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**THE COMBINED USE OF A SAND SCREW,
HYDROCYCLONES, AND GEL-LOGS TO TREAT
PLACER MINE PROCESS WATER**

**A
Thesis**

**Presented to the Faculty of the
University of Alaska in Partial Fulfillment
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**By
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ABSTRACT

This study describes a low-maintenance tailings treatment system for placer mines. A sand screw and two 20 inch hydrocyclones were used to remove gravel, sand and silt from a sluice box discharge. Gel-logs were introduced subsequent to the main settling pond to reduce the suspended solids. Results drawn from a two year study at a placer mine in interior Alaska indicated that: (1) Most of the plus 50 mm particles can be removed from the sluice tailings by the combination of a sand screw and 20 inch hydrocyclones, (2) When using the hydrocyclone overflow, without further cleaning, in recycle operation, the performance of the sand screw and hydrocyclones change due to the build up of fine particles. (3) In laboratory testing cationic type gel-logs proved to be effective in reducing the turbidity of settling pond water from 4,000 NTU to 80 NTU. However, at the mine this test result could not be duplicated even though 14 logs were applied to treat 25% (c. 500 gpm) of the pond overflow. The cost of installation and operation of the sand screw and hydrocyclone system was about 11% of the total operational cost of the mine in 1986.

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Chapter 1. Introduction

1: Placer Mining in Alaska

The discovery of several significant placer gold deposits at the turn of this century led to the establishment of Alaska as a gold producer, since then the placer mining industry has played a major role in the exploration and development of Alaska and has been of considerable importance to the State's economy (Peterson et al., 1986). The gold produced in Alaska for the past five years is listed in Table 1.

While the price of gold in the world market jumped during this period, the average price in 1986 was \$380/oz, and the amount of gold produced in Alaska showed fluctuations as a result of the controversy between the local placer miners and federal and state environmental regulatory agencies. The major issue has been the quality and impact of the mine effluent discharged from the processing operation.

A placer deposit can be described as unconsolidated material, usually gravel, containing some valuable minerals that were eroded and concentrated in economic quantities. Other than the continuous process in dredging operations, conventional placer mining methods are cyclic in nature consisting of the following sequence: (1) excavation of the unconsolidated material, (2) transportation to the washing plant, (3) loading or feeding the sizing and recovery unit and (4) disposal of the tailings. Equipment used to accomplish these steps may be hydraulic giants, bulldozers, front-end loaders, backhoes, draglines, scrapers, conveyors or some combination of these units. The amount and type of equipment to be employed will depend upon the size and characteristics of the deposit, water supply, economic considerations and the personal preferences of the operator.

To recover gold in the process plant, gravity concentration methods are generally used due to the high specific gravity difference between the gold and the gangue minerals. The concentrating equipment employed may be a sluice box, jig, spiral, cyclone or combination of these units. In

Table 1. Reported Gold Production of Alaska: 1982-1986.

Year	Volume (Troy oz.)	Value (million)
1982	174,900	\$70
1983	169,900	\$67.6
1984	175,000	\$63
1985	190,000	\$62
1986	160,000	\$62

these processes water usage is essential and there are usually no chemicals applied. Therefore, the water quality parameters of primary concern are the total suspended solids (TSS) and settleable solids which result from washing the paydirt. The physical properties of water, temperature, conductivity, pH etc., as used in the process, have not been found to significantly effect effluent characteristics.

2: Water Quality Standards for Placer Mines

All existing gold placer mines are categorized as point sources and direct discharge (U.S.E.P.A., 1985). In determining the level of pollution, settleable solids and turbidity are the two water quality parameters commonly used.

The 1972 Federal Water Pollution Control Act (FWPCA, PL 92-500), started a new era in the water quality movement. It regulated the discharges from point sources, prohibited the inclusion of hazardous substances, included financial assistance for wastewater treatment plant construction, and created a permit system. Under section 301, effluent limitations for all point sources were required to reflect 'best practical control technology currently available' (BPT) by July 1, 1977, and 'best available technology economically achievable' (BAT) by July 1, 1983. A permit system, National Pollutant Discharge Elimination System (NPDES), created by section 402 in this Act declared that any discharge except in compliance with the limitations imposed in a permit, was unlawful. Section 208 of the Act also required all States to develop a continuous planning process and elaborate state and local water quality management plans.

A new amendment to the Act in 1977 was designated the Clean Water Act, (PL 95-217). In regards to the previous deadline for industry compliance with effluent limitations, the revisions adopted different requirements for different pollutants (Section 301, 304, and 307). For conventional pollutants, a new standard 'best conventional pollution control technology' (BCT), was established to be achieved by July 1, 1984. For the distinct mining and processing methods used in placer gold recovery, the Environmental Protection Agency (U.S.E.P.A) proposed a separate effluent limitations guidelines and standards specifically for them in 1985. It determined that based on technological and economic considerations these standards are appropriate for various groups of placer mines dependent upon the mining method and processing capacity. The proposed standards are effluent limits of 0.2 ml/l, for settleable solids, and 2,000 mg/l, for total suspended solid; these were listed for all operations processing greater than 20 yds³/day. Complete recycle of process waters were required for operations processing greater than 500 yds³/day. Design requirements for the sizing of settling ponds were also included.

Authority for adoption of State Water Quality Standards was vested in the Alaska Department of Environmental Conservation (ADEC) by the Alaska Legislation in Alaska Statutes Title 46, Chapter 3. As a result, Alaska Water Quality Standards designed to identify the uses of the State's water and set criteria for man-induced changes in natural water quality was established as Title 18, Chapter 70 of Alaska Administrative Code in October 1973 and amended in 1979.

The standards regarding turbidity were established not to exceed 5 Nephelometric Turbidity Units (NTU's) above background conditions when the natural turbidity is 50 NTU's or less and when the natural condition is more than 50 NTU's not to exceed a maximum increase of 25 NTU's. For settleable solids, no measurable increase in concentration above natural conditions was the established criteria.

The enforcement system for the State of Alaska was established as a tri-agency group involving the Department of Environmental Conservation, the Department of Fish and Game, and the Department of Natural Resources. As a result of this effort, the state has generated a list of priority streams, based on conflicting uses, on which the agencies will focus their enforcement efforts. Prior to operation, all mines must complete a tri-agency form, which serves as an annual application for a land and water use permit and a mining license. The State's placer mining enforcement policy will focus on flagrant violators who do not implement minimum pollution control measures.

3: Water Treatment Methods Used in Placer Mines

With the stringent water quality standards applied to the mining operation, the placer mining industry was faced with further cleaning their process water prior to discharging it into a receiving stream. Industry efforts have been assisted through the State's Placer Mining Grant Project to control the water pollution problems. These relatively simple techniques incorporating common mining procedures such as classification, recycle, by pass, improved settling pond design, and tailings filtration proved effective in reducing the volume of effluent. Another effective technique utilizing flocculants was successful in reducing turbidity at some mines, however, the quantity of water to be treated by this method is a determining factor as well. In general, the effectiveness of a water treatment method is highly dependent on the volume of effluent, therefore, reducing the water usage in a mining operation is a logical approach (Peterson et al., 1987).

Settling ponds are the most common means for removing sediment from placer mining process water, and are a required treatment scheme in Alaska. They provide a

relatively large, quiescent area where settleable solids can settle out, but generally are not conducive to settle fine silt and clays. To be effective the ponds must have sufficient surface area and have enough storage capacity to achieve the desired performance which is a function of a well engineered design. However, common characteristics of settling ponds used in Alaska studied by Brinbridge, (1979) indicated that those ponds are only partially effective because of their small size, generally shallow depth, and short retention time. Other factors which may limit the use of settling ponds are: the large land area required to build the ponds which may not be available at some mines, the man-hours required to operate and maintain the ponds increases the operational cost and non-ideal settling caused by short circuiting and weather conditions may disturb the performance of the ponds (Chapman, 1986). The extensive use of settling ponds as a treatment method is mainly due to their low cost relative to other treatment techniques (Peterson et al., 1984).

Because of the presence of clay and silt, turbidity removal is more difficult, and the 5 and 25 NTU criteria are virtually impossible to attain with settling ponds alone. In recent years, another method using chemical flocculants was introduced to Alaska and had limited success. Studies by Weber (1985) showed that inadequate mixing of the chemicals and the ultra-fine clay particles may hinder the effectiveness of this technique. U. S. Bureau of Mines (Smelley and Scheiner, 1986) performed tests with polyethylene oxide (PEO) at several Alaskan placer mines with promising results. In the Yukon, Canada a polyacrylamide was applied in placer mining operations leading to decreases in the amount of suspended solids being released into streams (Reid Crowther & Partners and Bethell Management, 1984). However, the high price of the flocculants, the mixing equipment needed to prepare the working solution and constant monitoring are cost factors for this system that should be studied over a long-term period.

4: Purpose and Scope

In conjunction with increasing environmental concerns placer miners are required to remove settleable and non-settleable solids from their effluent. In contrast to the standard settling pond configuration, which is commonly used to treat wastewater at most placer gold mines, this study focuses on another possible water treatment system which involves a sand screw, hydrocyclones, and gel-logs. A similar system has found successful application in a placer mine located in Northern California where 'the mine can operate with sensitivity to the environment' (Mining Engineering, March 1987). Therefore, the applicability of this water treatment system to an Alaskan placer mine was studied during the 1986 and 1987 mining seasons. Performance information for the sand screw and hydrocyclones,

properties of the process water under different modes of operation, the use of gel-logs as a means to reduce turbidity and an cost analysis of this system are examined in this study.

Chapter 2: Sand Screw and Hydrocyclone

1: Particle Fluid Interactions

The behavior of particles in a carrying fluid can be ideally described by an individual spherical-shaped particle in an infinite pool of still fluid. The particle then is acted upon by three forces: a gravity force, a buoyant force due to the displaced fluid, and a drag force F_D . The equation of particle motion is,

$$mg - m_o g - F_D = m \frac{dv}{dt} \quad (1)$$

where m is the mass of the particle, m_o is the mass of displaced fluid, v is the particle velocity, and g is the acceleration due to gravity. When these forces reached equilibrium the particle is falling at a constant velocity called terminal velocity.

The action of gravity and buoyant forces are obvious; however, the drag force is more complicated in nature and the understanding of its influence on the terminal velocity is important. Drag force is related to the relative velocity between particle and fluid, which is caused by two phenomena; viscous drag due to fluid viscosity, and pressure drag where velocity change around particle cause pressure imbalance. The relative importance of these factors depends on the fluid dynamic condition of particle encounters; this can be characterized by the values of the Reynolds number (R_p) of the particle which is,

$$R_p = \rho v d / \eta \quad (2)$$

where ρ is the fluid density, v is the particle velocity, d is particle diameter, and η is fluid absolute viscosity. The influence of pressure drag increases as the value of R_p increases. Three regimes of flow condition are recognized: laminar, transitional, and turbulent flow. The laminar flow condition prevails when the R_p value is less than 0.2; the turbulent condition prevails when the relative velocity is high and R_p exceeds 1,000. Intermediate R_p values lie in the transitional regime.

A general equation of the drag force can be expressed as:

$$F_D = C_D A \rho v^2 / 2 \quad (3)$$

where C_D is the drag coefficient, A is the projected area of the particle normal to the direction of motion and $\rho v^2 / 2$ is

termed the dynamic pressure.

In the laminar regime, drag force on a spherical particle is assumed to be entirely due to viscous resistance. The drag force is

$$F_D = 3\pi d \eta v_t \quad (4)$$

where v_t is terminal velocity. From equation (1) and equation (4) give

$$v_t = \frac{gd^2(\rho_s - \rho)}{18\eta} \quad (5)$$

This expression is known as Stokes' law.

Newton, on the other hand, assumed the drag force was totally due to turbulence, and the value of C_D is close to 0.4 for a Reynolds number between 10^3 and 10^5 .

Substituting equation (3) in equation (1) gives

$$v_t = \left(\frac{3.33gd(\rho_s - \rho)}{\rho} \right)^{1/2} \quad (6)$$

Newton's law.

In the transitional regime there is no simple expression of F_D , and the values of drag coefficient, C_D , require iterative calculation by using the equation

$$C_D = (21/R_p) + (6/R_p^{0.5}) + 0.28 \quad (7)$$

Generally, the particle mass determines the terminal velocity, but it is dependent on particle shape and on the rheology of the fluid as well. However, in practice, a number of factors other than those mentioned above could influence the magnitude of the terminal velocity such as wall effects, Brownian motion, particle spin and surface roughness.

Further, as the concentration of solids contained in the liquid increases, hindered settling conditions are established. Richardson and Zaki have shown that for 10 to 1000 micron spherical particles, the terminal velocity attained under hindered settling is given by

$$V_t = V_u (1 - C_v)^n \quad (8)$$

where C_v is the volume concentration of solids and V_u is the free settling velocity (Smith and Gochin, 1984). The index n is related to particle Reynolds number thus:

$$\begin{aligned} \text{for } R_p > 500, n &= 2.39 \\ 500 > R_p > 0.2, n &= 4.4/R_p^{0.1} \\ R_p < 0.2, n &= 4.65 \end{aligned}$$

2: Sand Screw

As shown in Figure 1, the sand screw, or spiral classifier, consists of a settling tank with parallel sides and a sloping bottom equipped with a spiral which rotates without contacting the sides or bottom of the tank. The name spiral explains its function, which is to provide the necessary pool agitation and to convey the settled sand to the discharge lip. Inside the tank, the upper portion is the drainage section. Here, the settled solids carried by the spiral are allowed to dewater. The lower portion is the settling pool, which has a flared structure to increase the pool area.

Feed is introduced at the pool level. This level is maintained by adjusting the height of the overflow weir, which also regulates the overflow discharge. The operation of this unit is based on the principle that with the introduction of the feed a settling pool is formed in which particles of high settling velocity quickly fall to the bottom of the tank. Above this coarse sand layer is a quicksand zone where essentially hindered settling takes place. The depth and shape of this zone depends on the classifier action, and on the feed pulp density. Above the quicksand zone is a zone of essentially free settling material, comprising a stream of pulp flowing horizontally across the top of the quicksand zone from the feed inlet to the overflow weir, where the fines overflow (Wills, 1982).

Sand screws are mainly utilized in close-circuit wet grinding operations, with other applications include desliming, dewatering and washing operations. The functions of sand screws in these applications are (Hitzrot, et al., 1985):

- (1) allow particles larger than the desired size to settle in the tank and provide an overflow product with a minimum of oversize particles;
- (2) produce an overflow product of sufficient solids content to meet the requirements of the subsequent processing steps;
- (3) agitate the pulp and permit separation of the entrapped undersize particles so they can report in the overflow product; and
- (4) drain and remove the underflow solids from the settling pool.

The information needed to select a sand screw includes the properties of the solids and liquid which will be processed. For solids it requires: feed rate, chemical and physical composition, density, size analysis, and desired separation size. For the liquid the following data would be needed: feed rate, density, viscosity, pH and temperature.

Design and operational variables (Hitzrot et al., 1985):

- (1) **Settling Area:** The pool area required to permit a particle larger than the separation size to settle depends on density and shape of particles, and on the density and viscosity of pulp within the pool. These criteria of the solids and pulp determine the settling rate which, in turn, determines the settling area required.
- (2) **Overflow Weir:** The hydraulic head, and thus the velocity at which the overflow crests the overflow weir, is one of the factors controlling the particle size of classification. Consequently, the width of the overflow weir must be designed to provide the overflow velocity which allows the particles larger than the desired size to settle in the working area of the pool.
- (3) **Settling Zone:** The several zones within the settling tank described above also determine the separation size, generally, the smaller the classification size desired, the greater must be the depth of the gently agitated settling pool above the turbulence caused by the spiral.
- (4) **Agitation:** Agitation generated by the spiral that removes the underflow sand liberates much of the entrapped water and allows undersize particles to rise and report as the overflow product. Increasing the speed of the raking increases the agitation, and thus increases the size of the particles reporting in the overflow.
- (5) **Tank slope:** The slope of the tank bottom has a direct influence on pool area, sand-raking capacity and sand discharge. It thereby, affects the separation size and moisture content of the sand discharge product.
- (6) **Wash water:** adequate removal of entrapped water and undersize particles from sand products frequently requires providing wash-water sprays on the drainage section. However, the sprays wash back much of the solids which has the net effect of reducing the sand-raking capacity in terms of the final sand product removed.

3: Hydrocyclone

Since their first reported application in the mineral industry in 1938, cyclones have become a common separation device in all of the the process industries. The widespread use of cyclones stems from their relatively low cost and their versatility. They can be employed to carry out liquid-liquid, solid-liquid and solid-solid separations (Flintoff and Plitt, 1985). The basic operation principle employed in hydrocyclones is centrifugal sedimentation, i.e. the sus-

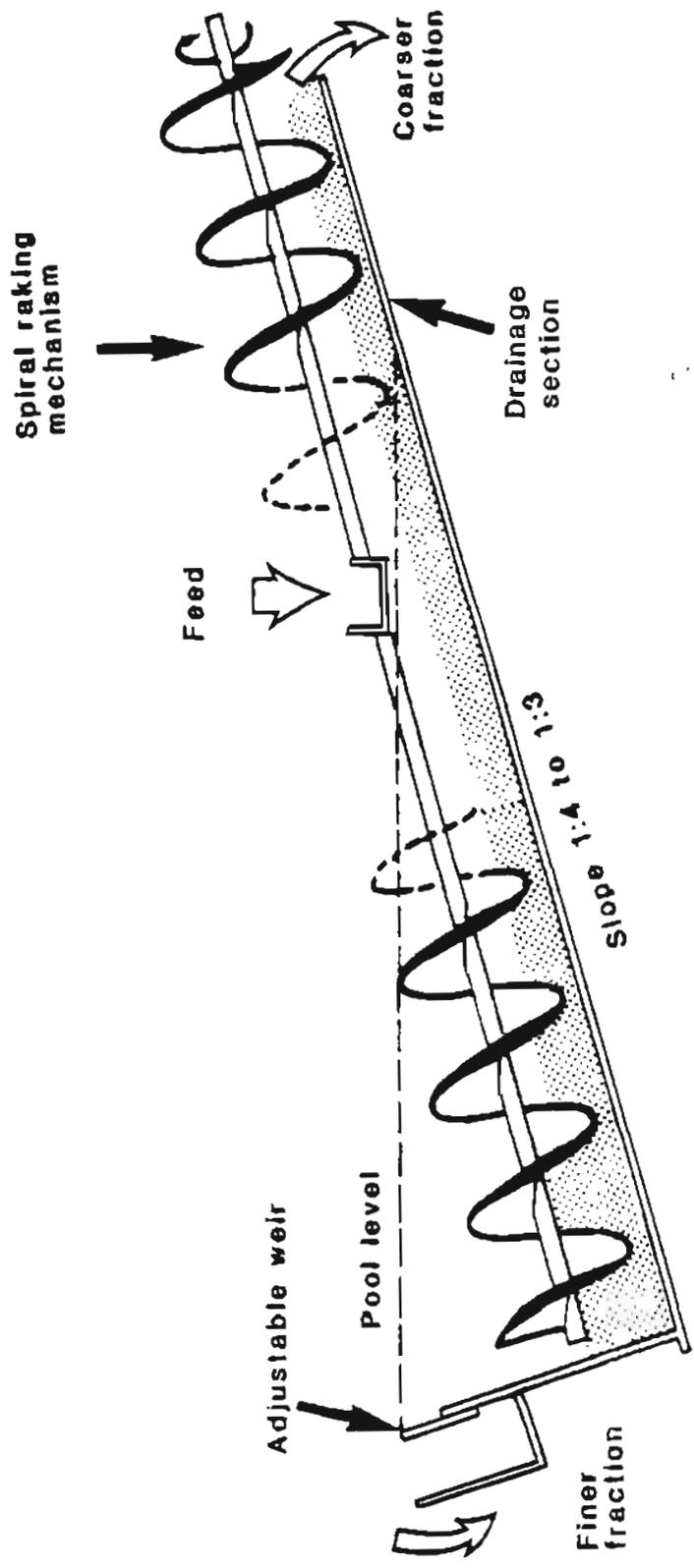


Figure 1. The Operation of a Sand Screw (Smith and Gochlin, 1984).

pended particles are subject to centrifugal acceleration, which makes them separate from the fluid. Unlike centrifuges, cyclones have no moving parts and the necessary vortex motion is performed by the fluid in conjunction with the design configuration.

