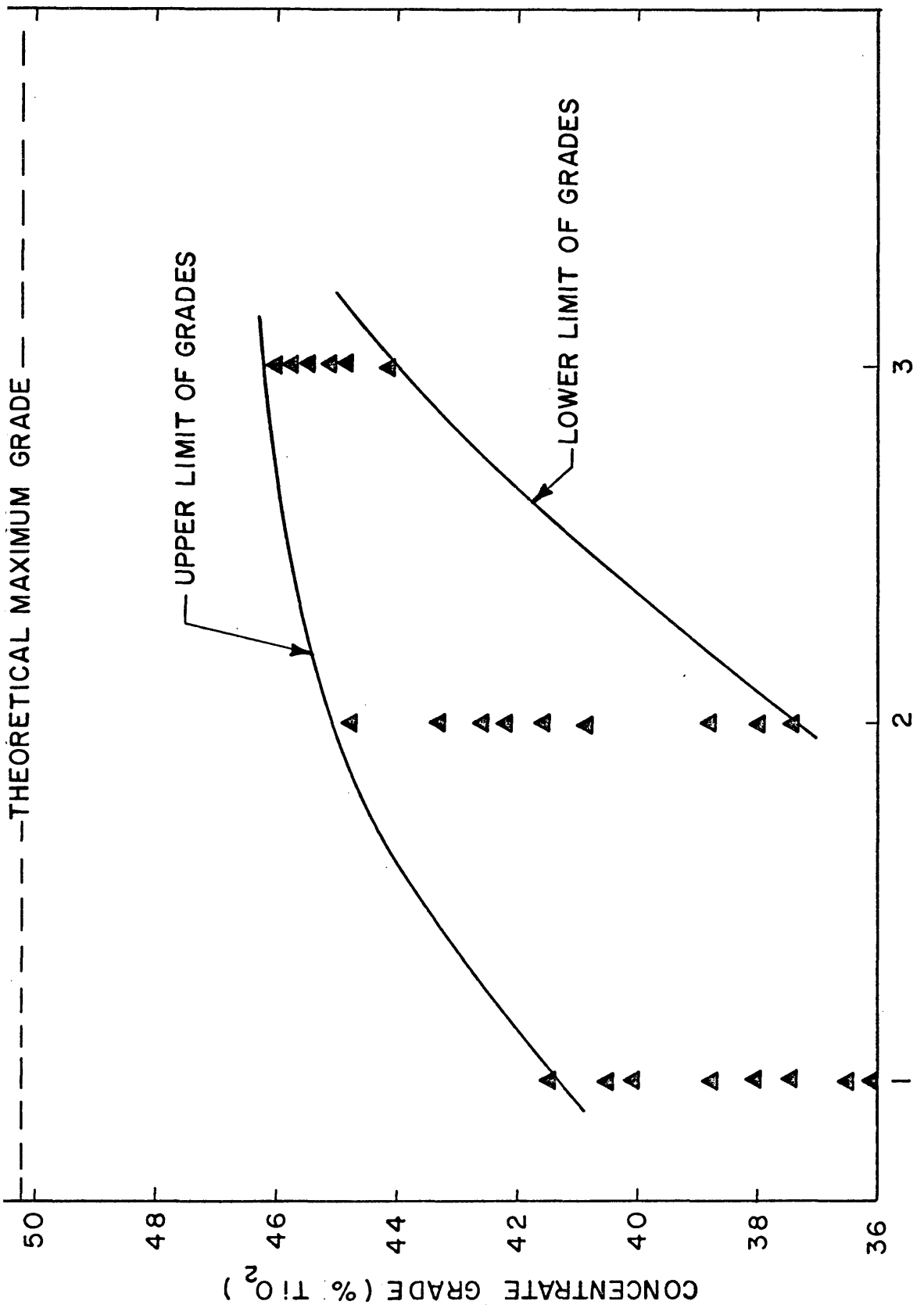

Figure 21. Concentrate grade versus number of stages of flotation

Figure 21 shows concentrate grade at successive stages converging towards an asymptote of 50.5% TiO_2. How many stages would be required to reach this theoretical maximum cannot be determined by extrapolation. However the National Lead refineries currently accept grades of 45% TiO_2 or better. Figure 21 shows this can be achieved with two stages of cleaning.

Prediction of recovery: Considerable difficulty was experienced in pumping the tailing of the final cleaner bank. This material foamed so readily that it was virtually impossible to pump in a conventional centrifugal pump. Three flotation tests were completed with the recleaner bank in closed circuit. The results of these are included as Flotation Test Nos. 24 to 26 on A 42 to A 44 in Section II of the Appendix. They are summarized in Table 10.

Table 10. Summary of Assays from Test Nos. 24 to 26

Test No.	Chemical Analysis (% TiO_2)				Recovery %
	Feed	Rougher	Cleaner	Recleaner	
24	22.6	38.2	43.2	44.9	90.8
25	21.9	40.3	43.0	45.1	89.1
26	20.8	36.9	42.6	45.3	91.9

Due to materials handling problems, the control of operating variables was poor. The results indicate that a recovery of 90% or better at a grade of 45% TiO_2 or better can be expected in three stages of flotation.

Selection of circulating loads: The use of circulating loads as a means of upgrading in flotation involves extensive use of auxiliary equipment such as pumps. If smaller circulating loads are used, further stages of cleaning are required. The relative cost and/or convenience of either method is the deciding factor in a choice. The practice in this study was to use two stages of cleaning with large circulating loads. This gave greater selectivity in the cleaning stages. The practice in the Tahawus mill is to use low circulating loads and add a third stage of cleaning. This third stage was necessary due to the lower selectivity achieved.

Figure 22 shows the change in upgrading with change in circulating load with constant original feed rate. This graph has as one of its ordinates the pulp density of the tailing, which is a function of the volumetric flow. Pulp density was used because it is much easier to monitor by instruments. It is therefore an im-

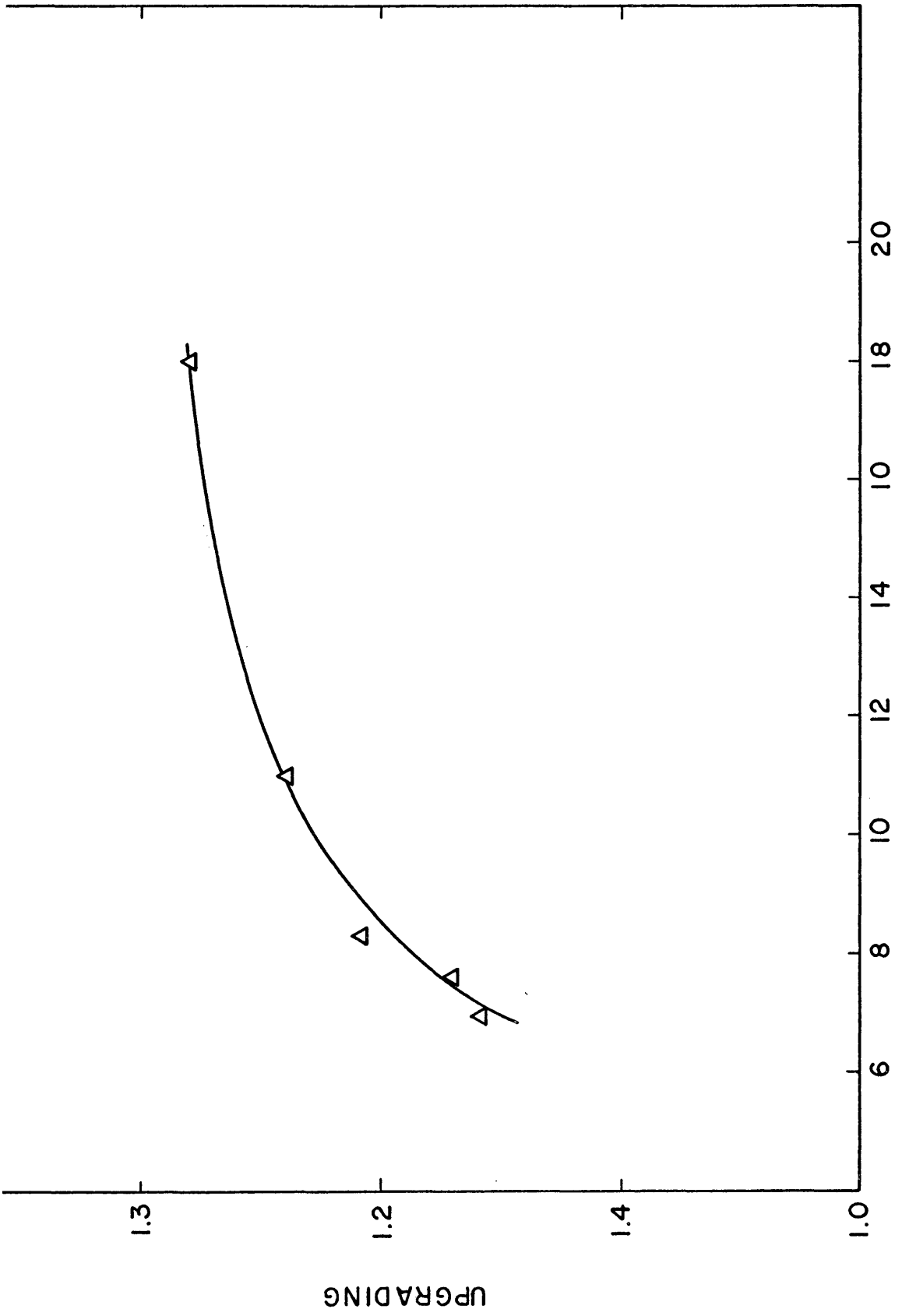

Figure 22. Pulp density of cleaner tailing versus upgrading

portant control variable for a flotation operator. Figure 22 shows there is a large increase in selectivity up to 10% solids but only a marginal increase beyond this. The results were obtained from the cleaner but a similar trend in the results could be expected from subsequent stages.

Selection of an optimum circuit: Based upon data obtained in this study, a circuit as shown in Figure 23 is suggested. A partial metallurgical balance is given in Table 11.

Table 11. Partial Metallurgical Balance on Proposed Flotation Circuit

Stream No.	Stream Flows(1) Solids	Water	Pulp	Pulp Density (% solids)	Expected Chemical Analysis (% TiO_2)
1	100	43	143	70	20.0
2	--	49.2	49.2	--	--
3	133.9	248.7	382.6	35	19.0
4	75.7	61.9	137.6	55	38.0
5	--	83.7	83.7	--	--
6	85.0	198.3	283.3	30	38.5
7	33.9	156.5	190.4	10	18.0
8	51.1	41.8	92.9	55	42.0
9	--	111.5	111.5	--	--
10	51.1	153.3	204.4	25	42.0
11	9.3	52.7	62.0	15	40.0
12	41.8	34.2	76.0	55	45.0
13	58.2	135.8	194.0	30	2.0

1/ Stream flows calculated on basis of 100 units by weight of dry solids in the feed.

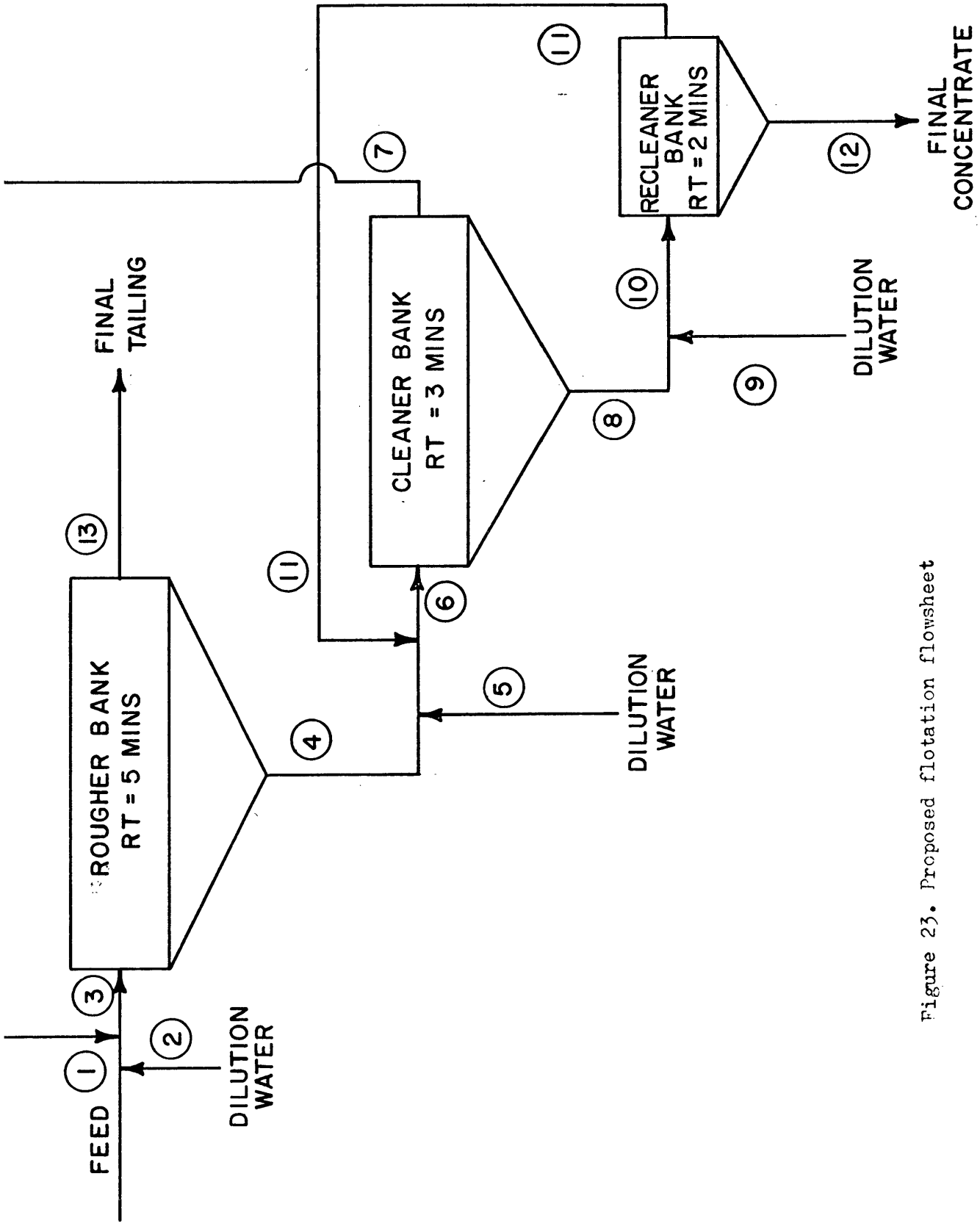

Figure 23. Proposed flotation flowsheet

Flotation Kinetics

The equation for cell recovery using a first order kinetic model is given by

$$R = \frac{K\lambda}{1 + K\lambda} \tag{21}$$

where R = fractional recovery

K = flotation rate constant

and λ = nominal cell residence time

By rearranging, Equation (21) becomes

$$K = \frac{R}{\lambda - R\lambda} \tag{22}$$

A cell by cell analysis of this bank was made while the cells were operating under similar conditions. These are included as Flotation Test Nos. 16 to 18 in Section II of the Appendix. A computer program was written to calculate cell conditions from the wet and dry weights of cell concentrates and the tailing. From the tailing volume from each cell the retention time ($=\lambda$) in each cell was calculated. From the TiO_2 assays on each cell concentrate and the tailing, the partial cell recoveries were calculated. From these the flotation rate

constant (=K) for TiO_2 was calculated for each cell. The results are shown in Tables 12 to 14.

Table 12. Flotation rate constants for TiO_2 in Flotation Test No. 18

Cell No.	Concentrate Volume (l/min)	Tailing Volume (l/min)	K ($secs^{-1} \times 10^{-5}$)
1	5.75	37.8	1486
2	1.98	35.8	1200
3	0.60	35.2	546
4	0.17	35.0	173
5	0.06	34.9	58
6	0.05	34.9	51

Table 13. Flotation rate constants for TiO_2 in Flotation Test No. 17

Cell No.	Concentrate Volume (l/min)	Tailing Volume (l/min)	K ($secs^{-1} \times 10^{-5}$)
1	9.97	33.9	3398
2	2.44	31.4	2590
3	0.13	31.3	249
4	0.11	31.2	202
5	0.06	31.1	29
6	0.00	31.1	--

Table 14. Flotation rate constants for TiO_2 in Flotation Test No. 16

Cell No.	Concentrate Volume (l/ min)	Tailing Volume (l/min)	K ($secs^{-1} \times 10^{-5}$)
1	20.59	28.7	4533
2	2.22	26.5	2357
3	0.21	26.3	283
4	0.02	26.3	31
5	0.09	26.2	119
6	0.00	26.2	

These results show that "flotation rate constant", as defined, is not a constant. It changes value from cell to cell in the same test, and it changes from test to test for the same cell. While there is a progressive decrease in its value from Cell 1 to Cell 6 the change does not appear to be regular. There is no constant proportionality factor from one cell's value to the next.

The products from Test No. 18 were screened and flotation rate constants for TiO_2 for each size fraction in each cell were calculated. These results are included in Section III of the Appendix. These data show that the flotation rate constant differs for each size fraction in each cell and it differs again from cell to cell. While there is some pattern in the change there is no systematic variation.

Roos (1969) proposes that the product $K\lambda$ is a constant. This product was calculated for each of the samples used in calculation of the flotation rate constant. The results of these calculations are also included in Section III of the Appendix. This proposed constant also changed without any apparent systematic variation.

It was thought that part of this random variation in the flotation rate constants may have been caused by the manner in which TiO_2 was

distributed in the samples. While Ti occurs mostly in ilmenite, it can also substitute in the lattices of garnets, magnetite and horneblendes. A flotation rate for TiO_2 does not necessarily measure a flotation rate for ilmenite. Hence the ilmenite concentration in these samples was determined by x-ray analysis. From these ilmenite concentrations, the flotation rate constants for ilmenite were determined. These results are shown in Table 15.