The structure of a hydrocyclone is a top cylindrical section with a tapering hollow cone as the bottom section, as shown in Figure 2. The suspension of particles is injected tangentially through the inlet opening in the upper part of the cylindrical section and, as a result of the tangential entry, a strong swirling motion is developed within the cyclone that generates the primary vortex. With the tapered structure of the conical section, only part of the suspension is discharged as underflow carrying the coarse particles. The bulk of the liquid carrying the residual fine particles forms an upward secondary vortex surrounding the core of the casing. Inside the core, a low pressure is generated, collecting all the air that has been carried in as bubbles or dissolved in the feed slurry (Truwinski, 1976). The separation size or cut point of a hydrocyclone refers to a specific size at which 50% of the particles report to each of the cyclone products. The size is often termed d_{50} size.

Flow Pattern and Particle Motion in A Cyclone

A knowledge of the flow pattern in the cyclone is essential for understanding its function and subsequently for the optimal design and evaluation of the particle behavior, which in turn allows prediction of the performance.

The flow pattern in a hydrocyclone has a circular symmetry, as indicated in Figure 2. This consists of a primary vortex containing a secondary vortex, with the exception of the region in and just around the tangential inlet duct where a short circuit flow occurs against the roof due to an obstruction of the tangential velocity. An eddy flow also exists because the overflow opening cannot handle the natural upflowing vortex. Figure 3 illustrates the above two phenomenon. The other two flow patterns, shown in Figure 4, are the locus of zero vertical velocity (LZVV) which follows the profile of the hydrocyclone, and the air core which rises from the apex and passes out through the vortex finder indicating the stability of the vortex.

The velocity of flow at any point within the cyclone can be resolved into three components: the tangential velocity V_t , the radial velocity V_r , and the vertical or axial velocity V_a (Kesall, 1953).

Kesall's experimental data, shown in Figure 5, were obtained with an optical device which did not interfere with the flow. In spite of their limitations consisting of the diluted suspension of aluminum particles and the cyclone dimensions, these data are the basis of some theoretical correla-

tions. At a level below the rim of the vortex finder, the tangential velocity V_t increases considerably with decreasing radius down to a given radius, which is smaller than the exit radius of the vortex finder. This can be described by the relationship

$$V_t r^n = \text{constant} \quad (9)$$

where n is normally $0.6 < n < 0.9$. As the radius is further decreased, the tangential velocity decreases and is proportional to r , this relationship holds until the cylindrical air column is reached.

For the vertical velocity, there is a strong downward flow along the outer wall of both the cylindrical and conical portions. This flow is essential for cyclone operation since it removes the particles that have been separated into the underflow orifice. This downward current is partially counterbalanced by an upward flow in the core region, depending on the underflow-to-throughput ratio.

The radial velocity is inward and its magnitude decreases with decreasing radius. This component is normally much smaller than the other two components and difficult to measure accurately, however, it can be calculated from continuity conditions.

The gravity effect is normally neglected in hydrocyclones, so that only centrifugal and drag forces are considered which account for the motion of particles. The movement of a particle in both the tangential and vertical directions are unmatched by any forces, so that the velocity component in these directions can be taken to be equal to the corresponding flow velocity component V_t and V_a . Since centrifugal force acts in a radial direction it prevents the particles following the inward radial. Therefore, if centrifugal forces acting on a particle exceed the drag, the particle moves radially outwards and, if the drag is greater, the particle is carried inwards.

When solid particles enter near the cylindrical wall they can be dispersed radially inwards because of the intensive turbulent mixing in the feed section. There is, however, very little information about the behavior of the liquid in the cylindrical section; this portion of the cyclone is usually treated as a preliminary separation zone and the precise separations are thought to be performed in the conical section.

As was suggested by Kesall, any particles present in the downward flow near the conical wall, can move radially inward only if the liquid moves inward. It is, therefore, obvious that if a fraction R_f of the feed liquid goes into the underflow, then the same fraction R_f of all particles, inde-

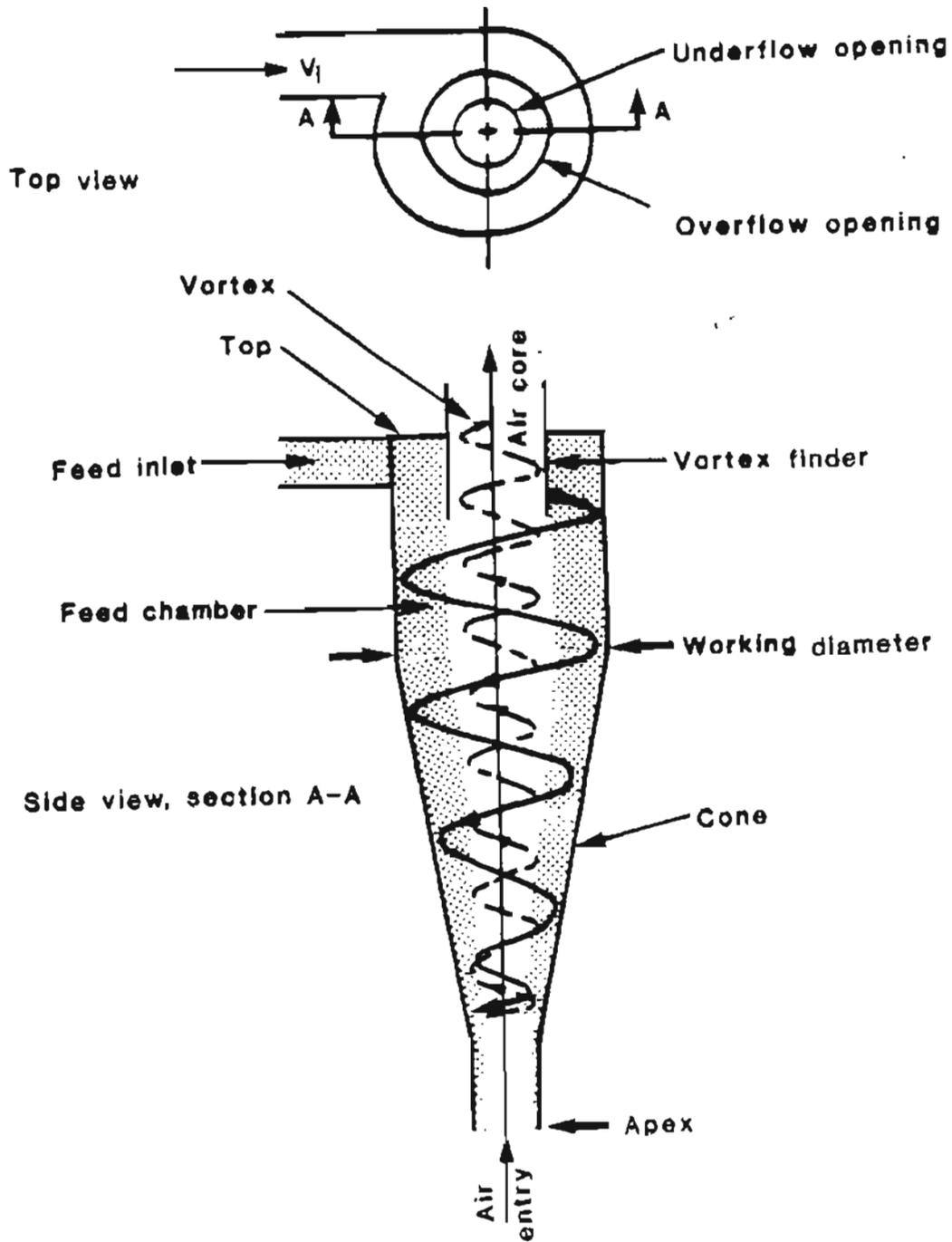


Figure 2. A Typical Hydrocyclone in Operation (Whitla, 1982).

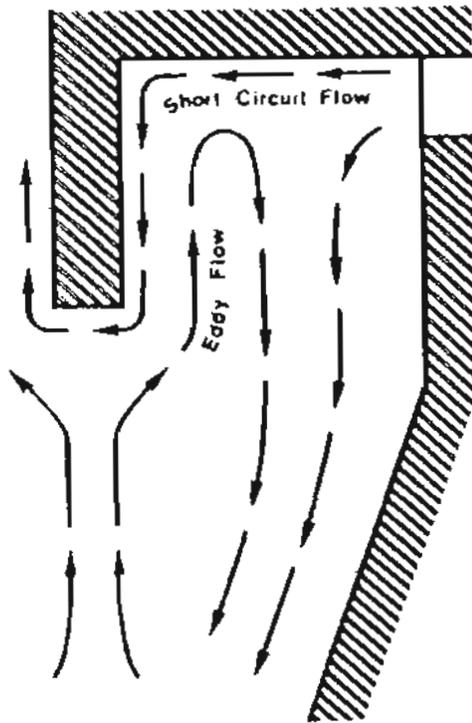


Figure 3. Flow Patterns in the Upper Region of a Hydrocyclone (Bradley, 1959).

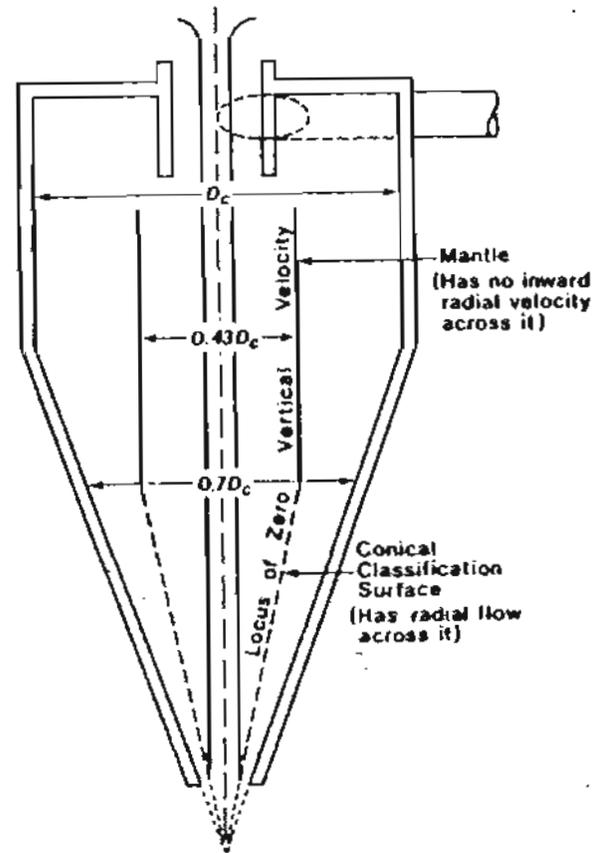


Figure 4. The LZVV In a Hydrocyclone (Bradley, 1959).

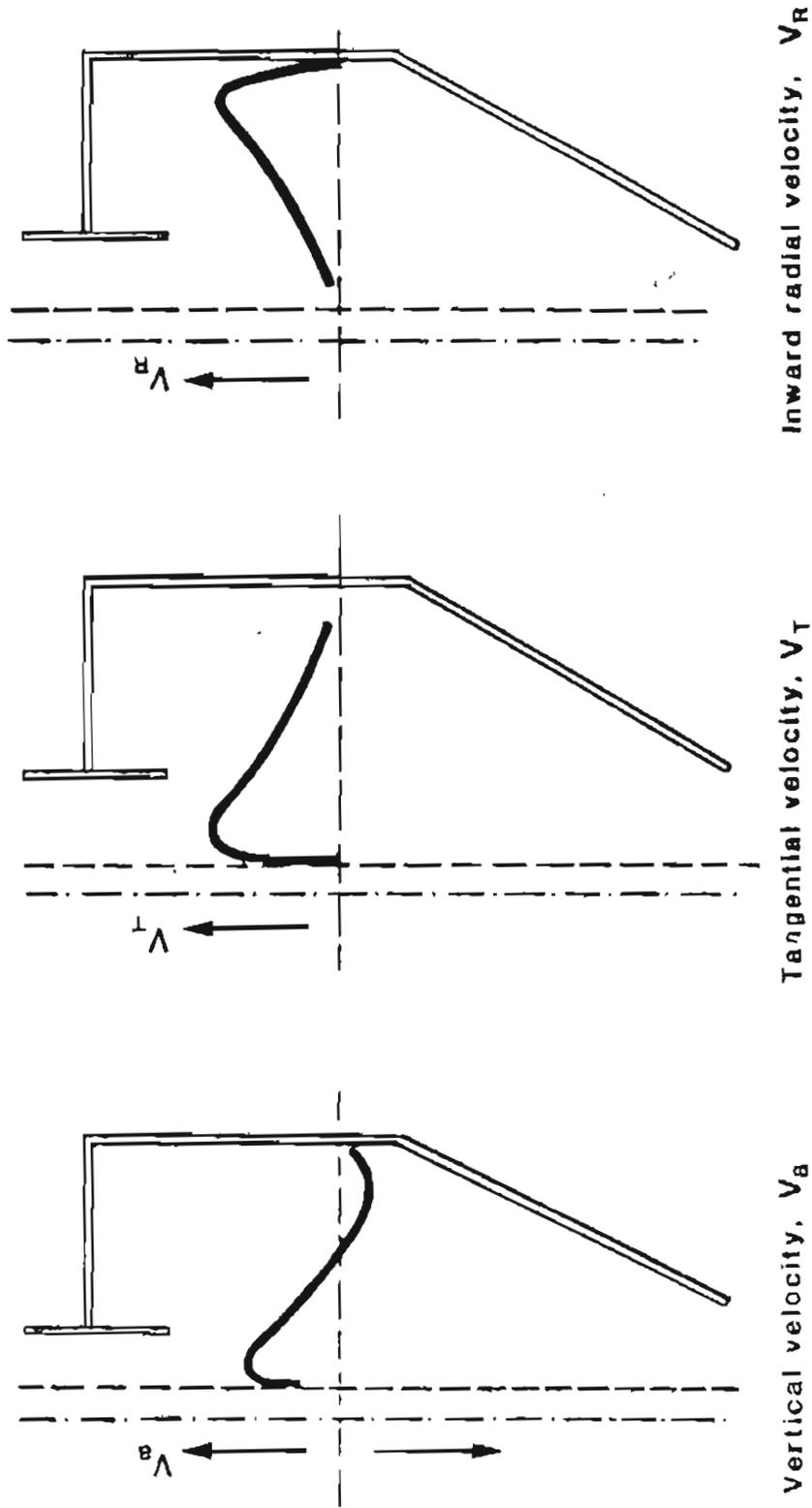


Figure 5. Fluid Velocities in a Hydrocyclone (Smith and Gochin, 1984).

pendent of their settling rate, must also go with the liquid, together with the particles separated from the remaining fraction of the liquid $1 - R_f$ leaving the underflow (Svarovsky, 1984).

The variables which influence the behavior of the hydrocyclone can be broadly classified into two categories, design variable and operating variable (Nageswararao, 1985).

Design variables:

- (1) diameter of the cyclone.
- (2) feed inlet diameter.
- (3) shape and geometry of feed inlet.
- (4) position of inlet.
- (5) type of feed inlet.
- (6) vortex finder diameter.
- (7) length of vortex finder.
- (8) spigot diameter.
- (9) length of the cylindrical section.
- (10) cone angle.
- (11) interior lining material.
- (12) collection arrangements of the products.

Operating variables:

- (1) properties of the fluid media.
- (2) specific gravity of the feed solid.
- (3) size and shape distribution of the feed solid.
- (4) solid content of the feed pulp.
- (5) cyclone feed pressure.
- (6) apparent viscosity of the pulp.

Cyclone operations can be classified according to the main characteristics of the products. Five basic operations cited are (Weyhar, 1969):

- (1) Clarification - characterized by a vortex product with a minimum of particles, e.g., dust collecting cyclone.
- (2) Thickening - characterized by maximum solid/liquid ratio in the apex-product, as in the preparation of feed slurry for a filter.
- (3) Classification - characterized by the separation of particles into size ranges by utilizing the strong influence of the size of particle on its terminal settling velocity. An example is its use in closed circuit grinding operations.
- (4) Gravity separation in a dense medium - characterized by separating particles according to their specific gravity, utilizing the opposed buoyant forces acting on particles of different densities if suspended in a liquid of intermediate density.

- (5) Gravity separation without use of a dense media - characterized by force conditions emphasizing particle density.

Modeling of a Hydrocyclone

For the past decade, attempts have been conducted in the mineral industry to mathematically quantify the hydrocyclone which is important in performance analysis, process simulation, optimal design, and on-line size analysis. Despite its geometric and operational simplicity, the separation mechanisms within a hydrocyclone are complex and not well understood. The theoretical model still is of little success, while empirical modeling which is based on the observation have received considerable attention.

There were two cyclone models presented in the mid seventy's which have been widely used in the mineral processing industry; one developed by Lynch and co-workers (1975) in Australia and the other developed by Plitt (1976) in Canada. They have been applied for many years in plant practice and both of them have had an updated version (Nageswararao, 1985 and Flintoff and Plitt, 1985).

These two models rely on the supposition that three basic parameters must be quantified for any given set of operating conditions. These are :

- (a) The flow split of water between the overflow and underflow products.
- (b) The corrected cut size or cut point.
- (c) The classification efficiency.

Generalized Lynch's Cyclone Model:

This model assumed fluid characteristics do not influence the performance of a hydrocyclone because of the highly turbulent flow condition in the hydrocyclone. The solids properties, such as density, shape etc. are incorporated in a constant called material constant which is to be determined before use in different situations, and the functional relationship between independent variables are mononormal power functions.

The performance parameters are:

- (a) Capacity: The throughput to different design and operating variables of the hydrocyclone determined from the equation:

$$Q = K_{Qc} D_c^{1.9} (D_v/D_c)^{0.67} (D_f/D_c)^{0.45} (L_c/D_c)^{0.2} \theta^{-0.1} (P/\rho_p)^{0.5} \quad (10)$$

- (b) Cut Size: The size at which the cyclone makes its separation (not accounting for the fraction of

feed that by-passes the classification process) is calculated from the equation:

$$d_{50c}/D_c = K_{do} D_c^{-0.65} (D_i/D_c)^{0.52} (D_v/D_c)^{0.47} (D_f/D_c)^{-0.5} (L_c/D_c)^{0.2} \theta^{0.15} \quad (11)$$

(c) Classification Efficiency: The normalized corrected efficiency curve is described by the equation:

$$E_c = \frac{(\exp \alpha (d/d_{50c}) - 1)}{(\exp \alpha (d/d_{50c}) + \exp \alpha - 2)} \quad (12)$$

where α is constant, to be determined for a given installation. The higher the value of α the more efficient is the separation.

(d) Water Recovery to Underflow and Volumetric Recovery of Pulp to Underflow: The value of water recovery is equal to the mass fraction of feed solids flowing directly to the underflow product.

$$R_f = \frac{K_{wo} (D_v/D_c)^{2.4} (L_c/D_c)^{0.22}}{(D_i/D_c)^{0.5} (D_v/D_c)^{1.19} \theta^{0.24} (P/\rho_p g D_c)^{0.53} \lambda^{0.27}} \quad (13)$$

$$R_v = \frac{K_{vo} (D_v/D_c)^{1.83} (L_c/D_c)^{0.22}}{(D_v/D_c)^{0.94} (D_i/D_c)^{0.25} \theta^{0.24} (P/\rho_p g D_c)^{0.31}} \quad (14)$$

where: K_{Qc} , K_{do} , K_{wo} , and K_{vo} are material constants

D_c , D_i , D_v , and D_u are dimensions of the cyclone, the inlet, the vortex finder and the apex, cm

L_c : free vortex length, cm

θ : cone angle

P : cyclone feed pressure, KPa

ρ_p : pulp density of cyclone feed, Kg/l

β : volumetric fraction of solids in cyclone feed

λ : $\beta/(1-\beta)^3$

g : acceleration due to gravity, m/sec²

A General Cyclone Model. Plitt's Model:

In this model both the fluid and solids properties are in an explicit form like the other independent variables.