Table 15. Comparison of flotation rate constants for TiO_2 and ilmenite

Cell	Flotation Rate Constants					
	Test 16		Test 17		Test 18	
No.	Ilmenite	TiO_2	Ilmenite	TiO_2	Ilmenite	TiO_2
1	4030	4533	3671	3398	1832	1486
2	2487	2357	2031	2590	1089	1200
3	381	283	307	249	550	546
4	21	31	151	202	189	173
5	150	119	57	29	80	58
6	--	--	--	--	31	51

These results show that there was no constancy in the results for ilmenite, nor was there any regular systematic change. The peak height method for determination of mineralogical concentration is not a very accurate one. Since the flotation rate constants for TiO_2 and ilmenite were similar, it was concluded that TiO_2 assays did bear a

relation to ilmenite assay. Vogel (1952) says that a pyrosulphate fusion does not dissolve "glassy" silicates such as garnets. Most of the Ti not in the ilmenite occurs in the garnet. In view of the lack of complete solution, the Ti reported in the TiO_2 analyses is mostly from ilmenite.

Significance of results: The aim of this work was to determine whether a first order kinetic equation could be used to explain results from the flotation of a real ore. The validity of this equation had been demonstrated by Bull (1966) for a single cell and by Roos (1969) for flotation of pure quartz in a bank. In both these studies near perfect mixing occurred in the flotation vessel. Impulse tracer tests during this study showed imperfect mixing occurred at the flow rates used.

In Figure 7 the flow regimes occurring in a bank are represented. Within the active volume of the cell perfect mixing occurs. Within this region flotation in the manner described by a first order kinetic equation could be assumed to occur. If this is done then a correction factor to account for the deviant flow is required. A correction factor ($=\alpha$) is introduced in Figure 24 to account for material by-passing the stirring action. Like the mixing efficiency parameter used in this study, this correction factor is a simplified one and

does not take into account all the deviant flows shown in Figure 7.

While a mathematical model based on the approach used in Figure 24 would explain the results obtained, it would not be particularly useful as a means of predicting performance under a changed set of conditions. The selection function ($= \alpha$) would bear some relationship to the mixing efficiency parameter ($= \epsilon$). It was shown in the impulse tracer tests that ϵ varied from cell to cell and from test to test. The variation did not follow any regular or predictable trend. Hence ϵ would have to be determined for each cell at each set of conditions if this mathematical model were to be used as a means of mill control. Considerable effort and expense would be needed to measure all the variables required to calculate α. This may not be justified.

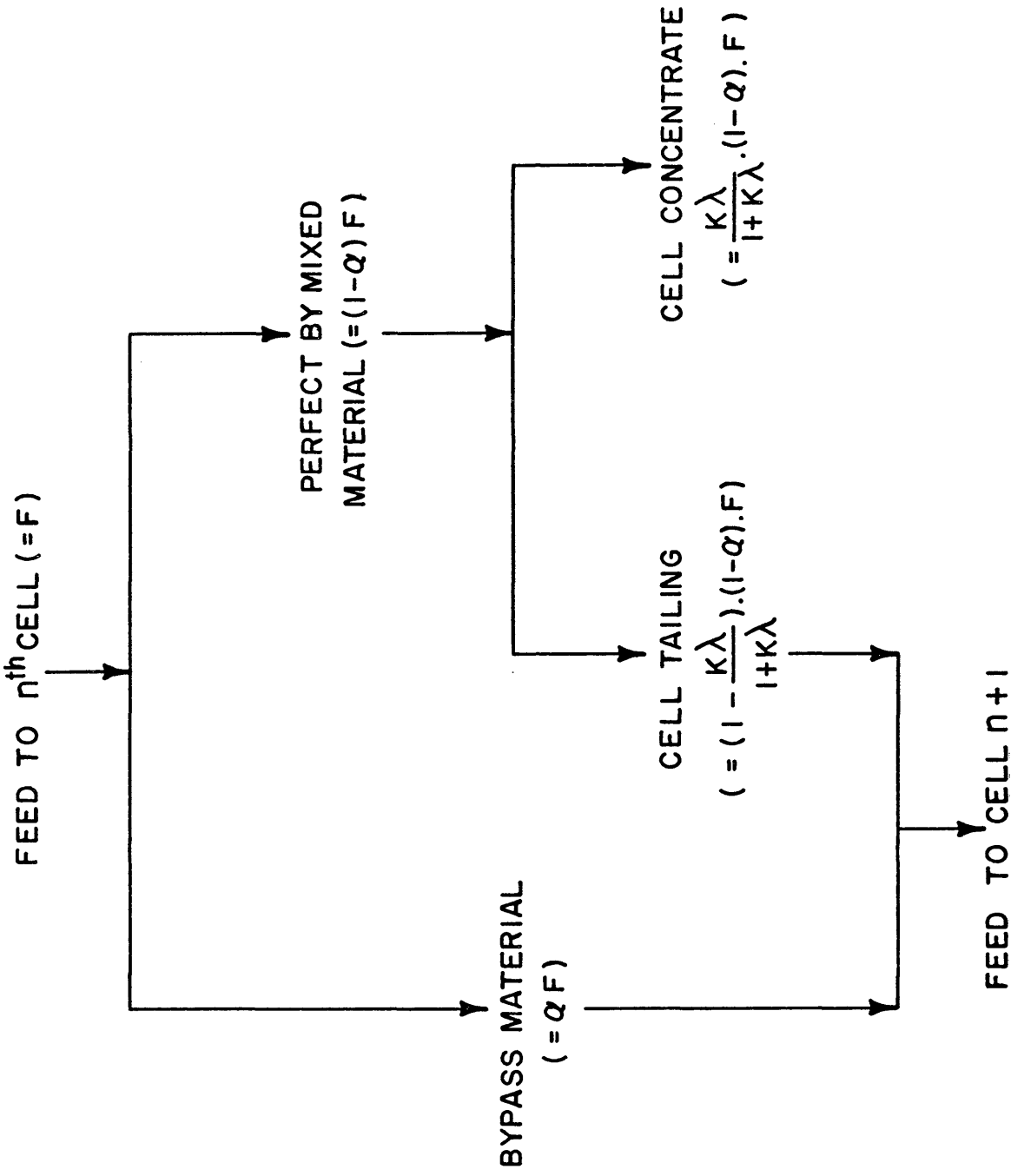

Figure 24. Proposed model of flotation process

CONCLUSIONS

The two main aims to this work were:

(a) to determine what, if any, improvement should be made to the flotation circuit in the Tahawus mill, and

(b) to determine the applicability of a first order rate equation to the flotation of ilmenite.

It was found that concentrate grades of 45% TiO_2 or better with a recovery of 90% or better could be achieved in a three stage circuit. On the basis of the results obtained, a flotation circuit was proposed. If this circuit were adopted in the Tahawus mill, the third stage of cleaning currently being used could be removed. These cells could then be used to expand current mill capacity without a change in grade or recovery.

The removal of this third stage of cleaning was made possible by the use of much larger circulating loads. It was found that this gave much greater selectivity in the cleaning stages.

Impulse tracer tests done on both the commercial and laboratory banks of cells showed a certain amount of non-ideal mixing occurring at the flow rates studied. No way was found to isolate and quantitatively measure all the deviant flows. A "mixing efficiency" parameter ($=\epsilon$) was used to quantitatively measure the departure of the mixing observed from ideality. It was shown that this deviance from ideality followed a very obscure pattern. There was no systematic variation in the parameter and it was impossible to predict, quantitatively, what the mixing efficiency under changed cell conditions would be.

It was found that a first order kinetic equation could not be used to interpret the flotation results obtained. A modification to the kinetic equation to account for this deviance was proposed. This modification introduced a selection function ($= \alpha$) which was a measure of the material by-passing the impeller. The material which is stirred in the active region of the cell was assumed to float in accordance with a first order rate equation.

A mathematical model of any process must satisfy two conditions to be of any practical value. They are:

(a) It must be able to interpret past results.

(b) It must be able to predict future results under changed operating conditions.

The mathematical model proposed would satisfy the first criterion. No simple method was found to modify it to fit the second criterion.

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APPENDIX

Section I. Impulse Tracer Test Data
Section II. Flotation Test Data
Section III. Flotation Kinetics Data

Section I. Impulse Tracer Test Data

Section I

Tracer Test No. 1

Purpose: To determine the mixing characteristics of the pilot plant rougher bank under the following conditions.

Flow Rate,	86 liters/minute
Nominal Retention Time,	0.5 minute
Impeller Speed,	1050 rpm.

Procedure: See "Analytical and Experimental Methods--Impulse Tracer Tests."

Results:

Time		Concentration (ppm NaCl)					
min	dimensionless	Cell No. 1	Cell No. 2	Cell No. 3	Cell No. 4	Cell No. 5	Cell No. 6
0.1	0.2	669	290	13	11	0	0
0.2	0.4	454	322	151	75	9	4
0.4	0.8	281	281	211	141	53	21
0.8	1.6	162	196	202	170	119	89
1.0	2.0	126	164	177	173	143	113
1.5	3.0	75	100	134	143	149	149
2.0	4.0	45	70	98	113	155	137
3.0	6.0	21	36	53	68	89	110
5.0	10.0	6	6	19	23	25	32
10.0	20.0	0	0	4	2	2	4

Section I

Tracer Test No. 2

Purpose: To determine the mixing characteristics of the pilot plant rougher bank under the following conditions.