The parameter estimated:

(a) Cut Size: The classification size of the cyclone is calculated from the equation:

$$d_{50c} = \frac{F_1 39.7 D_c^{0.46} D_i^{0.6} D_o^{1.21} \eta^{0.5} \exp(0.063 \phi)}{D_u^{0.71} h^{0.38} Q^{0.45} \left[\frac{(\rho_s - 1)}{1.6} \right]^{\gamma}} \quad (15)$$

(b) Classification Efficiency: The sharpness of the classification is characterized by the value of an exponent m in an equation that is used to describe the form of the normalized corrected efficiency curve:

$$p_i = R_f + (1 - R_f) \left(1 - \exp \left(-0.693 \left[\frac{d_i}{d_{50c}} \right]^E \right) \right) \quad (16)$$

$$E = F_2 1.94 \exp \left[-1.58 \frac{S}{(1+S)} \right] \left[\frac{D_c^2 h}{Q} \right]^{0.15} \quad (17)$$

(c) Pressure Drop: The relationship between pressure drop and the design and operational parameters is given by:

$$P = \frac{F_3 1.88 Q^{1.78} \exp(0.0055 \phi)}{D_c^{0.37} D_i^{0.94} h^{0.28} (D_u^2 + D_o^2)^{0.87}} \quad (18)$$

(d) Water Split: In this model pulp split is determined instead of water flow, as shown in the following equation:

$$S = \frac{F_4 18.62 \rho_p [D_v/D_o]^{3.31} h^{0.54} (D_u^2 + D_o^2)^{0.36} \exp(0.0054 \phi)}{D_c^{1.11} P} \quad (19)$$

and the value of the required water recovery to underflow R_f by:

$$R_f = \frac{\frac{S}{(1+S)} - \frac{R_g \phi}{100}}{1 - \frac{\phi}{100}} \quad (20)$$

Since R_f is in itself a function of the water recovery, the final value of R_f and of the cyclone products must be determined by an iterative method.

Where: F_1 , F_2 , F_3 , and F_4 : are calibration parameters, usually set = 1

D_c , D_i , D_v , D_u : are the cyclone diameter, inlet orifice, vortex finder orifice, and apex orifice, cm

E: sharpness of separation coefficient
 Q: volumetric flow rate of cyclone feed, l/min
 P: cyclone inlet pressure, KPa
 R_f : mass recovery of water
 R_s : mass recovery of feed solids
 S: volumetric flow split
 h: free vortex height in cyclone, cm
 p: partition factor of certain particle sizes, %
 γ : exponent for the effect of solid density on d_{50}
 η : viscosity of carrier fluid, cp
 ρ_p and ρ_s : density of pulp and solid
 ϕ : volumetric concentration of solid in cyclone feed, %

Comparison of these two models:

In general, both of the models are derived from experimental data conducted by the researchers; however, there are some differences present in their models. First, the classification mechanism assumed in Plitt's model is the Rosin-Rammler function, while Lynch's model uses an exponential sum. The parameters estimated by Plitt are: P, d_{50} , S, and m, but in Lynch's are: Q, d_{50} , R_f , and R_s . In modifications to their previous versions, in Plitt's model solids density effect is taken into consideration, and Lynch's model incorporates variables as D_i , L_c , and θ .

Lynch's model needs additional test work to determine material constants for different applications. On the other hand, Plitt's model is easier to use involving more data (297 data sets were analyzed). Therefore, only Plitt's model is adopted in this study with the purpose of examining the performance of hydrocyclones in the SS-H system.

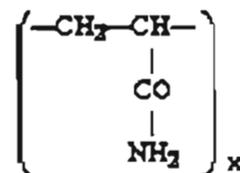
Chapter 3: Flocculants and Flocculation

I: Flocculant Chemistry

Generally, there are three types of flocculating agents (Purdy, 1986): (1) Inorganic electrolytes such as lime and alum are used to provide a cationic charge where needed; (2) Natural polymers such as starch and guar gum have been used in the past and still find isolated application today; (3) Synthetic polymers are water soluble polymers with high molecular weight (>100,000 AMU), capable of promoting flocculation of solids in a liquid suspension. Other commonly used terms are polymeric flocculants, polyelectrolytes, polymers, and flocculants.

For many engineering applications, such as solid-liquid separation, and dewatering, synthetic polymers are more cost effective. Based on the charge they carry synthetic flocculants fall into three classes: non-ionic, anionic, and cationic. The following discussion gives general characteristics and applications of some commonly used flocculants.

Non-ionic: The most common non-ionic type synthetic polymer is polyacrylamide:



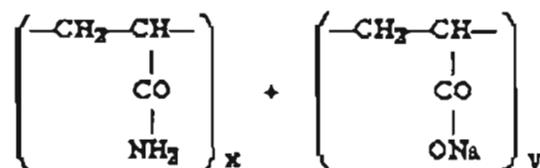
Less commonly used, due to its cost, but very effective in certain applications (Scheiner et al., 1985) is polyethylene oxide:



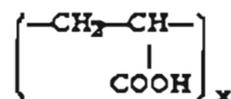
Usually these polymers are used in neutral or acidic pulps. They are also effective in basic pulps in combination with lime or synthetic cationic flocculants.

Anionic: Sodium or ammonium acrylate can be copolymerized with acrylamide, or polyacrylamide can be partially hydrolyzed to form the most common anionic synthetic flocculant.

Acrylamide and sodium acrylate:



Polyacrylics:

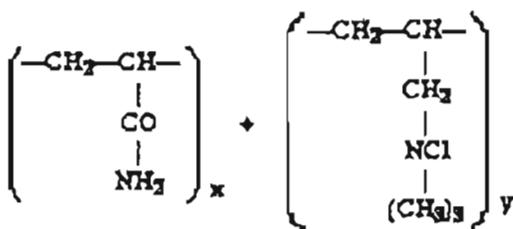


These polymers are usually used in neutral or basic circuits, since acrylates are not ionized at a low pH.

Cationic: Many different types of synthetic cationic polymers are commercially available, all based on a positively charged nitrogen in an amine group. Two types predominate in the mineral industry. The most common is a polyamide:



The second type is a copolymer of acrylamide and propyl trimethylammonium:



In both anionic and cationic types, y can be varied from 5 to 100% and thus a wide range of polymers from totally cationic through non-ionic to fully anionic can be produced. These electrically charged flocculants have certain advantages over the non-ionic flocculants. They accomplish charge neutralization and bridging as one function. Also, the charge they carry can repel itself which helps uncoil the flocculant; this phenomenon is called polyelectrolyte effect. The degree of polymerization, x or $x + y$, can also be varied to produce a range of polymers of different average molecular weight. Thus flocculants can be 'tailor made' to suit virtually every pulp and slurry and dewatering duty.

Several characteristics of the polymers and the environment of their use are important for proper flocculant choice (Purdy, 1986).

- (1) **Molecular Weight:** An important aspect in the development of synthetic flocculants has been concerned with attempts to attain higher intrinsic viscosity and hence, high molecular weight. The higher molecular weight flocculants can improve the solids recovery and they tend to induce larger shear resistant flocs.
- (2) **Charge and Charge density:** The type of charge necessary will depend on the characteristics of the solids and other factors in the pulp. The degree of charge, anionic or cationic, can be varied by choosing the appropriate flocculant. Testing of different charge densities is recommended in all cases to find the most efficient combination, since the optimal charge cannot be predicted in most instances.
- (3) **Dosage, Flocculant Dilution, and Shear:** For every flocculant application there is an optimum dosage, above which the settling velocity of flocs suffers. Excess polymer can restabilize or disperse the slurry. Diluting the flocculant before adding to the pulp is necessary for better mixing. Concentrated flocculant solutions are very viscous and it is difficult to get thorough flocculant/substrate contact. Shear forces will break long polymer chains and cause floc degradation which can only be counteracted by subsequent addition of more flocculant. Mixing sufficiently intense

for good polymer dispersion through the pulp is necessary but high shear conditions should be avoided.

- (4) **Pulp Density and Particle Size:** The highest practical pulp density is favorable for faster settling and more efficient floc formation, assuming sufficient flocculant dilution and good mixing. Finer solids particles will virtually always require higher polymer dosages for effective flocculation.
- (5) **Pulp pH and Ionic Strength:** The surface charge of most mineral particles varies with pH. Anionic polymers are most useful in natural to basic condition, non-ionic polymers are relatively unaffected by pH. Polymers, especially of the high molecular weight anionic variety, are less soluble and less efficient in high ionic strength solutions.

2: Gel-log

An alternative approach to conventional liquids and powders is a new gelatinous form of polymeric flocculant. This polymeric gel is usually sold as 40 lb cylindrical 'logs'. These logs are 9 inches in diameter by 18 inches long. As with traditional liquids and powder polymers, various cationic, anionic, and non-ionic formulations are available.

Advantages of utilizing the gel-log system are (Aldrich et al., 1987):

- (1) The gel-logs are easy to handle and require little equipment investment and maintenance and no electrical connections.
- (2) Dosage rates can be easily adjusted by addition or removal of gel-logs.
- (3) The system can function unattended and automatically responds to variation in flow rates, because the water flow over the logs regulates the material dispersal into the system.
- (4) The gel-log polymers can be effective at low dosage, this is due to the slowly dissolving flocculant permitting the polymers to fully uncoil. In addition, there is no mixer used, which tends to shear a long chain polymer.

Three factors affect the dissolution rate of a gel-log cited by Coal Age (1982): (1) Water velocity of the flow in which gel-logs are applied, (2) The surface area of gel-logs and the portion of water flow contacting the logs, and (3) Water temperature.

Gel-logs have been used in the mineral industry since the early eighties and have found to be successful in several coal mines drainage treatment applications (Coal age, 1982 and Ackman et al., 1984). These applications concluded that pH control was an important factor in improving sludge settling.

3: Mechanism of Flocculation

Flocculation is an irreversible kinetic process in which two reactions, Figure 6, take place: (1) charge neutralization and collision, followed by (2) bridging and agglomeration. Normally, a particle in suspension will not be attracted to a particle of the same surface charge and no collision will occur. When a polymeric flocculant of the proper functionality and charge density is added to the system, its structure is attracted to the particles and will neutralize their surface charge so that collision will occur between the suspended particles. The second reaction takes place when the polymeric flocculant bridges among several particles and pulls them together to form a larger, faster settling particle.

The bridging theory was first proposed in the fifty's by Ruehrwein and Ward and modified by La Mer in the sixty's (Henderson et al., 1986). It can be used to explain many phenomena of flocculation, including the superiority of very high molecular weight, hence, long-chain molecules which are capable of more and longer bridges. It also explains the existence of an optimum flocculant dosage: if too much polymer is added to a suspension, all the available adsorption sites on the particles are occupied and there are no free sites for bridging from another particle.

Rate of Flocculation

The bridging theory describes the flocculation principle, but cannot dictate the reaction rate, hence, an ideal flocculation rate is discussed as follows. The process of flocculation is a two step reaction: a lower surface charge followed by a collision event. Assuming that electrical repulsion is absent, as a result of pretreatment with a flocculant, two flocculation conditions may occur: one with no velocity gradient applied to the suspension where the rate of flocculation mainly depends on Brownian motion of the particles as in the case of perikinetic flocculation; in the other a velocity gradient is imparted to a dispersion by stirring, where orthokinetic flocculation dominates.

Supposing that every contact leads to the adherence of one particle to another, then the most rapid flocculation will occur, and the rate of perikinetic flocculation is given by

$$-\frac{dn}{dt} = \kappa_p n^2 \quad (21)$$

in which n is the number of particles present in a given volume and κ_p is a rate constant. Defining the half life for

particles, $t_{1/2}$, as the time required for the initial number of particles to be reduced to one half then it follows that

$$t_{1/2} = \frac{1}{\kappa_p n_0} \quad (22)$$

since the rate constant is related to the diffusion constant,

$$t_{1/2} = \frac{1}{8\pi D n_0} \quad (23)$$

where D is a function of the effective radius of the particles, r , and n_0 , the number of particles of radius r_0 initially present. Taking the diffusion constant as

$$D = \frac{kT}{6\pi\eta r_0} \quad (24)$$

where η is the viscosity of the liquid bulk phase, then

$$t_{1/2} = \frac{3\eta}{4kTn_0} \quad (25)$$

From the above equation, the higher the particle concentration in a suspension, the faster the flocculation will be.

The rate constant κ_p is also given by

$$\kappa_p = \frac{4kT}{3\eta} \quad (26)$$

which demonstrates the effect of thermal energy kT as reflected in the Brownian motion of the particles.

Brownian motion alone is unlikely to produce an aggregate of an acceptable size (1 mm) which leads to useful flocs in a reasonable time. Also when the aggregates of many particles are formed, the number of particles in the suspension decreases which results in a very slow flocculation reaction. If there is an energy imparted to the host fluid this may increase the probability of particle collision, so that the reduced particle concentration is partially offset. Therefore, orthokinetic flocculation consequently occurs. Again, suppose every collision results in an aggregate, then equation (21) becomes

$$-\frac{dn}{dt} = \frac{2}{3} G d^3 n^2 \quad (27)$$

in which d is the diameter of the particle and G is the shear rate.

Optimum Flocculation Conditions

The best flocculation conditions are those which rapidly form large separable flocs, leaving no residual primary particles or small flocs. One way to enhance flocculation is utilizing high velocity gradients. However, this is not

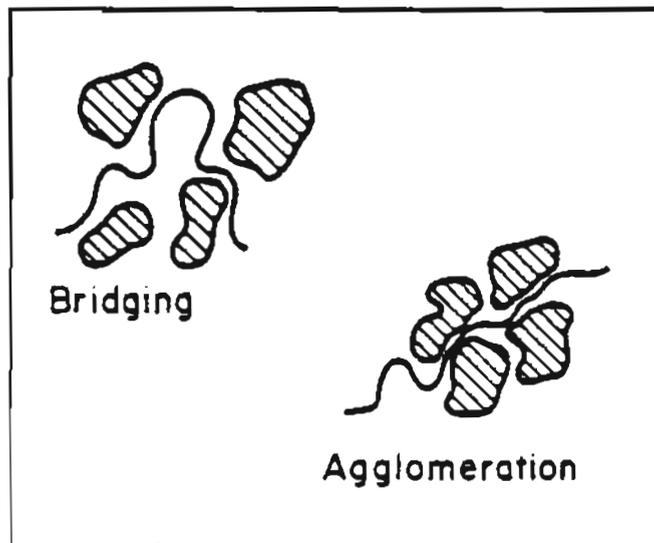
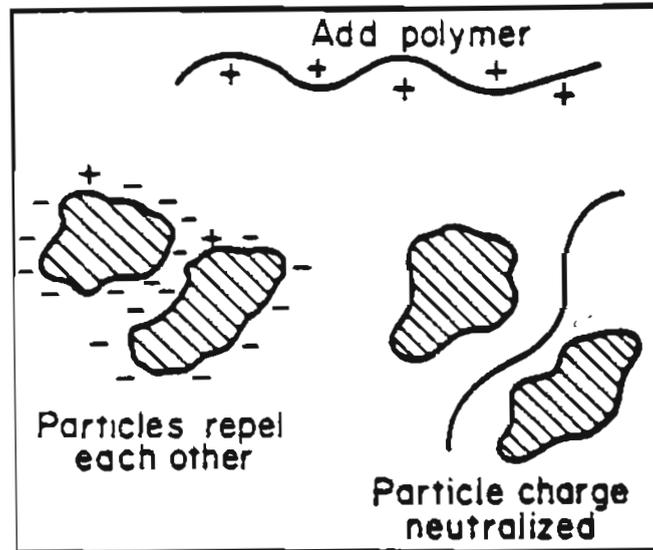


Figure 6. The Mechanism of Polymeric Flocculation (Ackman et al., 1984).

feasible due to the high shear induced from velocity gradient tending to break up flocs. Therefore, a balance has to be achieved between velocity gradients G and the time of flocculation t , one compensating for the other. The dimensionless product Gt derived from above equation, or termed 'Camp' number, can be used to characterize flocculation, and from observation the optimum value of Gt is set between 10^4 and 10^5 (Ives, 1981).

Chapter 4: Mine Operation and Field Test

1: General Description

The field tests for this study were conducted at GHD Resource Inc.'s placer gold operation which is located on Eagle Creek, in the Circle Mining District of Alaska, as shown on the location map designated as Figure 7.

Figure 8 indicates the basic layout of this open cut mining operation in which a total of three bulldozers were employed for stripping the overburden, transporting the paydirt and handling the tailings. Water was diverted from the nearby creek to a water supply pond, where it was pumped to the process plant. The effluent from the process plant flowed, via a tailrace, into settling ponds or under certain conditions, to a water supply pond for recycle. The processing capacity of this mine was 3,000 yds³/day and its water usage as estimated by the mine manager, was 2,000 gpm.

In the processing plant, shown in Figure 9, a pan feeder and conveyor belt were used to transfer the paydirt to a double-deck vibrating screen. The 2 inch and 5/8 inch screening surfaces were supplied with water sprays for washing purposes. The oversize of these two screen surfaces discharged to the coarse tailing conveyor, and minus 5/8 inch material reported to a sluice feed box located directly beneath the screen. Subsequent to gold recovery in the sluice box, the tailings are discharged to a water treatment system which consists of a sand screw and two hydrocyclones for removal of the settleable solids. The mine included this system primarily to recycle process water during the latter part of the mining season because of water shortage and to increase settling pond life.

2: Sand Screw-Hydrocyclone Water Treatment System

A balanced flowsheet of the sand screw-hydrocyclone (SS-H) system is shown in Figure 10. Tailings from the sluice box at 2,600 gpm with a pulp density of 17.4% are discharged to the sand screw. Sand screw underflow, at 97 TPH solids with 13% moisture, are deposited on a conveyor system. Sand screw overflow consisting of non-settled solids and water (9.7% solids) is pumped from a sump to two

20 inch hydrocyclones at a rate of 1,635 gpm. The underflow of the hydrocyclones is discharged onto coarse tailings being conveyed up by the sand screw. The 1,580 gpm overflow from the hydrocyclones, 3.6% solid, becomes the effluent of the processing plant. The specifications for the equipment used in this system are:

Sand Screw: Eagle Iron Works (EIW), Spiral Dewaterers, 25' X 36".

Hydrocyclone: Krebs Cyclones Model D20B-869.

Pump: 6X8 Series FHSA 19.5 Thomas Simplicity Slurry Pump.

Sump: Steel construction, 12' X 4 1/2' X 3'

The advantages of using this water treatment system are: (1) lower maintenance requirements, (2) this system functions as part of a continuous tailings handling process and (3) increased settling pond life. However, the purchase of equipment and extra power needed to operate this system may cause an increase in operational costs.

3: Testing and Sampling

The tests conducted in this study can be divided into two phases. In the first phase emphasis was on determining the performance of the SS-H system. In the second phase tests were focused on the determining the effectiveness of gelsols in reducing the turbidity of the mine effluent.

In the first phase, the mining operation and SS-H system were observed and tested under various modes of operation: (1) normal operation - only fresh water was used in the process plant, (2) recycle operation - process water was recycled at two levels 50% and 100%. Additional tests of smaller size hydrocyclones, 10 inch and 4 inch diameter, were also performed to see their effect on solids removal.

For tests of the SS-H system with 20 inch hydrocyclones, the operational parameters were varied by changing the hydrocyclone apex sizes and vortex finder sizes. The apexes available for testing were 2 1/2 inch and 3 1/2 inch. Vortex finders provided were 6 inch and 7 inch. The plant flow rates were measured with a Controlotron Series 180 Doppler flowmeter (Appendix 3), however, hydrocyclone underflow rate was measured using a bucket and a stop watch. Samples were collected from the SS-H system at points indicated by stars in Figure 9. Generally, each test was run over a 10 hour work shift, with samples being collected at different time intervals. During each test, pulp density was checked regularly with a Marcy's pulp density scale. The detailed testing and sampling procedure for each testing condition is described below.