Flow Rate,	43.3 liters/minute
Nominal Retention Time,	1 minute
Impeller Speed,	1050 rpm

Procedure: See "Analytical and Experimental Procedures--Impulse Tracer Tests."

Results:

Time		Concentration (ppm NaCl)					
min	dimensionless	Cell No. 1	Cell No. 2	Cell No. 3	Cell No. 4	Cell No. 5	Cell No. 6
0.1	0.1	858	376	26	4	2	0
0.2	0.2	639	443	138	54	9	0
0.5	0.5	410	389	227	171	71	32
1.0	1.0	222	264	246	218	156	115
1.5	1.5	151	188	210	210	186	179
2.0	2.0	102	138	171	184	194	184
3.0	3.0	60	84	117	130	153	160
5.0	5.0	28	39	54	63	78	89
7.5	7.5	9	19	19	24	28	39
10.0	10.0	4	4	9	11	13	24

Section I

Tracer Test No. 3

Purpose: To determine the mixing characteristics of the pilot plant rougher bank under the following conditions.

Flow Rate,	39.0 liters/minute
Nominal Retention Time,	1 minute
Impeller Speed,	1050 rpm
Air Condition,	On

Procedure: See "Analytical and Experimental Procedures--Impulse Tracer Tests."

Results:

Time		Concentration (ppm NaCl)					
min	dimensionless	Cell No. 1	Cell No. 2	Cell No. 3	Cell No. 4	Cell No. 5	Cell No. 6
0.1	0.1	838	346	76	18	0	0
0.2	0.2	650	443	170	37	4	0
0.5	0.5	419	393	247	126	48	17
1.0	1.0	283	294	264	296	117	89
1.5	1.5	197	225	222	227	158	145
2.0	2.0	153	179	188	190	180	168
3.0	3.0	99	125	143	162	175	166
5.0	5.0	58	76	91	110	121	130
7.5	7.5	32	43	52	65	71	84
10.0	10.0	19	26	30	39	41	43

Section I

Tracer Test No. 4

Purpose: To determine the mixing characteristics of the pilot plant rougher bank under the following conditions.

Flow Rate,	21.7 liters/minute
Nominal Retention Time,	2 minutes
Impeller Speed,	550 rpm

Procedure: See "Analytical and Experimental Methods--Impulse Tracer Tests."

Results:

Time		Concentration (ppm NaCl)					
min	dimensionless	Cell No. 1	Cell No. 2	Cell No. 3	Cell No. 4	Cell No. 5	Cell No. 6
0.1	0.05	860	219	12	0	0	0
0.2	0.1	867	301	57	0	0	0
0.5	0.25	618	402	132	20	4	0
1.0	0.5	426	362	203	80	41	18
1.5	0.75	292	327	265	129	69	49
2.0	1.0	261	281	228	163	130	86
3.0	1.5	193	215	211	181	174	144
5.0	2.5	133	140	166	173	176	176
10.0	5.0	59	70	90	110	115	121
20.0	10.0	18	23	31	39	41	41

Section I

Tracer Test No. 5

Purpose: To determine pilot plant mixing characteristics under the following conditions.

Flow Rate,	43.3 liters/minute
Nominal Retention Time,	1 minute
Impeller Speed,	550 rpm

Procedure: See "Analytical and Experimental Procedures -- Impulse Tracer Tests."

Results:

Time		Concentration (ppm NaCl)					
min	dimensionless	Cell No. 1	Cell No. 2	Cell No. 3	Cell No. 4	Cell No. 5	Cell No. 6
0.1	0.1	931	240	26	0	0	0
0.2	0.2	780	333	67	2	0	0
0.5	0.5	462	378	190	24	2	0
1.0	1.0	298	339	237	108	18	33
1.5	1.5	203	253	245	172	133	90
2.0	2.0	168	205	235	190	168	125
3.0	3.0	84	121	163	154	181	170
5.0	5.0	39	55	89	117	127	138
7.5	7.5	13	27	43	60	65	76
10.0	10.0	6	13	22	28	30	37

Section I

Tracer Test No. 6

Purpose: To determine the mixing characteristics of the pilot plant rougher bank under the following conditions.

Flow Rate,	87 liters/minute
Nominal Retention Time,	0.5 minute
Impeller Speed,	550 rpm

Procedure: See "Analytical and Experimental Methods--Impulse Tracer Tests."

Results:

Time		Concentration (ppm NaCl)					
min	dimensionless	Cell No. 1	Cell No. 2	Cell No. 3	Cell No. 4	Cell No. 5	Cell No. 6
0.1	0.2	950	415	9	4	0	0
0.2	0.4	616	436	69	9	0	0
0.3	0.6	588	445	192	30	6	6
0.5	1.0	147	417	289	108	32	11
1.0	2.0	143	255	287	227	167	110
1.5	3.0	71	136	203	222	219	188
2.0	4.0	39	76	136	187	210	212
2.5	5.0	24	45	94	140	160	186
3.0	6.0	11	32	57	100	119	145
5.0	10.0	4	4	13	22	31	35

Section I

Tracer Test No. 7

Purpose: To determine the mixing characteristics of the 48 cu ft cells in the "North Rougher Bank" of the Tahawus mill under the following conditions.

Flow Rate,	32.2 cu ft/minute
Nominal Retention Time,	1.50 minutes

Procedure: See "Analytical and Experimental Procedures -- Impulse Tracer Tests and Determination of Bulk Flow Rates by Salt Method."

Results:

Time		Concentration (ppm NaCl)							
min	dimensionless	Cell No. 1	Cell No. 2	Cell No. 3	Cell No. 4	Cell No. 5	Cell No. 6	Cell No. 7	Cell No. 8
0.2	0.13	1622	264	2	0	0	0	2	0
0.5	0.33	871	589	129	91	11	4	4	0
1.0	0.67	510	528	317	215	13	4	4	2
1.5	1.0	314	409	357	300	57	25	11	4
2.0	1.33	222	317	355	336	122	65	21	11
3.0	2.0	112	184	283	306	226	156	63	49
4.0	2.67	68	120	213	234	226	213	137	108
5.0	3.33	42	73	137	160	228	222	188	165
7.5	5.0	19	30	51	61	125	146	182	196
10.0	6.67	6	15	17	25	61	72	118	127

Section I

Tracer Test No. 8

Purpose: To determine the mixing characteristics of the 100 cu ft cells in the Tahawus mill under the following conditions.

Flow Rate,	62.2 cu ft/minute
Nominal Retention Time,	1.61 minutes

Procedure: See "Analytical and Experimental Procedures -- Impulse Tracer Tests and Determination of Bulk Flow Rates by Salt Method."

Results:

Time		Concentration (ppm NaCl)					
min	dimensionless	Cell No. 1	Cell No. 2	Cell No. 3	Cell No. 4	Cell No. 5	Cell No. 6
0.2	0.12	450	232	0	0	0	0
0.5	0.31	445	338	6	6	6	0
1.0	0.62	323	348	87	49	17	4
1.5	0.93	241	279	137	95	49	23
2.0	1.24	194	211	152	125	74	59
3.0	1.86	125	131	152	148	125	106
4.0	2.48	82	97	133	146	146	131
5.0	3.11	55	74	101	141	141	148
7.5	4.66	2	32	80	104	112	118
10.0	6.21	0	13	36	74	82	91

Section II. Flotation Test Data

Section II

Screen Assay Test No. 1

Sample: Concentrate from Cell No. 1 in Flotation Test No. 3.

Procedure: The complete cell concentrate was collected for a timed period. This was then filtered, oven dried, and weighed. Approximately 500 g was split out using a Jones splitter and dry screened on the screens shown below.

Results:

Screen Product (Tyler) Mesh		Weight g	Weight % Direct	Chemical Analysis TiO_2 %
Heads (assayed)				39.9
Heads (computed)		548.1	100.0	39.4
	+48	--	--	--
-48	+65	3.8	0.7	28.6
-65	+100	17.3	3.2	36.9
-100	+150	56.1	10.2	41.9
-150	+200	94.9	17.3	42.9
-200	+270	66.0	12.0	42.1
-270	+325	62.5	11.4	41.1
-325		247.5	45.2	36.7

Section II

Screen Assay Test No. 2

Sample: Concentrate from Cell No. 2 in Flotation Test No. 3.

Procedure: As for Screen Assay Test No. 1.

Results:

Screen Product (Tyler) Mesh		Weight g	Weight % Direct	Chemical Analysis TiO_2 %
Heads (assayed)				37.1
Heads (computed)		538.1	100.0	37.1
	+48	--	--	--
-48	+65	4.1	0.8	24.2
-65	+100	21.1	3.9	33.9
-100	+150	57.9	10.8	38.7
-150	+200	104.4	19.4	40.2
-200	+270	61.0	11.3	39.6
-270	+325	44.1	8.2	38.5
-325		245.5	45.6	35.0

Section II

Screen Assay Test No. 3

Sample: Concentrate from Cell No. 3 in Flotation Test No. 3.