Normal Operation

Normal operation refers to that period when only fresh water was used for processing. It served as the base line for

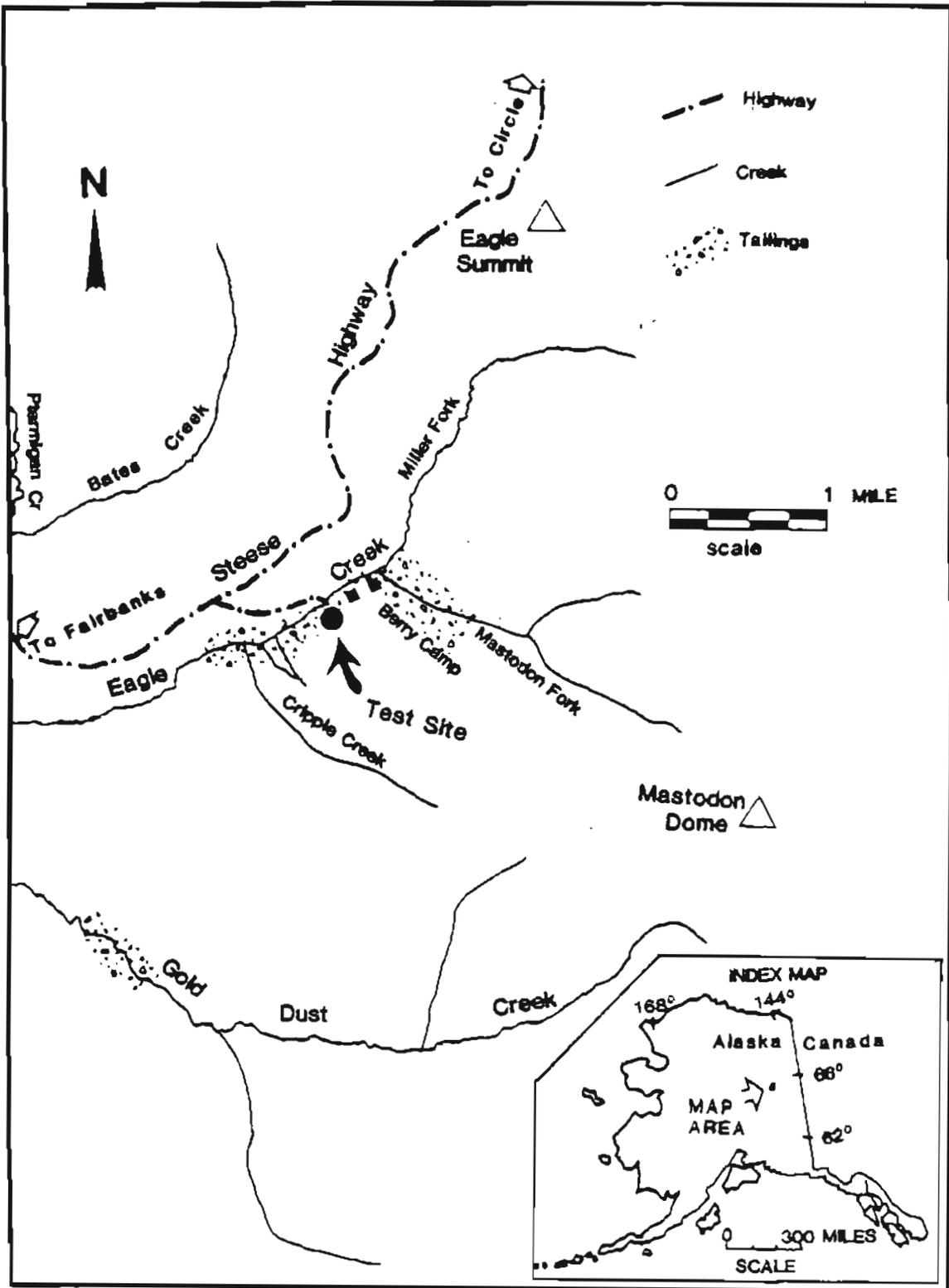


Figure 7. Location of GHD Inc.'s Mine.

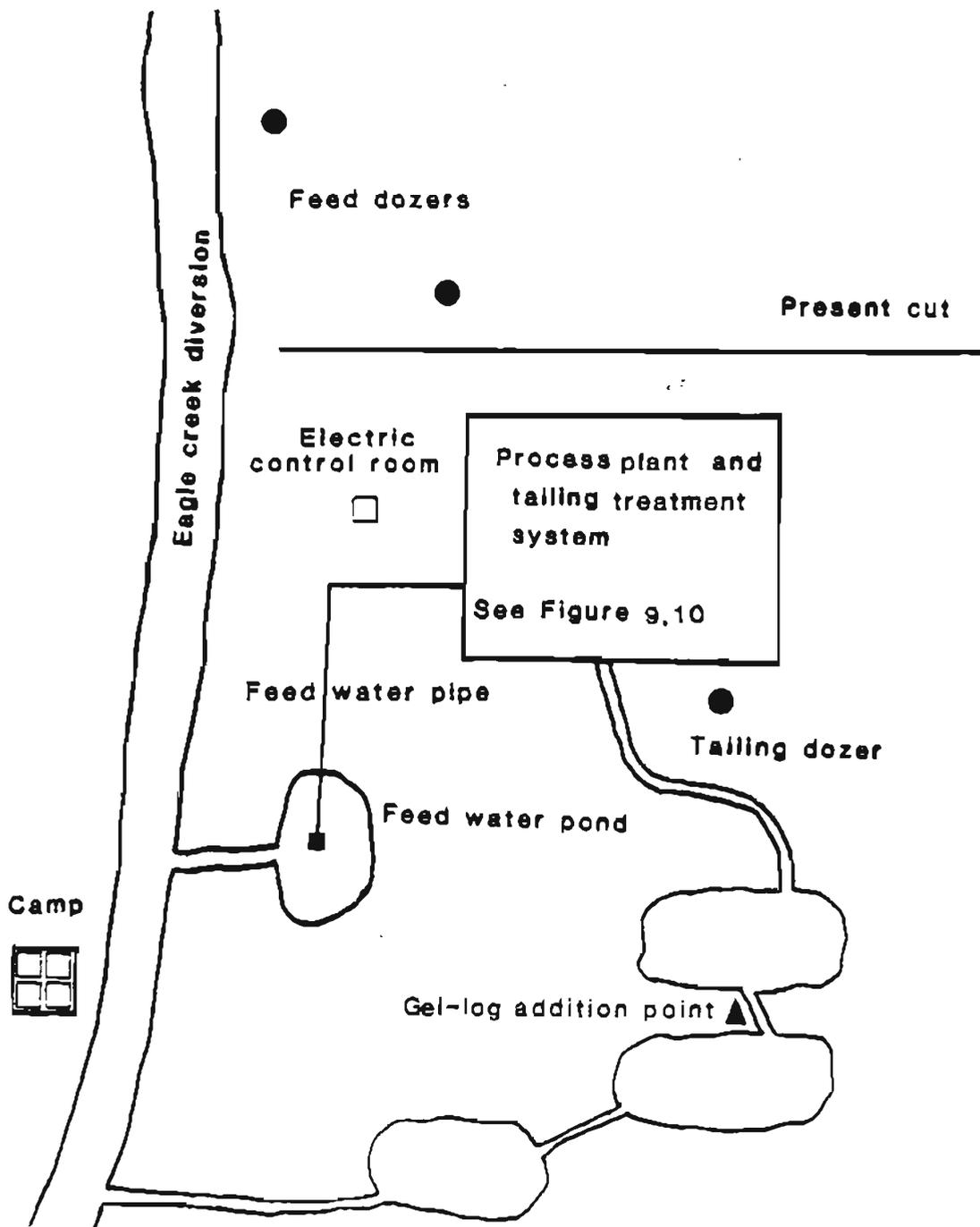


Figure 8. Plant Layout of Mining Operation.

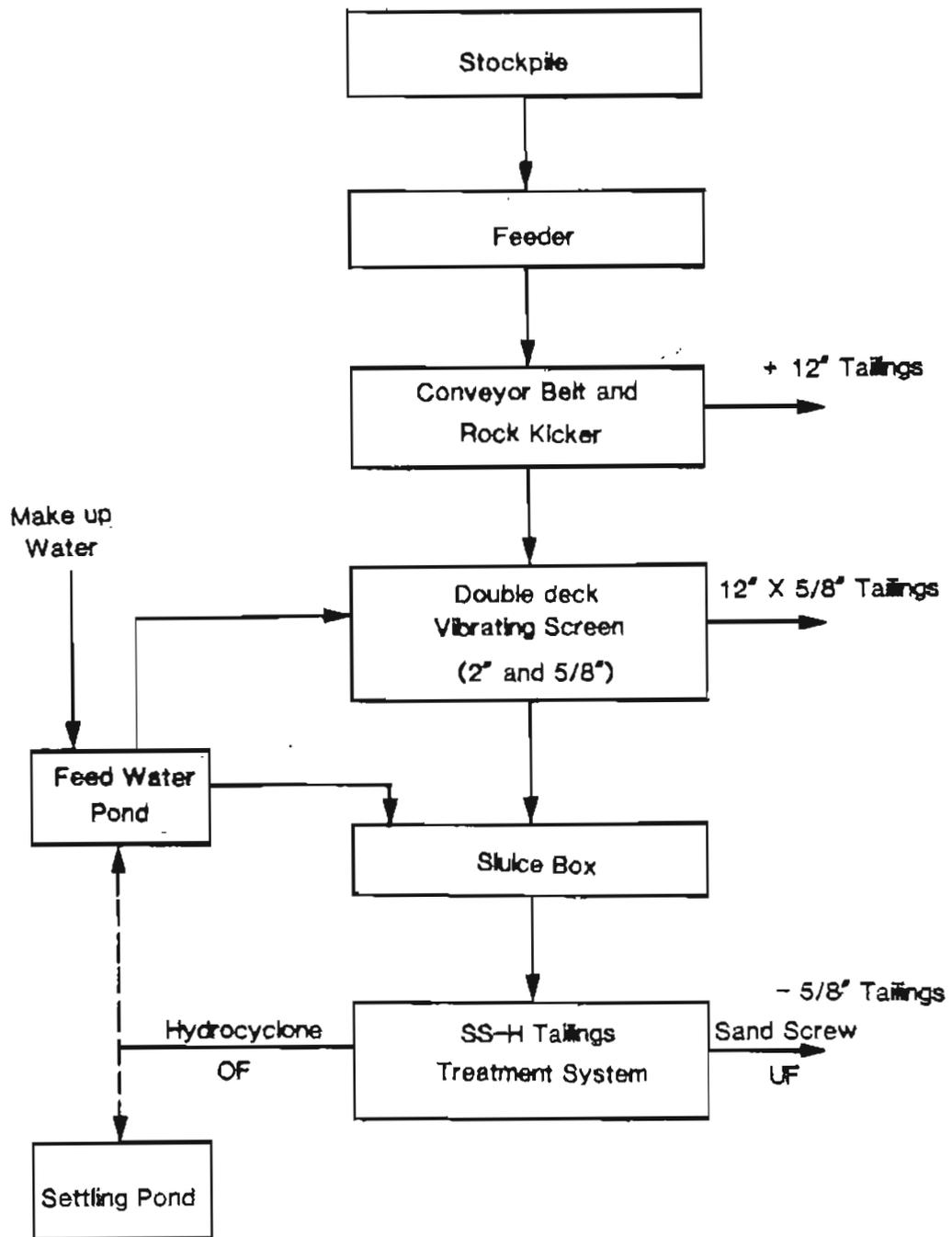


Figure 9. Block Diagram of Process Plant.

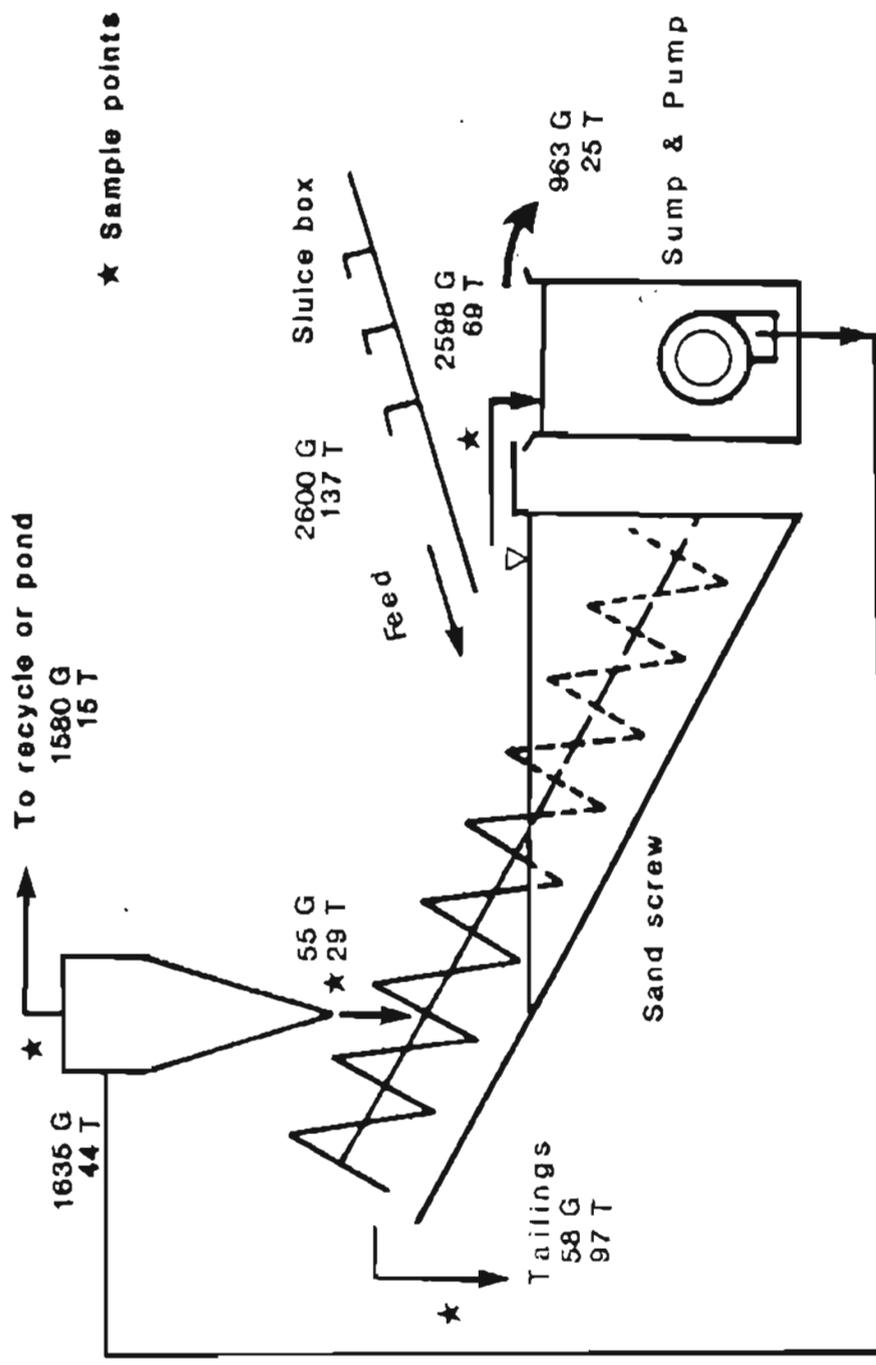


Figure 10. Sand Screw-Hydrocyclone Water Treatment System.

Not to scale

comparison of periods of recycle operation. In order to understand the performance of the sand screw alone, samples were taken from the sand screw underflow and overflow prior to operating the hydrocyclone. After hydrocyclone operation (3 1/2 inch apex) was initiated, samples were collected at 5, 30, and 60 minute intervals with the intention of studying the solids build up within the system. Normal operation was also studied using a 2 1/2 inch and a 2 1/4 inch apex, the former associated with a 6 inch vortex finder and the latter run with a 7 inch vortex finder. During this operation, samples were collected while hydrocyclone underflow was discharged onto the ground instead of onto the coarse solids within the sand screw. The hydrocyclone operating pressure was kept steady at 15 psi.

Recycle Operation

Recycle operation refers to that period when the hydrocyclone overflow was directed in part or in whole back to the water supply pond, mixed with make up water, and pumped back to the processing plant. During this operation, hydrocyclone pressure was 15 psi and the vortex finder size remained 6. Only apex sizes were changed. A total of 7 samples were collected and analyzed; a description of these sample is listed in Table 2.

Table 2. Samples Taken During Recycle Operation

Degree of Recycle	Apex Size (inch)	Hours into Recycle
50%	2 1/2	1,3,7
	3 1/2	9
100%	3 1/2	1,3
	2 1/2	2

Five process water samples were also collected during the recycle operation.

Additional tests were conducted with smaller diameter hydrocyclones to investigate their fine solids removal efficiency. A split from the main 20 inch hydrocyclones feed line was diverted to feed the smaller cyclone. A ball valve was used to regulate the flow to the smaller hydrocyclones and functioned as pressure controller. A 10 inch Krebs plastic hydrocyclone and a 4 inch Cyclone Engineering Sales Ltd. model were tested at 10, 15, and 20 psi. The pressure gauge was located at the hydrocyclone feed inlet.

In the second phase, a gel-log testing program was conducted. The logs were first wrapped with chicken wire and fastened in a 20 foot long flume situated at the overflow from the first pond. The flocculant dosage was controlled by adding logs to the flume and the water flow rate was regulated by moving the flume to receive all or part of the pond overflow. A maximum of 14 logs were used during the test.

Because the logs were unable to produce the desired performance during the test period, another test was conducted by dissolving a known amount of gel-log in water. This solution was then pumped at a constant rate to the flume and mixed with a controlled flow of settling pond overflow.

An additional test was conducted by adding a flocculant solution to the SS-H system with the intention of enhancing the solids removal efficiency. The flocculant used for this test was polyethylene oxide (PEO).

4: Sample Analysis

System samples were collected in 5 gallon plastic buckets, sealed, and brought back to the laboratory for analysis. Samples were analyzed for their pulp densities or percent moisture by filtration and/or drying procedures. Size distributions of plus 400 mesh solids were determined by ASTM sieve analysis. Minus 400 mesh particle size distributions were determined using a SediGraph 5000ET particle size analyzer (Appendix 2).

Water properties examined included: (1) settleable solids, measured with an Imhoff cone; (2) viscosity of the recycled process water, measured with a Fann viscometer (model 35A), (3) turbidity, measured with a HF Instruments Inc. DRT-100 turbidimeter, and (4) the elements present in the pond water were determined by a Beckman SpectraSpan V Direct Current Plasma spectrometry.

Clay minerals from the slurry were identified by using the ceramic tile method (Appendix 1), with a Rigaku x-ray diffractometer system model Geigerflex D/MAX IIA.

Chapter 5: Results and Discussions

The results of this study are presented in three sections. First, the performance of the SS-H system is presented as size distribution curves for the various operational conditions. Second, properties of the plant effluent and process water which include solids concentration, settleable solids, viscosity and turbidity are considered along with the clay mineralogy of the suspended solids. Third, the flocculation tests using gel-logs and PEO, in association with settling ponds and the SS-H system are discussed.

1: Performance of the SS-H System

The sluice box discharge served as feed to the SS-H system. The size distribution of the discharge is shown in Figure 11. This size analysis does not include the plus 5/8 inch material which was discarded as coarse tailings prior to sluicing. The minus 5/8 inch sluice tailings contained 12% minus 38µm (400 mesh) material and had a pulp density of approximately 18% (w/w).

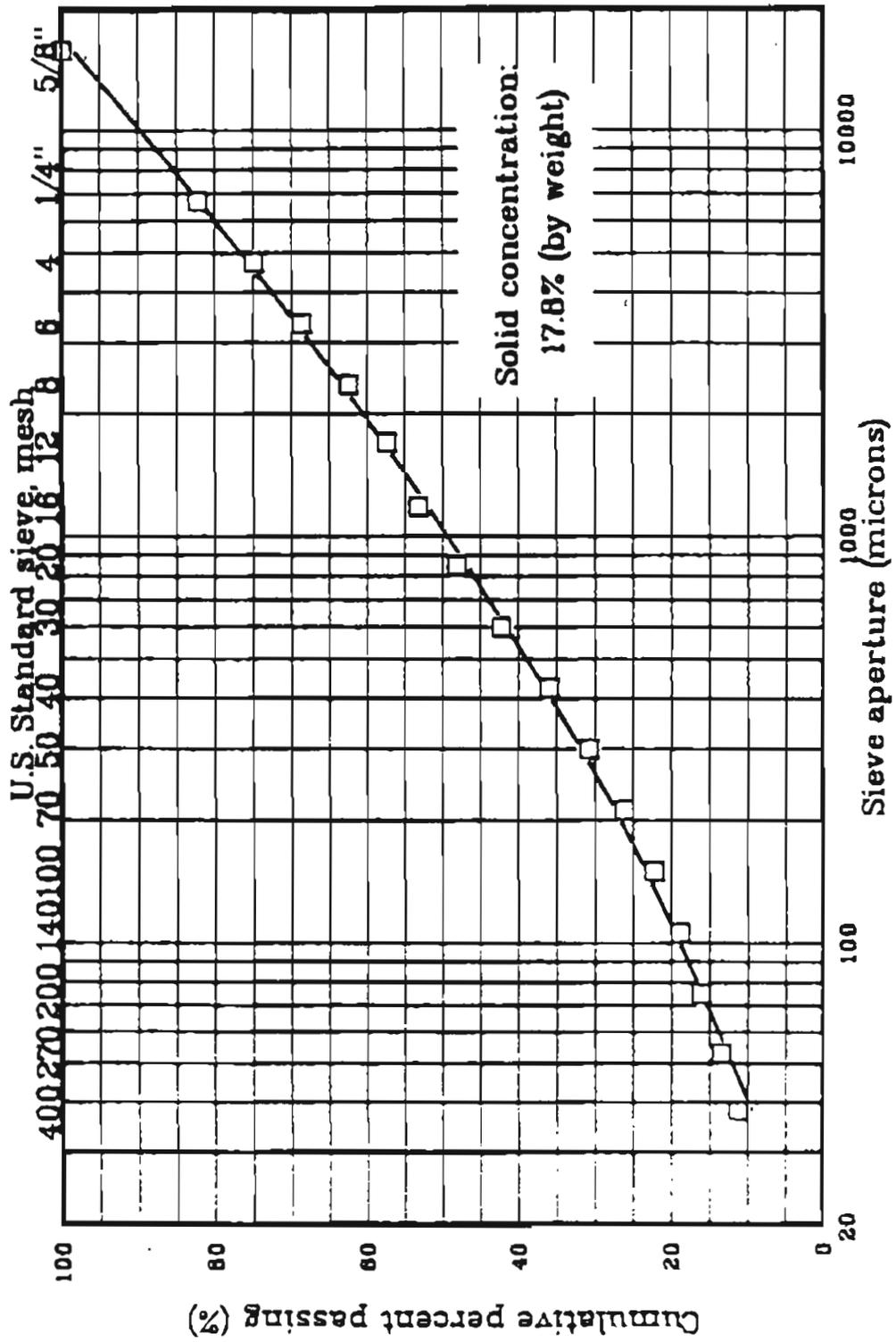


Figure 11. Size Distribution of Sludge Box Discharge.

Under normal operating conditions, the sand screw underflow contained 18-20% moisture, while the sand screw overflow had a pulp density of 16%. The two 20 inch Krebs hydrocyclones operated at 12 psig producing an overflow carrying 5% solids, and an underflow with a pulp density of 59-62%. The pulp underflow rate of the two hydrocyclones was 150 gpm with 2 1/2 inch apexes and 220 gpm with 3 1/2 inch apexes.

Figure 12 presents information on how the SS-H system dynamics changed over time. One set of curves shows the performance of operating only the sand screw, where sand screw overflow would flow to a settling pond. Initiation of hydrocyclone operation was followed by rapid equilibration of the system as shown by the size distributions of samples collected at 5, 30, 60 minutes after start up. With sand screw operation alone, the pulp density of the overflow of the sand screw was at 8% compared to the 16% maintained in conjunction with hydrocyclone operation. The solids concentration build up is due to the incomplete filtering action of the coarse sand screw underflow solids upon which the hydrocyclone underflow discharged. The upshift of the size distribution curves of the sand screw underflow is a measure of the success of coarse solids filtration.

As shown in Figure 13, different apex sizes and vortex sizes resulted in slight size distribution changes. The hydrocyclone underflow pulp density increased from 59% to 62% and to 67%, as the apex size decreased from 3 1/2 inch to 2 1/2 inch and to 1 1/2 inch, respectively. However, with 1 1/2 inch apex in operation near 'roping' discharge conditions were observed. The effect of different vortex finder sizes on particle size distribution is shown in Figure 14. A 7 inch vortex finder operated at 10 psig produced an underflow rate of 120 gpm with a pulp density of 60% and an overflow pulp density of 4%.

Figure 15 contrasts an open vs a closed loop operation. In the open loop operation the hydrocyclone underflow was discharged to the ground rather than into the sand screw underflow solids. As expected, an open loop operation reduced sand screw overflow pulp density to 8%. It also reduced the hydrocyclone underflow pulp density to 27% from the 59% observed in a closed loop operation. The pulp densities of the sand screw underflow and hydrocyclone overflow remained unchanged.

Recycle operation had an obvious influence on the performance of the SS-H system. During this period the hydrocyclones were operated at a higher pressure, 15 psig, rather than the normal 12 psig operation pressure. Consequently, a higher hydrocyclone underflow rate of 220-250 gpm was realized. The two underflow streams maintained about the same solid content of 82-85% for the sand screw and 63% for the hydrocyclones. However, the overflows

suffered from fine particle build up resulting in the sand screw overflow pulp density reaching 30-40%. Hydrocyclone overflow pulp density initially increased to 10% during partial recycle then soared to 30% under total recycle conditions.

Figures 16 and 17 indicate that apex size has some effect on the system performance during recycle operation, as compared with normal operation. In Figure 18 which compares the degree of recycle, a significant shift in the two overflow streams solids distributions can be seen.

A general rule in mineral processing is that the smaller the hydrocyclone the finer the cut size. A 10 inch and a 4 inch cyclone were tested to examine the cyclone diameter effect on cut points. This series of tests were run at cyclone feed pressures of 15, 10, and 5 psig, however, when these smaller cyclones were operated at a lower pressure of <10 psig roping discharge occurred. The 10 inch cyclone tests showed an underflow pulp density of 70%, and an overflow pulp density of 5% with a flow rate for underflow and overflow at 15 gpm and 355 gpm respectively. The 4 inch cyclone produced an underflow pulp density of 60-65% and a 4% solids overflow, with a flow rate of 51 gpm for the overflow and 4 gpm for the underflow. Partition curves of these three cyclones are presented in Figure 19. Cut sizes shown are 50, 43, and 38 μm for 20, 10 and 4 inch cyclones respectively. The size distribution of these products of the three cyclones are compared in Figure 20.

The cut sizes predicted from Plitt's cyclone model are: 48.6 and 64.4 microns for 20 inch cyclone equipped with 3 1/2 inch and 2 1/2 inch apexes respectively, for the 10 inch cyclone 43.4 microns, and 46.8 microns for the 4 inch cyclone.

2: Water Properties

The process water qualities are altered after the sluicing process primarily due to the solids content increase, especially in the non-settleable solids fraction. An understanding of these water quality changes at various operational conditions, the bulk properties of solids suspension and the characteristics of the solids are important for considering water treatment methods. The temperature of the settling pond water measured was 6-9°C and the pH was about 7.

The effect of different modes of SS-H system operation, such as various degrees of recycle, on the water quality (e.g. settleable solids, pulp density and turbidity) are summarized in Table 3.

Table 3 shows that the level of settleable solids in a normal operation remained near 50 ml/l, but built up to higher levels under a recycle operation. Turbidity readings also increased as degree of recycle increased. These

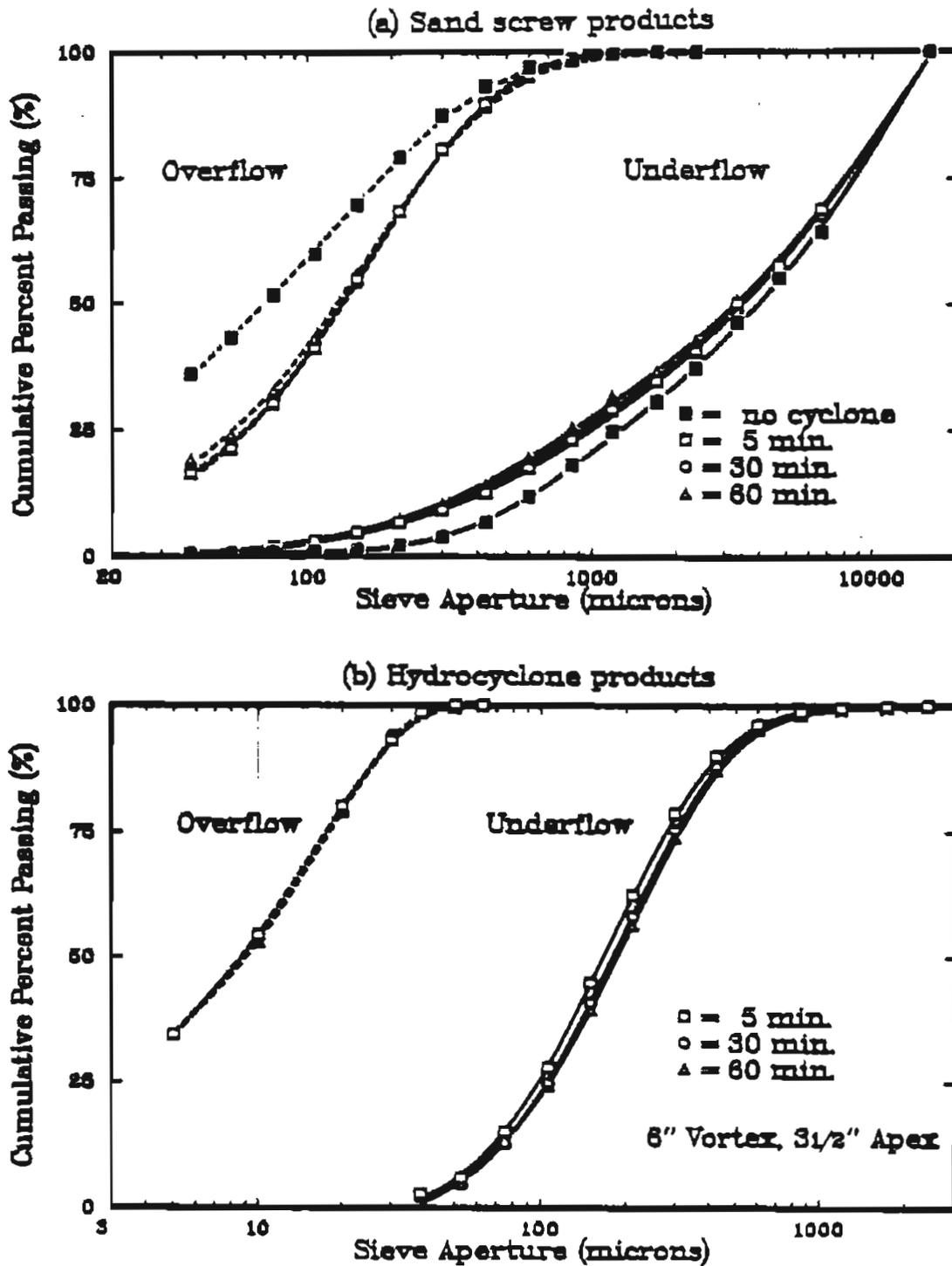


Figure 12. Size Distribution at Various Time into SS-H System Operation.

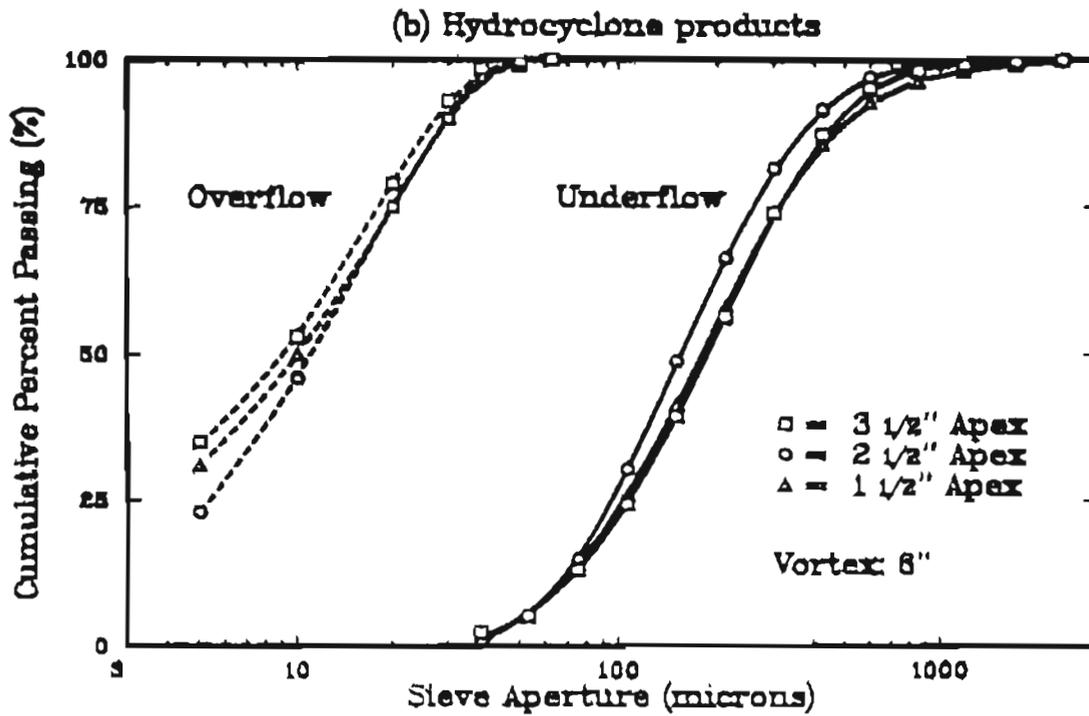
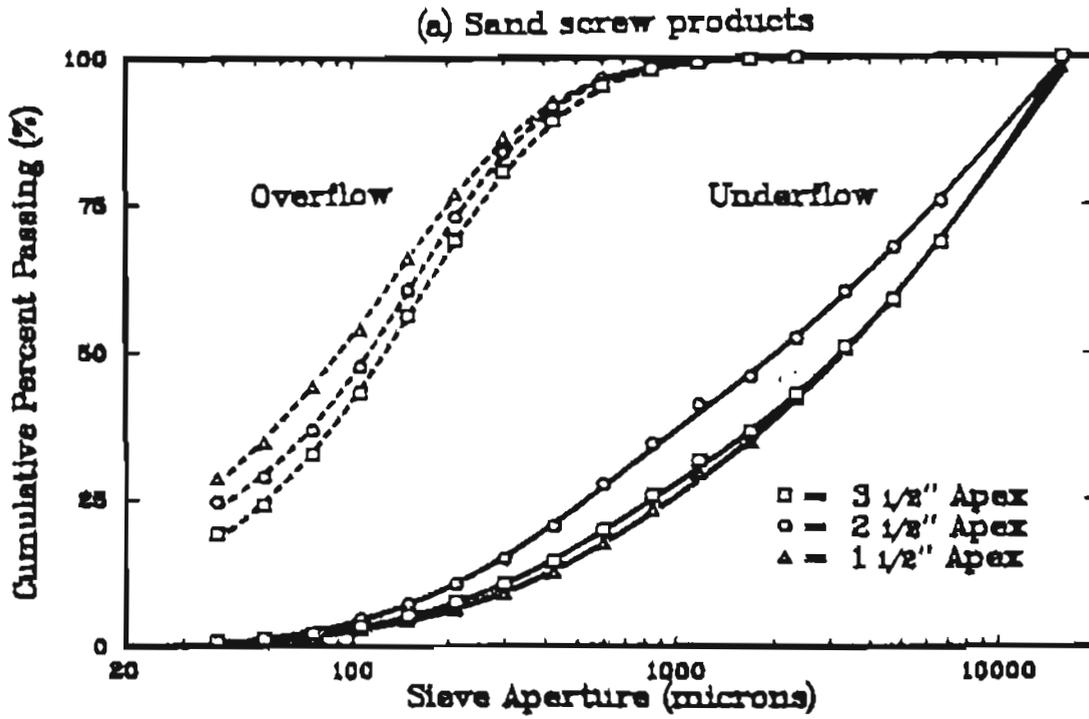


Figure 13. Size Distribution at Various Apex Diameters.

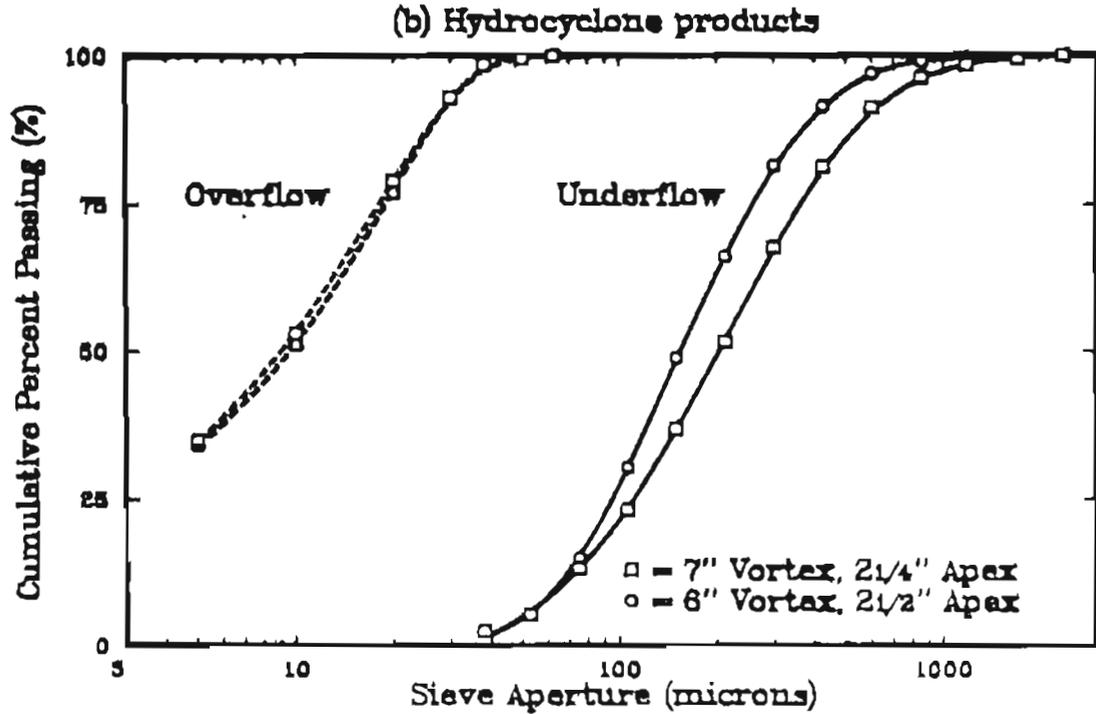
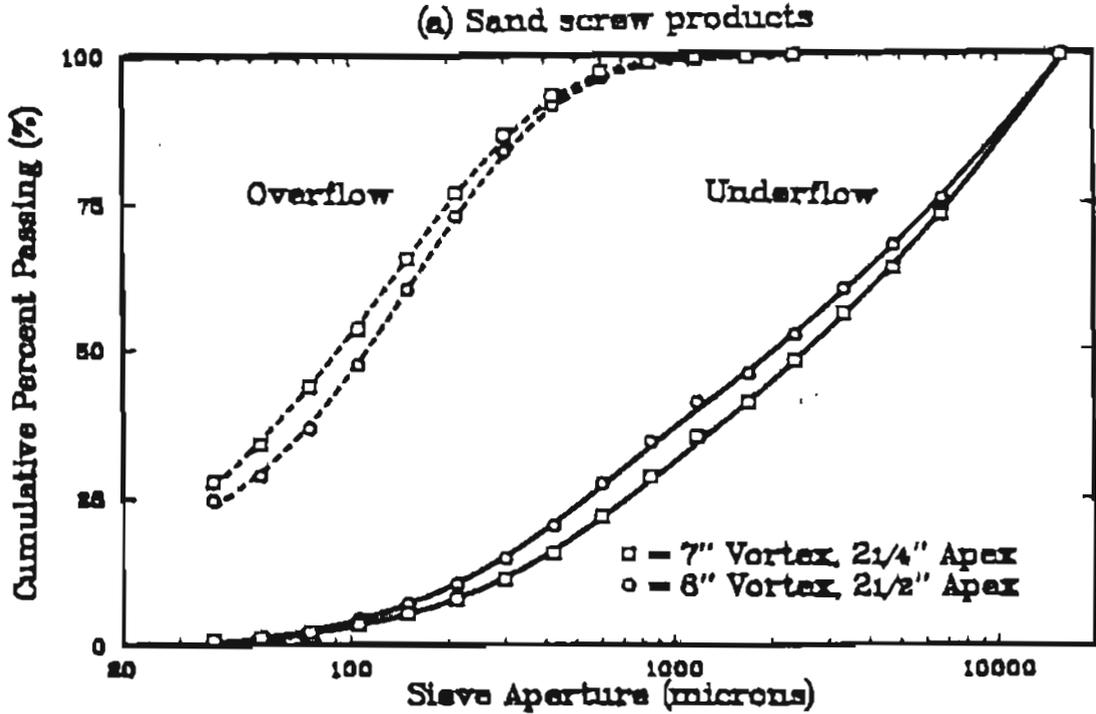


Figure 14. Size Distribution at Various Vortex Finder Diameters.