Procedure: As for Screen Assay Test No. 1.

Results:

Screen Product (Tyler) Mesh		Weight g	Weight % Direct	Chemical Analysis TiO_2 %
Heads (assayed)				35.2
Heads (computed)		499.3	100.0	35.2
	+48	--	--	--
-48	+65	4.0	0.8	28.9
-65	+100	16.8	3.4	34.4
-100	+150	51.0	10.2	38.4
-150	+200	81.1	10.2	39.0
-200	+270	58.1	11.7	37.6
-270	+325	57.1	11.4	36.3
-325		231.2	46.3	32.4

Section II

Screen Assay Test No. 4

Sample: Concentrate from Cell No. 4 in Flotation Test No. 3.

Procedure: As for Screen Assay Test No. 1.

Results:

Screen Product (Tyler) Mesh		Weight g	Weight % Direct	Chemical Analysis TiO_2 %
Heads (assayed)				26.8
Heads (computed)		513.9	100.0	27.7
	+48	--	--	--
-48	+65	3.6	0.7	15.1
-65	+100	14.1	2.7	24.4
-100	+150	36.1	7.0	29.9
-150	+200	76.0	14.8	31.1
-200	+270	50.4	9.8	29.4
-270	+325	39.0	7.6	27.7
-325		294.7	57.4	24.8

Section II

Screen Assay Test No. 5

Sample: Concentrate from Cell No. 5 in Flotation Test No. 3.

Procedure: As for Screen Assay Test No. 1.

Results:

Screen Product (Tyler) Mesh		Weight g	Weight % Direct	Chemical Analysis TiO_2 %
Heads (assayed)				22.8
Heads (computed)		512.8	100.0	22.3
	+48	--	--	--
-48	+65	2.9	0.6	17.3
-65	+100	10.0	2.0	24.4
-100	+150	28.2	5.5	28.7
-150	+200	53.9	10.5	27.7
-200	+270	46.5	9.1	25.1
-270	+325	58.0	11.3	22.8
-325		313.3	61.0	20.5

Section II

Screen Assay Test No. 6

Sample: Concentrate from Cell No. 6 in Flotation Test No. 3.

Procedure: As for Screen Assay Test No. 1.

Results:

Screen Product (Tyler) Mesh	Weight g	Weight % Direct	Chemical Analysis TiO_2 %
Heads (assayed)			20.3
Heads (computed)	493.1	100.0	20.1
+48	--	--	--
-48 +65	4.0	0.8	7.0
-65 +100	11.0	2.2	12.9
-100 +150	21.1	4.3	19.3
-150 +200	43.3	8.8	22.2
-200 +270	35.8	7.3	21.9
-270 +325	30.9	6.3	20.1
-325	347.0	70.3	19.9

Section II

Screen Assay Test No. 7

Sample: Tailing from Flotation Test No. 3.

Procedure: As for Screen Assay Test No. 1.

Results:

Screen Product (Tyler) Mesh		Weight g	Weight % Direct	Chemical Analysis TiO_2 %
Heads (assayed)				2.7
Heads (computed)		582.2	100.0	2.8
	+48	8.1	1.4	1.3
-48	+65	40.6	7.0	1.7
-65	+100	88.9	15.3	2.5
-100	+150	106.4	18.3	3.1
-150	+200	94.6	16.2	3.1
-200	+270	52.3	9.0	2.6
-270	+325	50.8	8.7	2.4
-325		141.5	24.1	3.1

Section II

Flotation Test No. 1

Purpose: All flotation tests had the common purpose of floating ilmenite. Operating variables were changed from test to test in order to find the relative effects of each of these variables. The results of these tests were used to predict an optimum circuit.

Sample: The feed to all flotation tests was drawn from the mill's final conditioning tank as described in "Analytical and Experimental Procedures."

Procedure: Flotation Tests 1 to 18 were conducted on the rougher bank in open circuit. Flotation Tests 19 to 23 were conducted on the cleaner bank in open circuit, and Tests 24 to 26 on the recleaner bank in closed circuit.

In all tests time was allowed for equilibrium to be established and then timed samples were taken. These were wet weighed, filtered, and oven dried to constant weight. Assay samples were split out from these samples using the Jones splitter.

In Test 2 the following feed conditions were used:

Pulp Density,	56.7%
Volumetric Flow,	32.7 liters/minute

In Tests 1 to 15 air flow rates were as follows:

Cells 1 to 3,	2 cu ft/minute/cell
Cells 4 to 6,	6 cu ft/minute/cell

Results: See page A 20.

Section II

Flotation Test No. 1 (continued)

Results:

Product	Dry Weight g/min	Weight %	Chemical Analysis TiO_2 %
Heads (assayed)			22.3
Heads (computed)	31, 356	100.0	22.3
Cell No. 1	5, 173	16.5	40.5
Cell No. 2	4, 464	14.3	39.0
Cell No. 3	4, 600	14.7	35.8
Cell No. 4	2, 772	8.8	30.6
Cell No. 5	1, 013	3.2	27.3
Cell No. 6	421	1.3	23.3
Tailing	12, 913	41.2	2.3

Section II

Flotation Test No. 2

Purpose: As for Test 1.

Sample: As for Test 1.

Procedure: The following feed conditions were used.

Pulp Density,	34.2%
Volumetric Flow,	43.0 liters/minute

Results:

Product	Dry Weight g/min	Weight %	Chemical Analysis TiO_2 %
Heads (assayed)			20.1
Heads (computed)	19,498	100.0	20.0
Cell No. 1	4,009	20.6	41.8
Cell No. 2	2,743	14.1	41.2
Cell No. 3	674	3.4	41.7
Cell No. 4	456	2.3	34.4
Cell No. 5	386	2.0	32.4
Cell No. 6	110	0.6	31.4
Tailing	11,120	57.0	4.3

Section II

Flotation Test No. 3

Purpose: As explained in Test 1.

Sample: As for Test 1.

Procedure: The following feed conditions were used:

Pulp Density,	42.0%
Volumetric Flow,	39.4 liters/minute

Results:

Product	Dry Weight g/min	Weight %	Chemical Analysis TiO_2 %
Heads (assayed)			20.2
Heads (computed)	23,950	100.0	20.0
Cell No. 1	4,660	19.5	39.9
Cell No. 2	4,580	19.1	37.1
Cell No. 3	1,730	7.2	35.2
Cell No. 4	670	2.8	26.8
Cell No. 5	390	1.6	22.8
Cell No. 6	170	0.7	20.3
Tailing	11,750	49.1	2.7

Section II

Flotation Test No. 4

Purpose: As for Test 1..

Sample: As for Test 1.

Procedure: The following feed conditions were used:

Pulp Density,	33.1%
Volumetric Flow,	39.2 liters/minute

Results:

Product	Dry Weight g/min	Weight %	Chemical Analysis TiO_2 %
Heads (assayed)			25.0
Heads (computed)	17,037	100.0	20.9
Cell No. 1	5,303	31.1	38.4
Cell No. 2	2,509	14.7	36.0
Cell No. 3	995	5.9	33.3
Cell No. 4	915	5.4	26.2
Cell No. 5	347	2.0	19.9
Cell No. 6	190	1.1	15.6
Tailing	6,778	39.8	1.8

Section II

Flotation Test No. 5

Purpose: As for Test 1.

Sample: As for Test 1.

Procedure: The following feed conditions were used:

Pulp Density,	28.4%
Volumetric Flow,	39.4 liters/minute

Results:

Product	Dry Weight g/min	Weight %	Chemical Analysis TiO_2 %
Heads (assayed)			21.9
Heads (computed)	14,070	100.0	21.3
Cell No. 1	2,620	18.6	41.9
Cell No. 2	2,030	14.4	40.6
Cell No. 3	770	5.5	39.9
Cell No. 4	750	5.3	37.5
Cell No. 5	380	2.7	35.8
Cell No. 6	70	0.5	35.8
Tailing	7,450	53.0	4.3

Section II

Flotation Test No. 6

Purpose: As for Test 1.

Sample: As for Test 1.

Procedure: The following feed conditions were used:

Pulp Density,	24.6%
Volumetric Flow,	38.5 liters/minute

Results:

Product	Dry Weight g/min	Weight %	Chemical Analysis TiO_2 %
Heads (assayed)			21.3
Heads (computed)	11,540	100.0	18.5
Cell No. 1	3,671	31.8	39.5
Cell No. 2	1,071	9.3	38.9
Cell No. 3	85	0.7	38.5
Cell No. 4	123	1.0	30.5
Cell No. 5	131	1.1	24.6
Cell No. 6	55	0.5	20.5
Tailing	5,604	48.6	2.7

Section II

Flotation Test No. 7

Purpose: As for Test 1.

Sample: As for Test 1.

Procedure: For Tests 7 to 15 bulk samples for the rougher bank were collected--not from each individual cell. The following feed conditions were used in Test 7:

Pulp Density,	22%
Volumetric Flow,	58.3 liters/minute

Results:

Product	Dry Weight g/min	Weight %	Chemical Analysis TiO_2 %
Heads (assayed)			18.4
Heads (computed)	15, 474	100.0	19.3
Concentrate	5, 326	34.4	41.5
Tailing	10, 148	65.6	7.6

Section II

Flotation Test No. 8

Purpose: As for Test 1.

Sample: As for Test 1.

Procedure: The following feed conditions were used:

Pulp Density,	29%
Volumetric Flow,	38.9 liters/minute

Results:

Product	Dry Weight g/min	Weight %	Chemical Analysis TiO_2 %
Heads (assayed)			14.5
Heads (computed)	14, 334	100.0	18.0
Concentrate	6, 390	44.6	37.4
Tailing	7, 944	55.4	2.3

Section II

Flotation Test No. 9

Purpose: As for Test 1.

Sample: As for Test 1.

Procedure: The following feed conditions were used:

Pulp Density,	35%
Volumetric Flow,	26.8 liters/minute

Results:

Product	Dry Weight g/min	Weight %	Chemical Analysis TiO_2 %
Heads (assayed)			18.8
Heads (computed)	12,606	100.0	23.2
Concentrate	8,552	67.8	33.2
Tailing	4,054	32.2	2.1

Section II

Flotation Test No. 10

Purpose: As for Test 1.

Sample: As for Test 1.

Procedure: The following feed conditions were used:

Pulp Density, 23%
Volumetric Flow, 71.2 liters/minute

Results:

Product	Dry Weight g/min	Weight %	Chemical Analysis TiO_2 %
Heads (assayed)			19.1
Heads (computed)	19,731	100.0	19.5
Concentrate	9,276	47.0	38.2
Tailing	10,455	53.0	2.9

Section II

Flotation Test No. 11

Purpose: As for Test 1.

Sample: As for Test 1.

Procedure: The following feed conditions were used:

Pulp Density,	31%
Volumetric Flow,	47.4 liters/minute

Results:

Product	Dry Weight g/min	Weight %	Chemical Analysis TiO_2 %
Heads (assayed)			19.7
Heads (computed)	19,104	100.0	20.3
Concentrate	10,232	53.6	36.0
Tailing	8,872	46.4	2.2

Section II

Flotation Test No. 12

Purpose: As for Test 1.

Sample: As for Test 1.

Procedure: The following feed conditions were used:

Pulp Density,	37%
Volumetric Flow,	35.4 liters/minute

Results:

Product	Dry Weight g/min	Weight %	Chemical Analysis TiO_2 %
Heads (assayed)			19.7
Heads (computed)	17,728	100.0	22.9
Concentrate	10,844	61.2	35.9
Tailing	6,884	38.8	2.0

Section II

Flotation Test No. 13

Purpose: As for Test 1.

Sample: As for Test 1.

Procedure: The following feed conditions were used:

Pulp Density,	25%
Volumetric Flow,	80.4 liters/minute

Results:

Product	Dry Weight g/min	Weight %	Chemical Analysis TiO_2 %
Heads (assayed)			20.4
Heads (computed)	23,691	100.0	20.1
Concentrate	11,776	49.7	37.2
Tailing	11,915	50.3	3.2

Section II

Flotation Test No. 14

Purpose: As for Test 1.

Sample: As for Test 1.

Procedure: The following feed conditions were used:

Pulp Density,	34%
Volumetric Flow,	53.0 liters/minute

Results:

Product	Dry Weight g/min	Weight %	Chemical Analysis TiO_2 %
Heads (assayed)			20.0
Heads (computed)	23,556	100.0	20.0
Concentrate	13,452	57.1	33.5
Tailing	10,104	42.9	2.1

Section II

Flotation Test No. 15

Purpose: As for Test 1.

Sample: As for Test 1.

Procedure: The following feed conditions were used:

Pulp Density,	41%
Volumetric Flow,	36.4 liters/minute

Results:

Product	Dry Weight g/min	Weight %	Chemical Analysis TiO_2 %
Heads (assayed)			19.6
Heads (computed)	21, 392	100.0	22.3
Concentrate	13, 336	62.3	34.5
Tailing	8, 056	37.7	2.0

Section II

Flotation Test No. 16

Purpose: Tests 16 to 19 were designed to test the effect of changing air flow rates to each cell. Conditions from cell to cell were held constant so that data from these tests could be used in subsequent kinetic studies.

Sample: As for Test 1.

Procedure: The following feed conditions were used:

Pulp Density,	31.9%
Volumetric Flow,	49.3 liters/minute
Air Flow,	6 cu ft/minute/cell

Results:

Product	Wet Weight g/min	Dry Weight g/min	Weight %(1)	Chemical Analysis TiO_2 %
Heads (assayed)				22.5
Heads (computed)	63,940	20,416	100.0	22.2
Cell No. 1	28,489	10,435	51.1	34.3
Cell No. 2	3,654	1,888	9.2	34.2
Cell No. 3	343	170	0.8	36.7
Cell No. 4	41	18	0.1	37.6
Cell No. 5	141	65	0.3	35.9
Cell No. 6	--	--	--	
Tailing	31,272	7,840	38.4	2.7

1/ Based on dry weights.

Section II

Flotation Test No. 17

Purpose: As for Test 16.

Sample: As for Test 1.

Procedure: The following feed conditions were used:

Pulp Density,	33.3%
Volumetric Flow,	43.9 liters/minute
Air Flow,	4 cu ft/minute/cell

Results:

Product	Wet Weight g/min	Dry Weight g/min	Weight % (1)	Chemical Analysis TiO_2 %
Heads (assayed)				23.4
Heads (computed)	58,113	19,728	100.0	22.9
Cell No. 1	16,014	7,885	40.0	38.7
Cell No. 2	4,700	2,796	14.2	36.8
Cell No. 3	272	177	0.9	38.9
Cell No. 4	216	126	0.6	38.4
Cell No. 5	80	18	0.1	37.1
Cell No. 6	--	--	--	
Tailing	36,831	8,726	44.2	

1/ Based on dry weights.

Section II

Flotation Test No. 18

Purpose: As for Test 16.

Sample: As for Test 1.

Procedure: The following feed conditions were used:

Pulp Density,	34.1%
Volumetric Flow,	43.5 liters/minute
Air Flow,	2 cu ft/minute/cell

Results:

Product	Wet Weight g/min	Dry Weight g/min	Weight %(1)	Chemical Analysis TiO_2 %
Heads (assayed)				21.0
Heads (computed)	57,456	19,640	100.0	20.3
Cell No. 1	9,430	4,760	24.2	40.6
Cell No. 2	3,738	2,275	11.6	40.2
Cell No. 3	1,186	753	3.8	41.0
Cell No. 4	343	212	1.1	41.7
Cell No. 5	116	69	0.4	41.7
Cell No. 6	97	57	0.3	42.4
Tailing	42,546	11,514	58.6	6.0

Section II

Flotation Test No. 19

Purpose: Flotation Tests 19 to 23 were conducted in the cleaner bank in open flow. The aim was to determine the effect that changing the circulating load would have on the upgrading achieved.

Sample: As for Test 1.

Procedure: For Tests 19 to 23 a constant feed rate was used and the volume of tailing was changed by variation of the air flow rates. For all these tests the rate was less than 2 cu ft/minute/cell which was below the limits of the air flow meters installed. For all tests the feed rate was approximately 36 liters/minute at 23% solids. For Test 19 tailing pulp density was 17.9%.

Results:

Product	Dry Weight g/min	Weight %	Chemical Analysis TiO_2 %
Heads (assayed)			34.5
Heads (computed)	9,546	100.0	36.7
Concentrate	1,716	18.0	44.9
Tailing	7,830	82.0	34.9

Section II

Flotation Test No. 20

Purpose: As for Test 19.

Sample: As for Test 1.

Procedure: Tailing pulp density was 10.6%.

Results:

Product	Dry Weight g/min	Weight %	Chemical Analysis TiO_2 %
Heads (assayed)			34.5
Heads (computed)	12, 224	100.0	36.7
Concentrate	8, 752	71.6	42.6
Tailing	3, 472	28.4	21.7

Section II

Flotation Test No. 21

Purpose: As for Test 19.

Sample: As for Test 1.

Procedure: Tailing pulp density was 8.3%.

Results:

Product	Dry Weight g/min	Weight %	Chemical Analysis TiO_2 %
Heads (assayed)			34.4
Heads (computed)	12, 248	100.0	36.4
Concentrate	9, 476	77.4	41.6
Tailing	2, 772	22.6	18.4

Section II

Flotation Test No. 22

Purpose: As for Test 19.

Sample: As for Test 1.

Procedure: Tailing pulp density was 7.6%.

Results:

Product	Dry Weight g/min	Weight %	Chemical Analysis TiO_2 %
Heads (assayed)			34.9
Heads (computed)	11,768	100.0	34.9
Concentrate	9,184	78.0	40.9
Tailing	2,584	22.0	13.5

Section II

Flotation Test No. 23

Purpose: As for Test 19.

Sample: As for Test 1.

Procedure: Tailing pulp density was 7.0%.