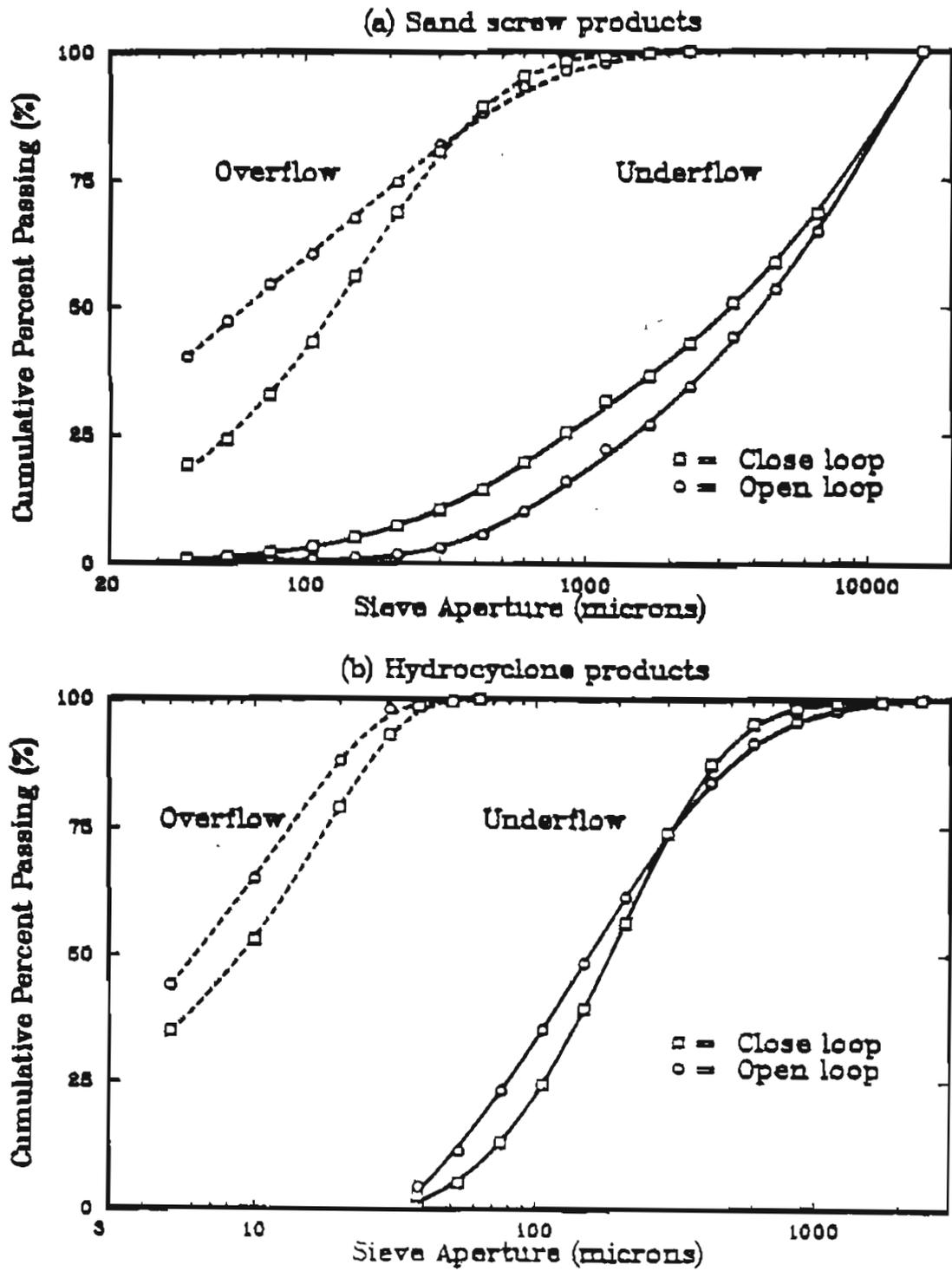


Figure 15. Size Distribution at Various Operational Modes.

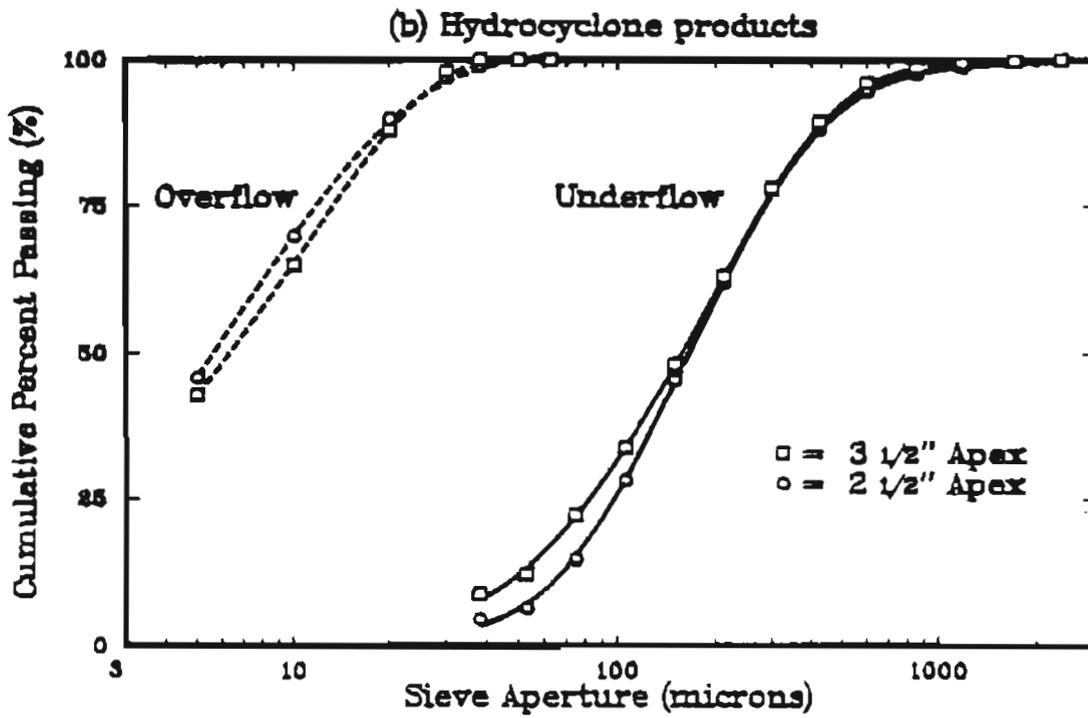
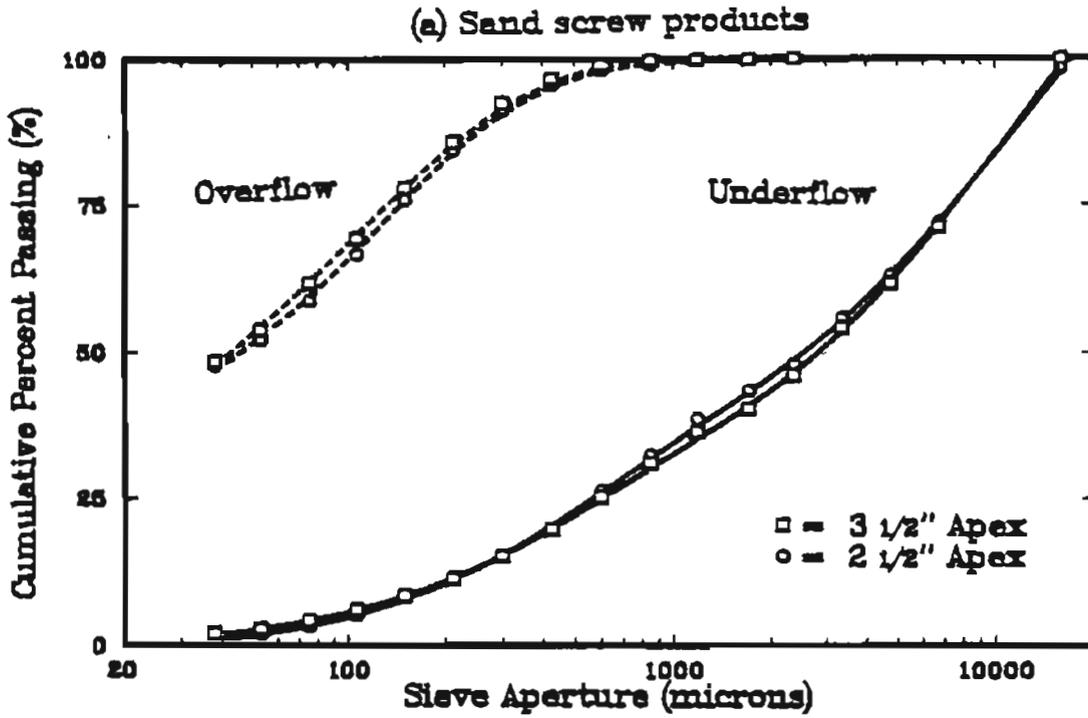


Figure 16. Size Distribution at Partial Recycle.

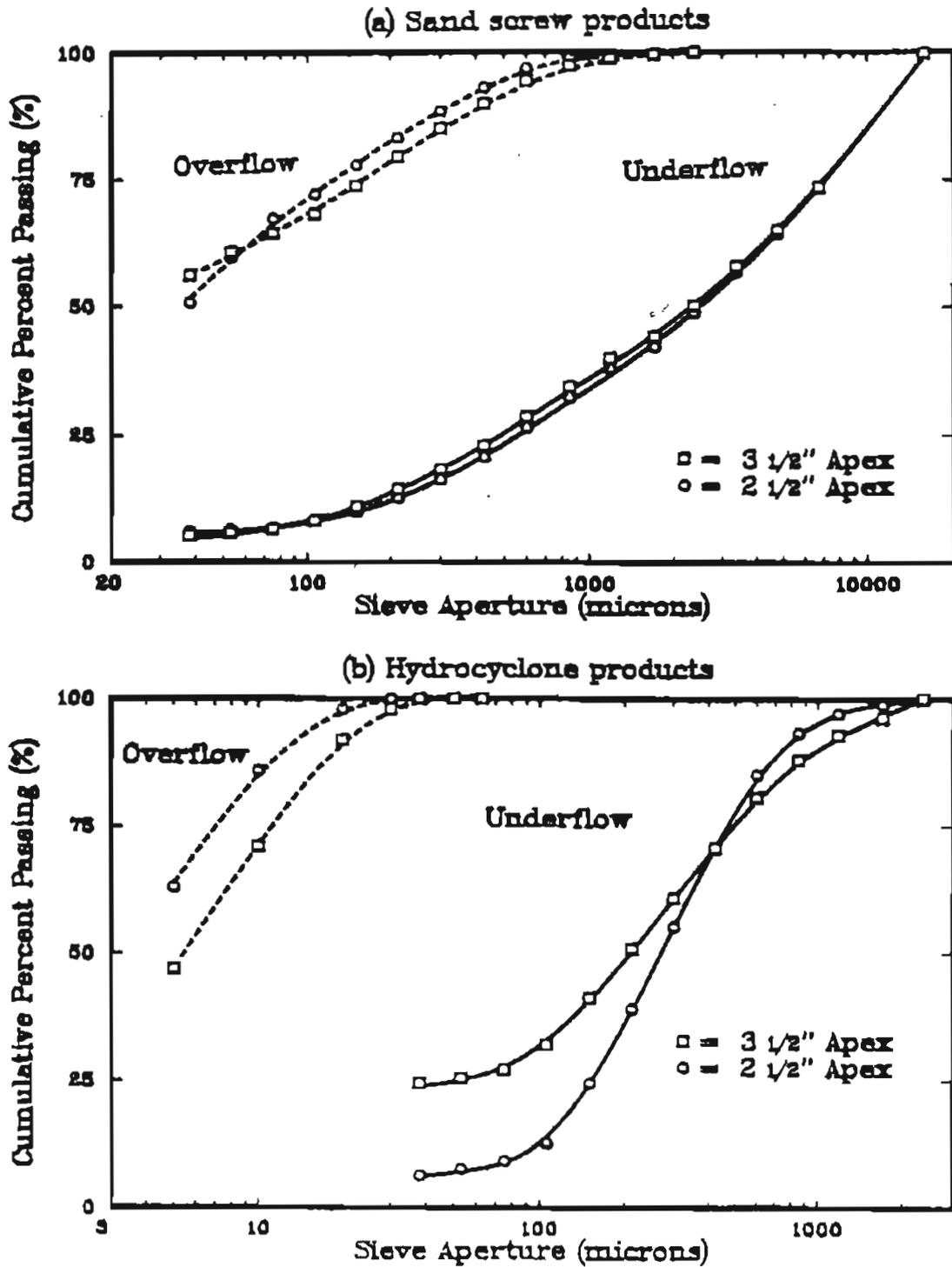


Figure 17. Size Distribution at Total Recycle.

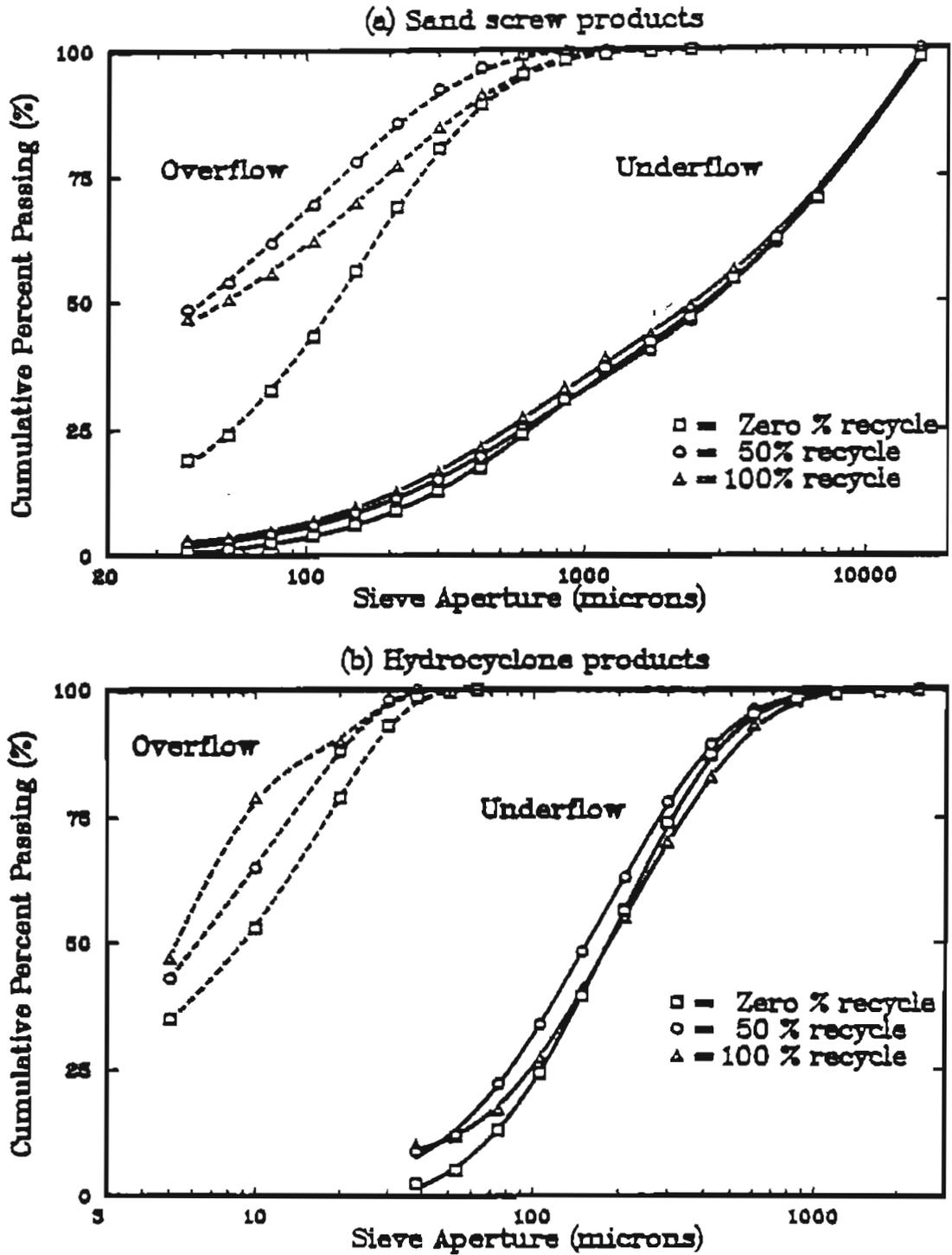


Figure 18. Size Distribution at Various Recycle Mode.

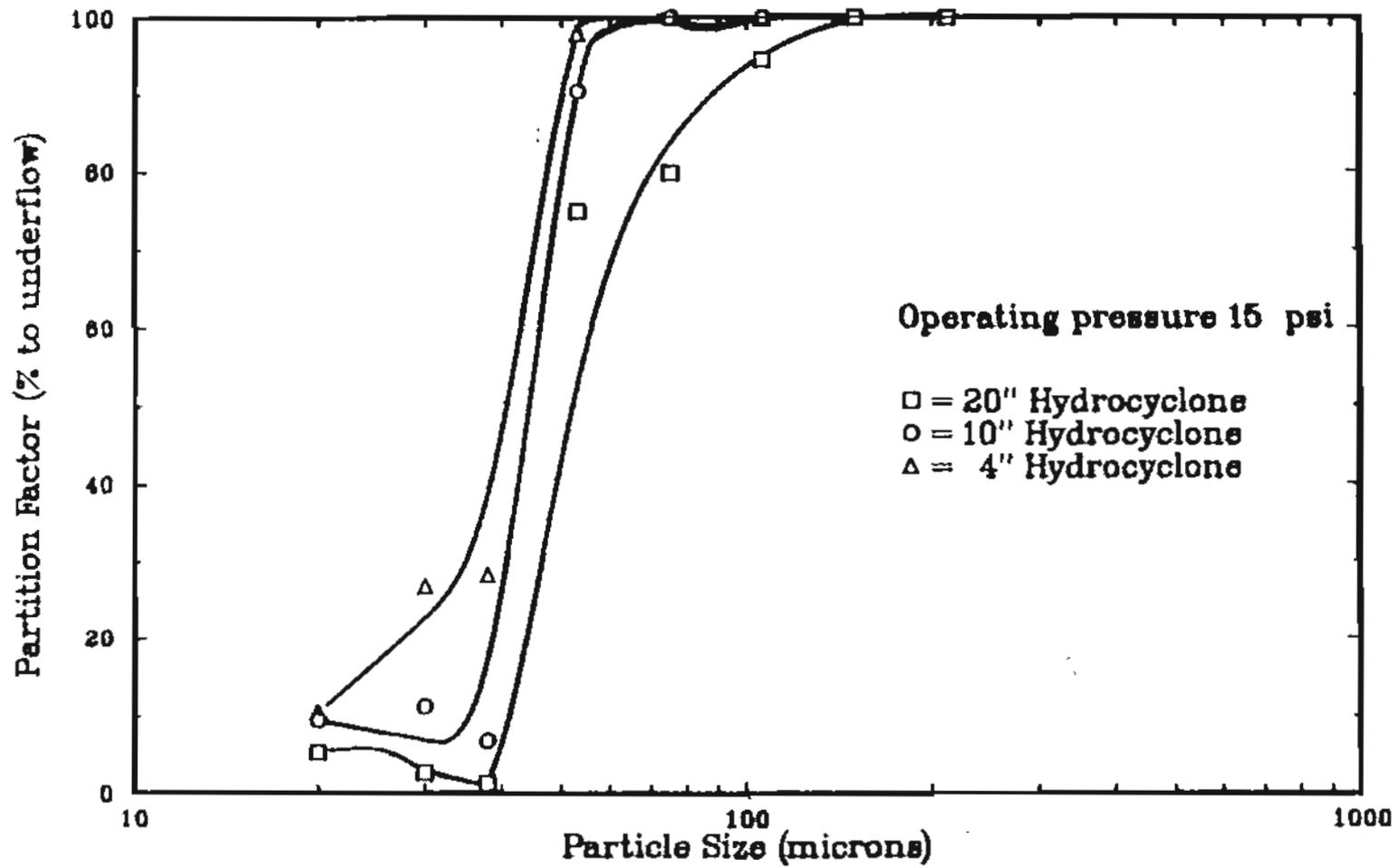


Figure 19. Partition curves of Various Diameter Hydrocyclones.

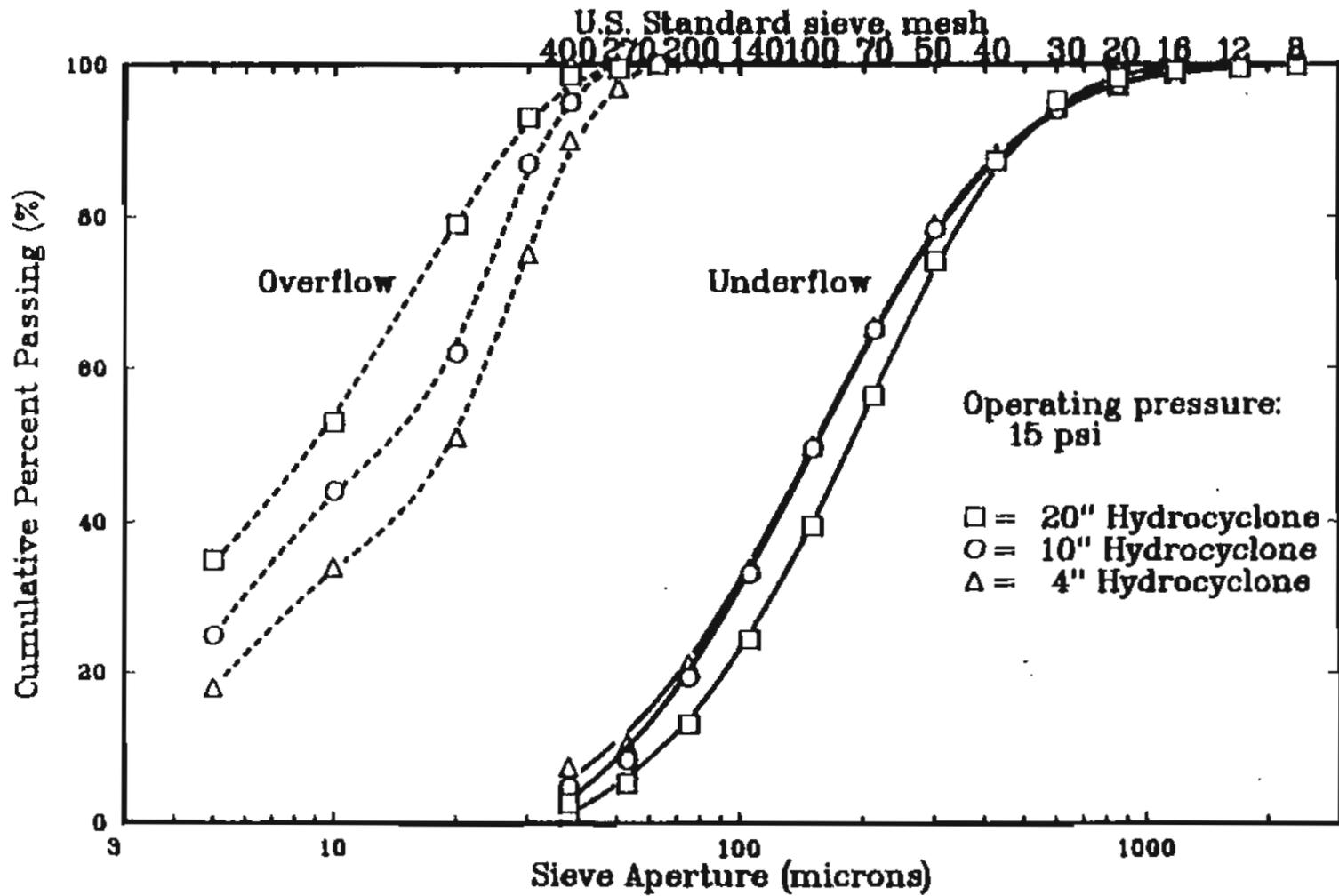


Figure 20. Size Distribution for Product of Various Hydrocyclone.

Table 3. SS-H System Effluent Properties

Operation Mode	Apex Sizes (inch)	Time into Operation (hour)	Settleable Solids (ml/l)	Turbidity (NTU)	Pulp Density %(w/w)
Normal operation	2 1/2	1	55	—	6
	3 1/2	1	47	6,000	5
50 % recycle	2 1/2	1	62	—	6
		3	80	60,000	9
		7	80	—	11
	3 1/2	9	90	58,000	11
100% recycle	2 1/2	2	175	92,000	29
	3 1/2	1	200	—	17
		3	210	72,000	28

measures appear to show some correlation to the apex size employed, as did the cyclone products size distributions.

Table 4 shows settleable solids contained in the plant process feed water vs recycle condition. These values, when compared with those of Table 3, show that approximately 40% of the settleable solids in the SS-H system effluent settle out in the tailrace and feed water pond prior to recycle.

Table 4. Settleable Solids of Recycled Process Water

Sample	Apex (inch)	Settleable Solids (ml/l)
50% recycle	3 1/2	57
	2 1/2	37
100% recycle	3 1/2	125
	2 1/2	155

The bulk suspension properties which affect settling velocity are density and viscosity. Density is related to the pulp density as:

$$\rho_p = \frac{100}{C_s/\rho_s + (100 - C_s)/\rho_l} \quad (28)$$

where ρ_p is the density of the suspension, C_s is the solids concentration, ρ_s and ρ_l are the density of the solids and liquid respectively.

The viscosity of the process water at various degrees of recycle was measured using a Fann viscometer (Table 5).

The results shown in Table 5 indicate that the viscosity of the process water increased as the degree of recycle increased. The influence of clay-water suspension on the settling velocity of gold particles has been indicated by

Walsh (1986), as suspension viscosity and density increase, the settling velocities of gold particles are reduced.

The suspended solids in the pond water ranged in size from a tenth of a micron to a few microns, the larger particles having settled in the tailrace. The fine clay particles do not settle readily, and this resulted in the water quality problems. The clay minerals contained in the mine effluent were separated and analyzed to identify their mineralogy. The results showed kaolinite, mica, and chlorite are the three major clay minerals in this mine. A general characteristic of these clays is that they are of the non-expanding type. As indicated by Walsh, 1986 and Peterson (1984), kaolinite has less effect on the settling velocity of gold particles and sluice box riffle packing than expandable clays, such as montmorillonite.

Henderson et al. (1987), indicated that some dissolved elements, such as Al, and Fe, tend to inactivate flocculants. A water sample was collected from the settling pond and filtered to remove the suspended solids. Concentrated HNO₃ was then added to keep the sample pH less than 2 and ready for analysis (American Public Health Association et al., 1981). The results of this analysis by the DC Plasma spectrometry, are shown in Table 6.

3: Flocculation Test

As with conventional flocculants, gel-logs are supplied by manufacturers in, non-ionic, cationic, or anionic forms. The effectiveness of each type for various applications must be determined by a series of tests. Consequently, a total of eight gel-log samples were tested in this study. The test procedures consisted of allowing selected log samples to soak in beakers containing a suitable amount of water for hydration purposes, after which the chunks of hydrated logs was shaken with approximately 500 ml water sample taken

Table 5. Viscosities of Plant Process Water

Sample	Temperature °C	Fann Viscometer (rpm)	Apparent viscosity (cp)	
Tap Water	4	600	1.6	
		300	1.7	
		200	1.5	
		100	1.5	
50 % recycle	5	600	1.9	
		300	2.0	
		200	1.5	
		100	1.5	
	7	600	2.0	
		300	2.0	
		200	1.5	
		100	1.5	
	100\% recycle	5	600	3.5
			300	3.5
			200	3.0
			100	3.0
7		600	4.0	
		300	4.0	
		200	3.8	
		100	3.0	

Table 6. Dissolved Elements in Settling Pond Water

Element	Al	Si	Fe	Ca	Mg	Mn	Ti	K
µg/ml	0.13	<1	<1	17.6	3.3	0.4	<0.1	1

from the mine. A visual comparison of the results was used to determine the relative effectiveness of each log type. The results are shown in Table 7.

The cationic and non-ionic gel-logs appeared most effective in producing a clear supernatant. Photofloc 1130 was selected for field testing. Although, cationic and non-ionic gel-logs were effective in cleaning the effluent in this test, the possibility of using a combination of different types should be pursued.

Another point worth noting here is that maximum hydration of gel-logs, as suggested by the manufacture, is

an important step in the preparation of gel-log use. A simple test was conducted in the laboratory to examine the weight gain of a gel-log during hydration. This consisted of immersing a fresh Photofloc 1130 gel-log in a drum. The log was weighed every six hour during the first day and once a day for two successive days; the water was changed after every measurement. The results of these tests showed first day weight gains of 2.7%, 4.4%, 6.3%, and 6.8%, a second day weight gain up to 7.7%, and a third day gain up to 9.9%.

Because a gel-log is slowly dissolved in a flowing stream the optimal dosage is on a trial and error basis. Therefore, a field test was conducted in an attempt to

quantitatively determine the gel-log concentration required for GHD's application. A stock solution (0.24%) was obtained by dissolving one pound of Photofloc 1130 gel-log in 50 gallons of water in a 55 gallon drum. This solution was metered into the flume to mix with settling pond overflow water. The gel-log solution was pumped at 10 or 24 l/min and pond water flow rates were controlled by moving the flume, the estimated flow rates were 1,000, 500, 300, and 150 gpm. Table 8 shows the test results:

As indicated, in Table 8, the dosage needed to obtain a 500 NTU level was 0.39 pound per 1000 gallons of effluent.

A test utilizing PEO in association with the SS-H system was also conducted at the field site with the intention of enhancing solid removal. During this test the cyclone was operated with a 7 inch vortex finder and 2 1/4 inch apex; the feed slurry to the cyclones was pumped at 1507 gpm. A 0.07% PEO solution was mixed with make-up water to produce a 0.008% working solution which was sprayed into

the sand screw pool area at a rate of 60 gpm. This equivalent to 0.038 lb/min PEO rate or 0.026 lb PEO per 1,000 gallon slurry. After the addition of PEO solution to the pool in the sand screw floccs were observed immediately. The results of this test showed a marked increase in settleable solids contained in the hydrocyclone overflow, from 73 ml/l before PEO addition to 130 ml/l after PEO addition. Figure 21 shows the hydrocyclone partition curves for conditions before and after PEO was added.

Chapter 6. Cost Analysis

Generally the capital cost of a water treatment system is a major determining factor on the feasibility of the system. This chapter compares the capital and operational costs of the SS-H system studied with the cost of operating the system of ponds employed at the mine in the previous year. Since the gel-logs were only used on a trial basis during the test period, no full scale operational cost information can be obtained. However, a cost estimate can be drawn from the

Table 7. Qualitative Gel-log Test

Photofloc Product	Floc Formation	Settling Rate	Floc Stability	Floc Size	Supernatant Quality
1132 ⁺	Rapid	Very Fast	Strong	Large	Clear
1123 ⁻	Rapid	Moderate	Good	Mid-sized	Cloudy
1152.6 ⁺	Rapid	Very Fast	Strong	Large	Clear
1143.4 ⁻	Rapid	Slow	Poor	Mid-sized	Cloudy
1134 ⁺	Rapid	Very Fast	Strong	Large	Clear
1112.5 ^o	Rapid	Very Fast	Strong	Large	Clear
1130 ⁺	Rapid	Very Fast	Strong	Large	Clear
1126 ⁻	Rapid	Moderate	Fair	Small	Cloudy

- + Cationic type
- Anionic type
- o Non-ionic type

Table 8. Gel-log Test with Known Dosage Stock Solution

	Flow Rate (gpm) (lpm)	Flocculant Flow Rate (min)	Test Length (ppm)	Flocculant Conc. (NTU)	Turbidity (lb/1000gal)	Dosage
test 1	1000	10	2	6.3	2,880	0.05
test 2	1000	24	2	15.2	1,320	0.13
test 3	500	24	1	28.2	730	0.24
test 4	300	24	1	46.5	500	0.39
test 5	150	24	1	98.8	460	0.82

original turbidity of pond water: 4400 NTU

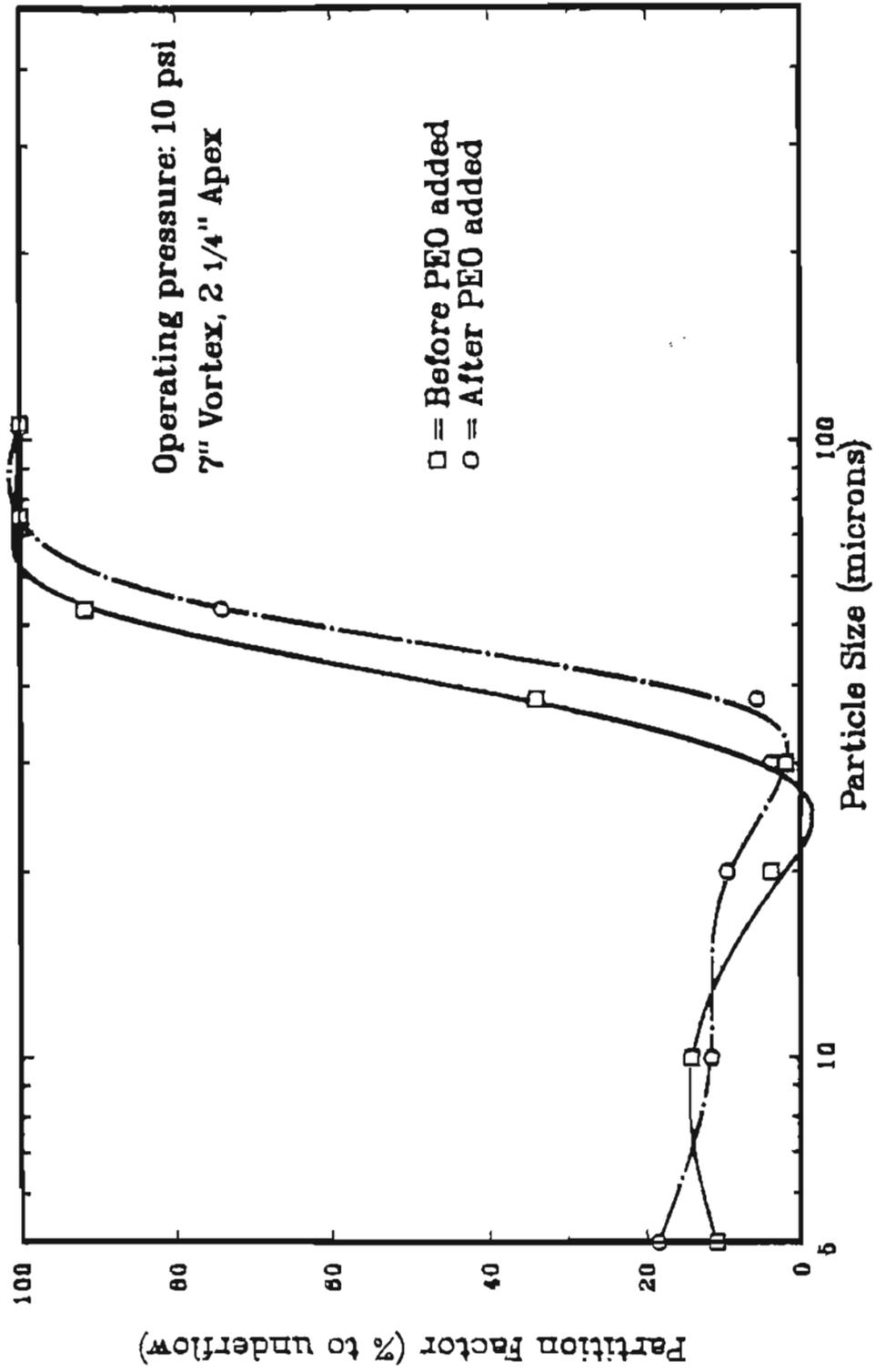


Figure 21. Partition Curves of PEO Test.

tests conducted. A breakdown of the SS-H system's capital cost is shown in Table 9.

Operational costs arose from: (1) lost production time and (2) extra power requirements. For every plant move, from cut to cut, there was an additional 2-3 hours spent to set up the system. Six moves were made in the 1986 mining season assumed that there was one day of production lost which amounted to approximately \$10,000. The operation of the hydrocyclone feed pump required a change of generators; from 100 KW to 150 KW. The difference in the fuel consumption between these two units amounted to an additional \$22,000 for 1986.

A series of ten settling ponds were built and maintained during the 1985 season, requiring 148 hours of equipment time and 51 man days of labor. A breakdown of pond system costs is given in Table 10.

The 1986 SS-H system costs more than twice those of the 1985 ponds system. However, the cost of the SS-H system included one-time equipment purchases, while the construction and maintenance of ponds required intensive labor cost. Total operational cost (including amortization, testing, and bonus etc.) for GHD Inc. for 1985 and 1986 were

provided by the mine manager, Table 11, and show that for these two years operational costs decreased, while the sluicing capacity increased.

Based on the gel-log dosage test conducted at this mine (Table 8) the amount of gel-logs required to reduce the pond effluent to a 500 NTU level was 0.39 lb/1000 gal. The amount of water to be treated per day, based on a 2,000 gpm water usage 24 hours a day, was 2,880,000 gal. This would require 28 logs per day. The cost of purchasing gel-logs was \$3.50 per pound or \$140.00 per log.

Table 9. 1986 Capital Cost of SS--H System.

Equipment	Cost(U.S.\$)
Sand Screw	21,000
Hydrocyclones	14,000
Pump	11,000
Conveyor (used)	13,000
Fabrication	17,000
Freight	4,000
Total :	80,000

Table 10. Total Cost of 1985 Pond System

Equipment Time	Cost
JD 450 Backhoe	19 hrs \$30/hr \$570
Cat D8H Tractor	60 hrs \$85/hr \$5,100
Cat D8L Tractor	52 hrs \$120/hr \$6,240
Cat D9L Tractor	17 hrs \$150/hr \$2,550
<hr/>	
Labor	Cost
Pond Construction and Maintenance	
25 man/days \$275/day	\$6,875
Supervision, Surveying	
26 man/days \$350/day	\$9,100
Materials and Supplies	\$4,800
Equipment Repairs	\$2,640
Total:	\$37,875

Table 11. Comparison of Operational Costs 1985,1986

Year	Operation Cost	Stripping	Sluicing
1985	\$1,020,000	180,000 yds 3	192,000 yds 3
1986	\$971,000	180,000 yds 3	300,000 yds 3

Chapter 7. Conclusions and Recommendations

1: Conclusions.

1. The performance of the SS-H system under present operational conditions is considered appropriate. The cut size, d_{50} , for the hydrocyclone was about 50 microns and the SS-H system as a whole recovered 85% of the solids from the sluice tailing stream. In addition, the sand screw underflow solids served as an adequate filter media for the hydrocyclone underflow. The effluent of this system under normal operation conditions has 47 ml/l of settleable solids and a 6,000 NTU turbidity.
2. From the test results, feed conditions were singled out as the most influential operational factor for this system. When changed to a recycle operation the process water feed condition changed and the system performance varied greatly. Other operation variables such as apex sizes, and vortex finder sizes, did not drastically alter the performance of the SS-H system. When operated in open circuit with the sand screw, the cyclone underflow product was too wet to be pushed by a dozer. This was not true when hydrocyclone underflow discharged onto the coarse solids of the sand screw underflow.
3. The sand screw employed in the SS-H system not only settled the coarser solid particles in the tailings discharged from the sluice box but also transported them to a tailing pile immediately. From a material handling point of view, this has the advantage of reducing equipment needed to handle the tailings. More importantly, this allows equipment to be used to increase stripping and/or paydirt pushing. The addition of the hydrocyclones increased solids removal efficiency by nearly 50% from the sand screw overflow which significantly increases settling pond life.
4. The performance of 10 inch and 4 inch cyclones, tested in this study, indicated that smaller cyclones would have an effect on the cut sizes. Another independent study utilizing cyclones to treat the placer mine tailings was conducted in interior Alaska (Peterson et al., 1987). In the study, the tailings from a jig were directly pumped to a 15 inch primary cyclone. The overflow from the primary hydrocyclone was pumped to a 4 inch secondary hydrocyclone. The overflow from the 4 inch hydrocyclone was collected and recycled.