Results:

Product	Dry Weight g/min	Weight %	Chemical Analysis TiO_2 %
Heads (assayed)			34.4
Heads (computed)	12, 116	100.0	34.4
Concentrate	9, 764	80.6	39.9
Tailing	2, 352	19.4	11.3

Section II

Flotation Test No. 24

Purpose: Tests 24 to 26 were conducted on the 3-stage pilot plant in closed circuit. Optimum bank conditions as determined in previous tests were used to determine circuit performance using these conditions.

Sample: As for Test 1.

Procedure: Due to the materials handling problem described in the text, control of operating variables was poor. After equilibrium was judged to have been established timed samples were taken from the streams shown below.

Results:

Product	Pulp Density %	Weight %(1)	Chemical Analysis TiO_2 %
Heads (assayed)			22.6
Heads (computed)			20.0
Original Feed	67	100.0	22.6
Rougher Feed	41	105	22.0
Cleaner Feed	34	79	38.2
Recleaner Feed	37	46	43.2
Recleaner Tailing	21	32	38.1
Cleaner Tailing	14	16	17.1
Final Concentrate	43	40.5	44.9
Final Tailing	38	59.5	3.1

1/ Weights expressed as percentage of dry weight of original new feed.

Section II

Flotation Test No. 25

Purpose: As for Test 24.

Sample: As for Test 1.

Procedure: As for Test 24.

Results:

Product	Pulp Density %	Weight %(1)	Chemical Analysis TiO_2 %
Heads (assayed)			21.9
Heads (computed)			21.5
Original Feed	66	100.0	21.9
Rougher Feed	33	113	23.1
Cleaner Feed	24	67	40.3
Recleaner Feed	26	49	43.0
Recleaner Tailing	11	15	39.4
Cleaner Tailing	4	11	18.6
Final Concentrate	29	42.5	45.1
Final Tailing	24	57.5	4.1

1/ Weights expressed as percentage of dry weight of original new feed.

Section II

Flotation Test No. 26

Purpose: As for Test 24.

Sample: As for Test 1.

Procedure: As for Test 24. Due to mechanical difficulties, equilibrium was never established in the circuit. No samples for quantitative calculations were taken. Only two samples were taken for assay.

Results:

Product	Pulp Density %	Weight %(1)	Chemical Analysis TiO_2 %(2)
Heads (assayed)			
Heads (computed)			20.7
Original Feed	69	100.0	
Rougher Feed	44	154	
Cleaner Feed	37	98	
Recleaner Feed	39	92	
Recleaner Tailing	17	16	
Cleaner Tailing	21	30	
Final Concentrate	28	42.0	45.3
Final Tailing	37	58.0	2.9

1/ Weights expressed as percentage of dry weights of original new feed.

2/ Only two samples assayed.

Section III. Flotation Kinetics

Section III

```
C                Program to Compute Bank Performance
C                  Assuming Ideal Mixing Index
C
C    Data Required in Bank Test
C
C    A(I)..... Assay of TiO2 in Float from Cell (I) in weight percent
C    AFE..... Assay of Feed in Weight Percent
C    AT..... Assay of Tailing in Weight Percent
C    CV.....Cell Volume in Litres
C    N..... Number of Cells
C    SG(I)..... Specific Gravity of Solids from Cell (I)
C    SGT.....Specific Gravity of Solids in Tailings
C    T..... Length of Time for which Flotation Products From
C              the Cells are Collected (in seconds)
C    TT..... Length of Sampling Time for Tailings in Seconds
C    WTDT.... Weight of Dry Solids in Tailings Sample in Grams
C    WTD(I).... Weight of Dry Solids in Sample from Cell (I) in Grams
C    WTW(I)....Weight of Wet Solids in Sample from Cell (I) in Grams
C    WTWT.....Weight of Wet Solids in Tailings Sample in Grams
C
C
C    Other Variables
C
C    CREC(I)... Cumulative Recovery from Cell (I)
C    K(I)..... Flotation Rate of TiO2 in Cell (I) in Seconds -1
C    PROB(I)... Probability of Flotation
C    Q(I)..... Calculated Volume of Tailing Pulp from Cell (I)
C    REC(I).... Recovery of TiO2 in Cell (I) to Cell (I+1)
C              in Litres/Minute
C    RT(I)..... Retention Time in Cell (I) in Seconds
C    S(I)..... % Solids in Froth from Cell (I)
C    SFE..... % Solids in Feed
C    ST(I)..... % Solids in Tailing from Cell (I)
C    TI(I)..... Weight of TiO2 in Tailing in Grams
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Section III

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C     TOTI..... Total Weight of TiO2 in Feed in Grams
C     V(I)..... Volume of Froth from Cell (I) in Time T in Litres
C     VFE..... Volume Flow Rate of Feed in Litres/Minute
C     W(I)..... Dry Solids Flow Rate in Grams/Minute
C     WW(I).... Pulp Flow Rate in Grams/Minute
C     WWT..... Pulp Flow Rate in Tailing in Grams/Minute
C     WT..... Dry Solids Flow Rate in Tailing in Grams/Minute
C
C     DIMENSION SG(10), WTD(10), WTW(10), Q(10), RT(10), V(10),
C     1  TI(10), REC(10), A(10), CREC(10), RK(10), T(10), PROB(10),
C     2  S(10), W(10), ST(10), QT(10), WW(10)

      READ(2, 10)N
10    FORMAT(I2)
      READ(2, 20) AFE, (A(I), I=1, N), AT
      READ(2, 20) WWFE, (WTW(I), I=1, N), WTWT
      READ(2, 20) WFE, (WTD(I), I=1, N), WTDT
      READ(2, 20) SGFE, (SG(I), I=1, N), SGT
      READ(2, 20) TFE, (T(I), I=1, N), TT
      READ(2, 20) CV
20    FORMAT(10F8.3)
      DO 25 I=1, N
      W(I) = 60.  WTD(I)/T(I)
      WW(I) = 60.  WTW(I)/T(I)
25    V(I) = 60. ((WTW(I)-WTD(I))+WTD(I)/SG(I))/(1000.  T(I))
      WWT = 60.  WTWT/TT
      WT = 60.  WTDT/TT
      SFT = 100.  WT/WWT
      VT = 60.  ((WTWT-WTDT)+WTDT/SGT)/(1000.  TT)
      WFE = 0.
      VFE = 0.
      WWFE = 0.
      DO 30  I=1, N
      WWFE = WWFE + WW(I)
      WFE = WFE + W(I)
30    VFE = VFE + V(I)
      WWFE = WWFE + WWT
      WFE = WFE + WT
      VFE = VFE + VT
      Q(1) = VFE - V(1)
      DO 40  I=1, 5
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Section III

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40    Q(I+1) = Q(I) - V(I+1)
      DO 50  I=1, N
      S(I) = 100.   W(I)/WW(I)
50    RT(I) = (CV/Q(I))   60.
      SFE = 100.   WFE/WWFE
      TOTI = 0.
      DO 90  I=1, N
      TI(I) = 60.   WTD(I)  A(I)/(T(I)  100.)
90    TOTI = TOTI + TI(I)
      TOTI - TOTI + 60.  AT  WTDT/(TT  100.)
      E = WFE
      EW = WWFE
      DO  100  I=1, N
      E = E - W(I)
      EW = EW - WW(I)
      QT(I) = E
      ST(I) = 100.  E/EW
100   REC(I) = (TI(I)/TOTI)  100.
      RK(I) = REC(1)/(RT(1)  (100.  -REC(1))) Change & Until RK =
      Const.
      CREC(1) = REC(1)
      PROB(1) = RK(1)  RT(1)
      DO 110  I=2, N
      CREC(I) = REC(I) + CREC(I-1)
      RK(I) = REC(I)/(RT(I)  (100.  - CREC(I)))
110   PROB(I) = RK(I)  RT(I)
      LOSS = 100.  - CREC(N)
      WRITE (5, 120)VFE, VT, WWFE, WWT, WFE, WT, SFE, SFT, AFE,
      AT, CREC(6), LOSS
120   FORMAT(1H1,  'FEED VOLUME', F10.2,  'L/MIN', 33X,
      'TAILING VOLUME', F11.2, 'L/MIN'//' FEED PULP WT',
      F9.1, 'GRAMS/MIN', 29X, 'TAILING PULP WT', F10.1,
      'GRAMS/MIN'//'FEED SOLIDS', F10.1, 'GRAMS/MIN',
      29X, 'TAILING SOLIDS', F11.1, 'GRAMS/MIN'//'FEEDO/
      OSOLIDS', F8.1'0/0', 35X, 'TAILINGO/OSOLIDS', F8.1,
      '0/0'//'FEED ASSAY', F11.1, '0/0', 35X, 'TAILING ASSAY',
      F12.1, '0/0'//'RECOVERY', F13.1, '0/0', 35X, 'LOSS', F21.1,
      '0/0'//)
      WRITE(5, 130)
```

Section III

```
130   FORMAT('CELL', 8X, 'FROTH VOLUME', 6X, 'FROTH SOLIDS',
      1  6X, '(SOLIDS', 5X, 'CELL TAILING', 6X, 'CELL SOLIDS', 6X,
      2  '(SOLIDS'/'NUMBER', 8X, 'L/MIN', 14X, 'GRAMS', 22X,
      3  'VOLUME L/MIN', 10X, 'GRAMS'//)
      DO 140  I=1, N
140   WRITE(5, 150)I, V(I), W(I), S(I), Q(I), QT(I), ST(I)
150   FORMAT(13, 12X, F8.2, 12X, F8.2, 7XF6.1, 9X, F8.2, 10X,
      1  F8.2, 7X, F6.1)
      WRITE(5, 160)
160   FORMAT(////'CELL', 8X, 'FLOAT ASSAY', 6X, 'CELL RE-
      1  COVERY', 6X, 'CUMULATIVE RECOVERY', 6X, 'RETN
      2  TIME', 6X, 'RATE CONSTANT', 5X, 'PROB OF FLOT'/
      3  'NUMBER')
      DO 170  I=1, N
      WRITE(5, 180)I, A(I), REC(I), CREC(I), RT(I), RK(I), PROB(I)
      FORMAT(13, 11X, F8.3, 10X, F8.2, 14X, F8.2, 13X, F8.2, 10X,
      1  F8.5, 9XF8.4)
      CALL EXIT
      END
```

Section III

Flotation Test No. 16

Purpose: To determine rate constants as defined by Bull and by Roos for TiO_2 in bulk sample from each cell.