The underflow of these two cyclones was discharged to a stacking conveyor. This recycle system was abandoned because plant roots, sticks and other debris, that could not be separated by the cyclones, accumulated in the system and cause valve, pipe, and nozzle plugging problems. This problem did not present itself in the SS-H system because the pool area of the sand screw and the overflow at the sump removed these low-density materials.

5. Settling pond construction in the Alaskan placer mining industry has certain limitations. Space required to build adequate ponds is not always available in narrow stream valleys and the rapid dynamics, of such mining operations require continuous pond construction. Although the cost of installing a SS-H system may be a drawback, it does provide advantages in materials handling.
6. The present SS-H system will not produce a recyclable effluent or an acceptable discharge to meet water quality standards without further treatment. Solids removal may be improved by the use of polymer flocculants. The test conducted with gel-logs showed that a high dosage of the logs is required to clean the effluent. This may be due to the fine particle sizes found in the settling pond, 90% minus 2 microns, and the high level of total suspended solids, 6150 mg/l.
7. Experience gained from the dewatering of phosphatic slimes (Nagaraj et al., 1977) indicates that the suspension properties and clay minerals present make each desliming problem unique. Moreover, the structure of the floc and the flocculation conditions also have an effect on dewatering (Hogg et al., 1986).

2: Recommendations

The following recommendations are proposed as areas of study that would be beneficial the Alaska's placer mining industry:

1. Further study the hydration of gel-logs in relation to water flow and temperature at placer operations. Study combinations of different types of gel-logs to reduce turbidity.
2. Study the possible use of shear resistant flocculants in conjunction with the SS-H system.
3. Study the application of multiple stage hydrocycloning to increase efficiency of solids removal.

4. Study the properties of clay-water suspension and the appropriate flocculation conditions which are important to the development of suitable dewatering techniques.
5. A more detailed, long term cost analysis is suggested to compare the SS-H system against multiple pond construction and maintenance.

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APPENDIX

Appendix 1: Ceramic Tile Method of Clay Minerals Identification

General:

Clay minerals have a layered structure which is usually very fine-grained. Because they so often occur as submicroscopic particles X-ray Diffraction techniques are generally used to identify these minerals.

In most clay minerals the layer-like structures are similar in a-b directions, but diagnostic in c direction. Therefore, samples are usually prepared as an oriented aggregate to enhance the basal or 00l reflection. The diffraction pattern generally identifies the characters of the clay minerals as manifested by peak position, peak intensity and peak shape. Peak position is determined by the Bragg law, peak intensity is controlled by chemical composition and atomic position in the unit cell and peak shape for a given species is limited to non-mixed-layered clays. A simple identification procedure is based on the careful comparison of peak position and intensities to ASTM data published in tables or cards.

Many samples will produce peak interferences which preclude a simple approach or render the conclusion uncertain. Then, the sample will require one or more treatment methods such as: thermal treatment, ion saturation and/or organic molecule solvation.

Sample Preparation:

1. Silt-Clay fraction separation: Minus 400 mesh material was collected and centrifuged to separate the clay size sample at minus 2 microns.
2. Ceramic tile mounting: A minus 2 microns suspension of the sample was pipetted on a cleaned, porous ceramic tile to which a water aspirator vacuum was applied to remove the water

through the substrate and deposit the clay film on the surface. This was allowed to dry prior to examination by x-ray analysis conducted from 2° to 30° 2θ .

3. Special treatment: Three more samples were prepared using the same procedures mentioned above, then one of these samples was glycerol solvated, and the other two heated at 300°C and 550°C respectively. Glycerol solvation was performed for 8 hours and thermal treatment was performed for 1 hour. These treated samples were examined through a 2θ range from 2° to 15° .

Results:

1. The diffraction patterns for these analyses are shown in Figure 22, indicating major peaks are at 2θ angle of 6° , 8° , 12° , and 17° . With corresponding d-spacing of: 15.24\AA , 10.65\AA , 7.56\AA , and 5.13\AA .
2. The profiles at 8° are unaffected by glycol solvation and thermal treatment and with its characteristic d-spacing indicated that mica is one of the clay minerals in this sample.
3. The rest of the profiles are a mixture of clays with possible species being chlorite and kaolinite. Chlorite has a basal series of diffraction peaks based on a first order reflection at 14.2\AA and kaolinite has reflection based on a 7.1\AA structure. After thermal treatment, the 12° profile disappeared as the kaolinite structure collapsed at 550°C . On the other hand, chlorite dehydrated at this temperature but the peak did not disappear.

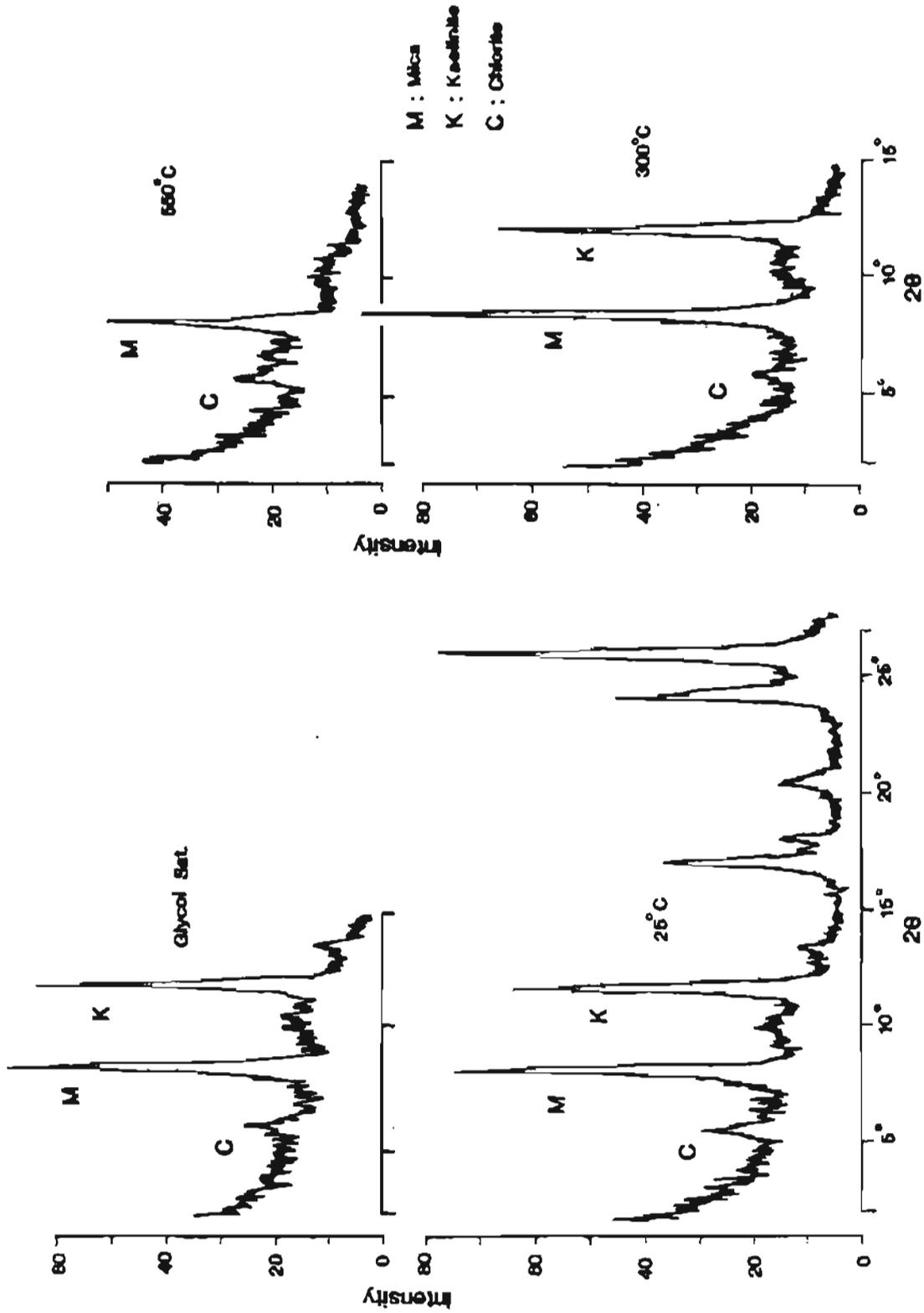


Figure 22. X-ray Diffraction Pattern of GHD's Sample.

Appendix 2: Particle Size Analyzer: SediGraph 5000ET

General:

With this instrument, sizes are measured by allowing particles dispersed in a liquid and contained in a cell to settle in accordance with Stokes' equation and yield a cumulative mass percent distribution in terms of equivalent spherical diameters.

The concentration of particles is determined by means of a finely collimated beam of x-rays. Since the absorption of x-rays is directly proportional to particle mass, the difference between initial and instantaneous transmitted x-ray intensity is electronically converted to a cumulative mass finer than a certain particle size on a X-Y recorder. To minimize the time required for analysis, the position of the sedimentation cell is continuously changed so that the effective sedimentation depth is inversely related to elapsed time. This cell movement is independently coordinated with the X-axis of the X-Y recorder to indicate the equivalent spherical diameter corresponding to the elapsed time and instantaneous sedimentation depth.

The time required for analysis is depend on particle-liquid density difference, liquid viscosity and the minium size desired. A functional diagram of the instrument is shown as Figure 23.

Specifications:

Power requirements: 115 VAC, 60 Hz, 350 VA.

Particle Size range: Equivalent spherical diameter from 100-0.1 microns.

Sample Volume: A dispersion of at least 25 cm³ is recommended.

Particle Concentration: 0.5-5% (by volume) depends on the x-ray absorptivity of particles.

X-ray Sources: Primary radiation is the L- α line of tungsten having an energy of approximately 10 kev and a wavelength of 0.125 nm.

Needed Materials: Stainless steel, Homalite (an aliphatic polycarbonate), plasticized polyvinyl chloride, silicon rubber, Viton, Pyrex, and Teflon.

Acceptable liquids: Any liquid compatible with the wetted materials and not highly absorptive of X-

ray can be used. Suitable liquids include water, glycols, kerosene, mineral oil, alcohols, and alkanes.

Analysis procedure:

1. Turn MASTER, RECORDER, and X-Ray keys to On position. Allow at least one hour for the X-ray tube to achieve stable output.
2. Disperse the sample to be analyzed with 0.55% Calgon solution.
3. Calculate and set the appropriate rate and establish the largest particle size requirement.
4. Install a clean sheet of graph paper on the recorder with the sample description and other pertinent information entered, also set the CYCLE switch according to the option desired.
5. Move the RUN switch to RESET and then return it to its center OFF position. Turn the 100 PERCENT knob fully clockwise.
6. Check the recorder reference baseline using the ZERO knob and the ZERO pushbutton.
7. Flush and fill the cell with pure, particle-free liquid and set the recorder baseline to 0% on the graph paper using the 0 PERCENT knob, check INTENSITY (meter reading should between 40-50 μ A for water).
8. Transfer the dispersed sample into the sample cell, the INTENSITY meter reading should between 30-35 μ A.
9. Adjust the recorder to 100% on the graph using the 100 PERCENT knob.
10. Check the starting diameter, rate, and inspect the sample cell for bubbles. Switch recorder pen to lower position.
11. Start the analysis by switching the RUN switch to the ON position. After the analysis is completed, reset the instrument by switching the RUN switch to RESET, clean and flush the cell leaving it filled with clean liquid.

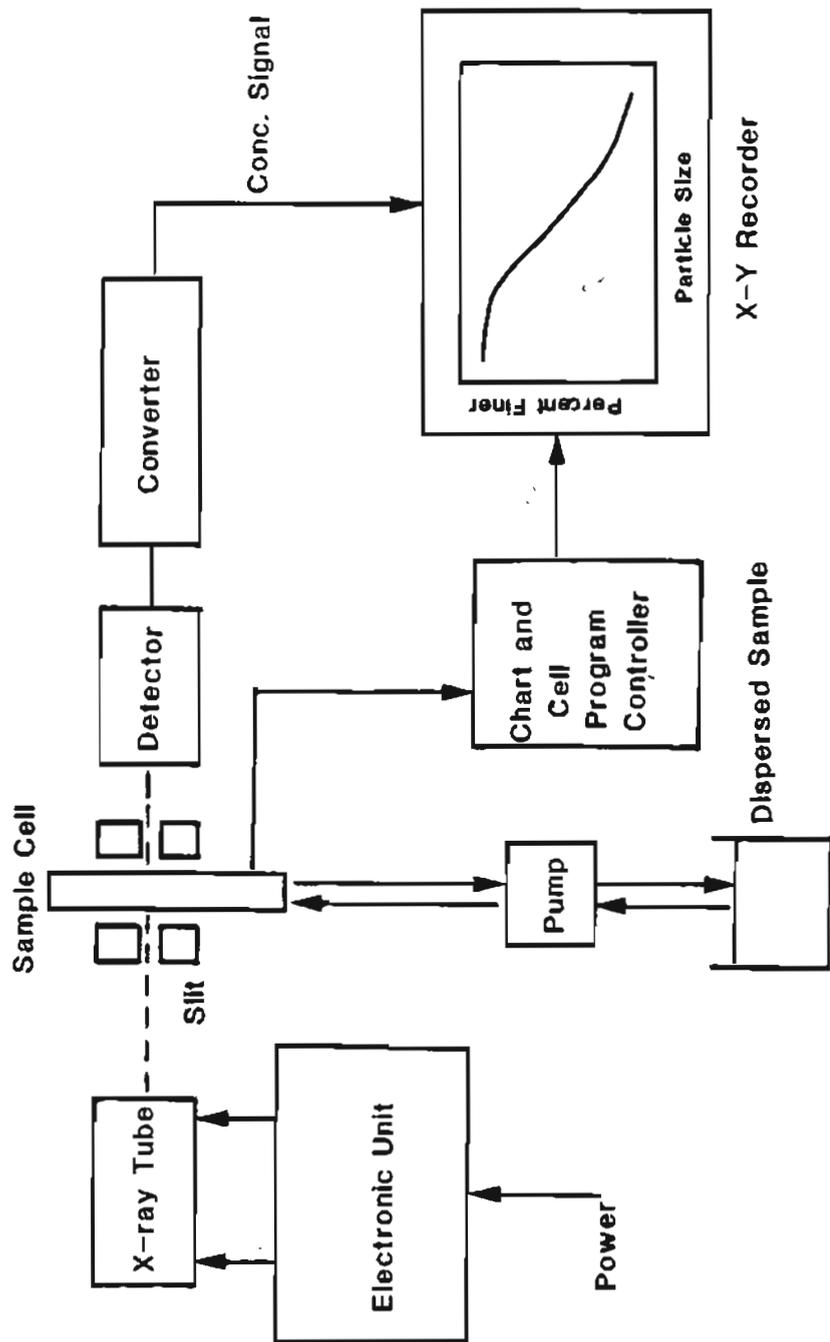


Figure 23. Functional Diagram of SedlGraph 5000ET Particle Size Analyzer.

Calibration:

Five reference materials of defined particle size, BCR No. 66 to 70, were used to verify the size distributions generated from this instrument. These materials are samples of natural quartz covering the sizes from 630 to 0.35 μm which had been analyzed at several laboratories in the United Kingdom and the Federal Republic of Germany by using sieving and pipette gravitation sedimentation method. Sample No. 67 and No. 69 consisting of particle sizes of 90 to 2.4 μm were selected as standards and run through this instrument for comparison with the results from the reported data.

The results of this calibration run was compatible with the published data. A point worthy of notice is that the temperature has an effect on the results as the liquid properties such as viscosity changes as the temperature varies. Another important point noted in the report (Wilson, et al, 1980) is that the reference materials may be used to test other methods of measuring the Stokes' diameter of the particle, but the size distribution must be determined with respect to mass. The size distribution obtained from the SediGraph 5000ET depends upon the absorption of x-rays at a certain particle size.

Appendix 3: Doppler Flowmeter: Controlotron model 183P

General:

To detect the flow rate within a pipe, two transducers are mounted outside the pipe. The 'transmit transducer' injects a beam of ultrasonic energy into the pipe which refracts into the liquid at an angle θ after passing through the pipe wall. If the liquid contains some 'scattering' particles, a reflected beam will enter the 'receive transducer'. Since the particles move at the flow velocity of the liquid, the apparent frequency of the reflected signal will be doppler shifted. This signal is then converted to a pulse rate that is proportional to the liquid flow velocity. The pulse train is ultimately used to operate the flow indicator, the totalizer and other circuits.

Figure 24 shows this detection principle.

In general, a liquid that is murky or scatters light is a suitable doppler liquid. Since the operation depends on the existence of a sonic reflection from particles moving at the fluid velocity, it is essential that the liquid carry some particles. In a water media a particle larger than 0.005 inch will reflect adequately.

The pipe wall must be a sonic conductor, a quality found in metal and plastic, but not in concrete or transite pipe. A pipe size of 3/4 inch or up with a wall thickness up to 1 inch can be used.

The location of the transducers is dependent upon the opaqueness of the liquid. For relatively sonic transparent liquids the transducers can be located at opposite sides of the pipe. For a liquid of medium opaqueness or when using metallic pipes the transducers are located at right angle to

each other, and for extremely opaque liquids, the transducers are located adjacent to each other. In order to enable coupling of sonic energy between transducers and pipe a coupling compound is used.

Operation Procedure:

1. Locate a suitable place on a pipe avoiding elbows or other areas that may cause abrupt flow change. It is also important that the location is free from grease or rust.
2. Install the transducers according to the condition of the liquid to be detected.
3. Before turning on the power, check the following front panel control: RANGE switch at X1 position, THRESHOLD switch at STD position, FUNCTION switch at OFF position, and TOTAL switch at TOT position. Dial in pipe diameter, and set I.D. multiplier switch to X1 for 0.5-10 inch pipe I.D. or X10 for 10-100 inch pipe I.D.
4. Start up and Operation: Set the FUNCTION switch to the INT FLOW position. Make sure the THREHLD control is in the STD position which permits the user to accommodate ambient noise conditions at application site.

Before taking any flow rate reading, rotate the FUNCTION switch to the SIG position. This permits the meter to display the Doppler signal activity as a percentage (0 to 100%). For a significant meter reading the signal should be above 15%.

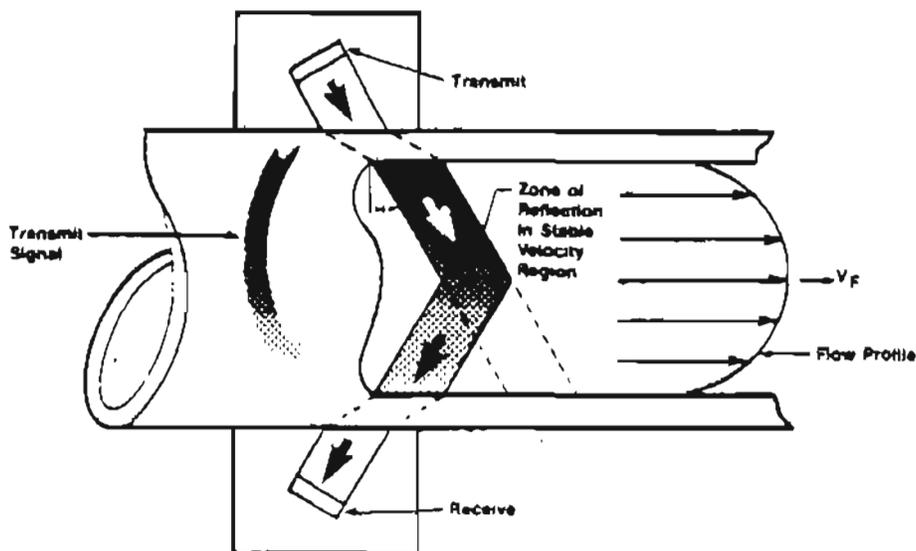


Figure 24. Simplified Detection Principle of Doppler Flowmeter.