Sample: Products from Flotation Test No. 16.

Procedure: The results shown below were computed by Computer Program No. 1 using the input variables shown in that program.

Results:

Cell No.	Froth Volume l/min	Froth Solids gram	Froth Solids %	Cell Tailing Volume l/min	Cell Solids gram	Cell Solids %
1	20.59	10435.00	36.6	28.72	9980.13	28.1
2	2.22	1888.00	51.6	26.49	8092.12	25.4
3	0.21	169.53	49.4	26.28	7922.60	25.1
4	0.02	18.06	44.3	26.25	7904.53	25.1
5	0.09	64.53	45.6	26.16	7840.00	25.0
6	0.00	0.00	0.0	26.16	7840.00	25.0

Cell No.	Float Assay	Cell Recovery	Cumulative Recovery	Retn Time	Rate Constant (Bull)	Rate Constant (Roos)
1	34.300	79.03	79.03	83.14	0.04533	3.7693
2	34.200	14.25	93.29	90.12	0.02357	2.1249
3	36.700	1.37	94.66	90.85	0.00283	0.2574
4	37.600	0.14	94.81	90.95	0.00031	0.0289
5	35.900	0.51	95.32	91.27	0.00119	0.1094
6	0.000	0.00	95.32	91.27	0.00000	0.0000

Section III

Flotation Test No. 17

Purpose: As for Test 16.

Sample: Products from Flotation Test No. 17.

Procedure: As for Test 16.

Results:

Cell No.	Froth Volume l/min	Froth Solids gram	Froth Solids %	Cell Tailing Volume l/min	Cell Solids gram	Cell Solids %
1	9.97	7885.00	49.2	33.88	11342.93	27.2
2	2.44	2296.00	54.6	31.43	9046.93	24.1
3	0.13	176.66	64.9	31.29	8870.26	23.8
4	0.11	125.93	58.3	31.18	8744.33	23.6
5	0.06	18.33	22.8	31.11	8726.00	23.6
6	0.00	0.00	0.0	31.11	8726.00	23.6

Cell No.	Float Assay	Cell Recovery	Cumulative Recovery	Retn Time	Rate Constant (Bull)	Rate Constant (Roos)
1	38.700	70.54	70.54	70.47	0.03398	2.3947
2	36.800	19.53	90.07	75.96	0.02590	1.9681
3	38.900	1.58	91.66	76.29	0.00249	0.1905
4	38.400	1.11	92.78	76.58	0.00202	0.1548
5	37.100	0.15	92.93	76.75	0.00029	0.0222
6	0.000	0.00	92.93	76.75	0.00000	0.0000

Section III

Flotation Test No. 18

Purpose: As for Test 16.

Sample: Products from Flotation Test No. 18.

Procedure: As for Test 16.

Results:

Cell No.	Froth Volume l/min	Froth Solids gram	Froth Solids %	Cell Tailing Volume l/min	Cell Solids gram	Cell Solids %
1	5.75	4759.50	50.4	37.75	14879.99	30.9
2	1.98	2275.00	60.8	35.76	12604.99	28.4
3	0.60	753.20	63.5	35.16	11851.79	27.4
4	0.17	211.55	61.7	34.98	11640.25	27.2
5	0.06	68.95	59.5	34.92	11571.30	27.1
6	0.05	57.30	59.2	34.87	11514.00	27.0

Cell No.	Float Assay	Cell Recovery	Cumulative Recovery	Retn Time	Rate Constant (Bull)	Rate Constant (Roos)
1	40.600	48.45	48.45	63.25	0.01486	0.9401
2	40.200	22.93	71.39	66.76	0.01200	0.8015
3	41.000	7.74	79.13	67.91	0.00546	0.3711
4	41.700	2.21	81.34	68.25	0.00173	0.1185
5	41.700	0.72	82.06	68.38	0.00058	0.0402
6	42.400	0.60	82.67	68.48	0.00051	0.0351

Section III

Flotation Test No. 18

Purpose: To calculate the rate constants as defined by Bull and Roos for TiO_2 in each size fraction in each cell.

Sample: Products from Flotation Test No. 18.

Procedure: The results shown below were calculated by computer.

Results:

Cell No.	Mesh Size	Cell Recovery of TiO_2(1)	Cumulative Recovery of TiO_2(1)	Rate Constant (Bull)	Retn Time	Rate Constant (Roos)
1	48.	0.00	0.00	0.00000	67.74	0.00000
	65.	25.06	25.06	0.00493	67.74	0.33451
	100.	29.27	29.27	0.00610	67.74	0.41384
	150.	36.40	36.40	0.00845	67.74	0.57249
	200.	37.62	37.62	0.00890	67.74	0.60325
	270.	40.47	40.47	0.01003	67.74	0.67986
	325.	44.96	44.96	0.01205	67.74	0.81692
	-325.	38.28	38.28	0.00915	67.74	0.62045
2	48.	0.00	0.00	0.00000	77.08	0.00000
	65.	24.00	49.06	0.00611	77.08	0.47120
	100.	32.30	61.57	0.01090	77.08	0.84072
	150.	35.05	71.45	0.01593	77.08	1.22820
	200.	38.97	76.60	0.02161	77.08	1.66620
	270.	35.27	75.74	0.01887	77.08	1.45460
	325.	29.83	74.79	0.01536	77.08	1.18397
	-325.	36.20	74.49	0.01841	77.08	1.41930
3	48.	0.00	0.00	0.00000	81.31	0.00000
	65.	10.66	59.73	0.00325	81.31	0.26490
	100.	10.80	72.37	0.00481	81.31	0.39113
	150.	12.36	83.82	0.00940	81.31	0.76470
	200.	11.90	88.51	0.01275	81.31	1.03703
	270.	13.07	88.82	0.01437	81.31	1.16923
	325.	14.73	89.53	0.01731	81.31	1.40748
	-325.	12.85	87.34	0.01249	81.31	1.01605

Section III

Flotation Test No. 18 (continued)

Results (continued)

Cell No.	Mesh Size	Cell Recovery of TiO_2(1)	Cumulative Recovery of TiO_2(1)	Rate Constant (Bull)	Retn Time	Rate Constant (Roos)
4	48.	0.00	0.00	0.00000	84.61	0.00000
	65.	2.13	61.86	0.00066	84.61	0.05594
	100.	2.34	74.72	0.00109	84.61	0.09263
	150.	2.57	86.40	0.00224	84.61	0.18977
	200.	3.35	91.87	0.00488	84.61	0.41342
	270.	4.47	93.29	0.00787	84.61	0.66666
	325.	2.90	92.43	0.00454	84.61	0.38419
	-325.	4.73	92.07	0.00705	84.61	0.59724
5	48.	0.00	0.00	0.00000	87.08	0.00000
	65.	0.79	62.66	0.00024	87.08	0.02142
	100.	1.06	75.78	0.00050	87.08	0.04395
	150.	1.09	87.50	0.00100	87.08	0.08784
	200.	1.24	93.11	0.00207	87.08	0.18066
	270.	1.53	94.82	0.00339	87.08	0.29568
	325.	2.06	94.49	0.00430	87.08	0.37453
	-325.	2.38	94.46	0.00495	87.08	0.43151
6	48.	0.00	0.00	0.00000	88.43	0.00000
	65.	0.00	62.66	0.00000	88.43	0.00000
	100.	0.26	76.05	0.00012	88.43	0.01111
	150.	0.29	87.79	0.00027	88.43	0.02398
	200.	0.35	93.47	0.00062	88.43	0.05518
	270.	0.44	95.27	0.00106	88.42	0.09454
	325.	0.45	94.95	0.00101	88.43	0.08979
	-325.	1.18	95.65	0.00307	88.43	0.27220

1/ Recovery is the percentage of TiO_2 in that size fraction which reports to the concentrate.