# Sustainable Improvement in Safety of Tailings Facilities
# TAILSAFE

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# Tailings Management Facilities

# Water Management and the Use of Thickened Tailings

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# Tailings Management Facilities - Water Management and the Use of Thickened Tailings

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**Other reports of the TAILSAFE project:**

Witt, K.J. & Schönhardt, M. (Eds., 2004): Tailings Management Facilities – Risks and Reliability. Report of the European RTD project TAILSAFE, http://www.tailsafe.com/, 176 p.

Engels, J. (Ed., 2004): Tailings Management Facilities – Intervention Actions for Risk Reduction. Report of the European RTD project TAILSAFE, http://www.tailsafe.com/, 150 p.

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Kreft-Burman, K., Saarela, J. & Anderson, R. (2005): Tailings Management Facilities – Legislation, Authorisation, Management, Monitoring and Inspection Practices. Report of the European RTD project TAILSAFE, http://www.tailsafe.com/, 66 p.

Niederleithinger, E., Kruschwitz, S. & Martin, T. (2005): Non-Destructive and Minimally Intrusive Methods for the Investigation and Monitoring of Tailings Impoundments. Report of the European RTD project TAILSAFE, http://www.tailsafe.com/, 51 p.

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# CONTENT

# 1 INTRODUCTION

For the avoidance of hazards, safe design and operation of a tailings facility is vital. Environmental requirements of any mining and mineral processing operations are high. Within the frame of the EU funded R&D project TAILSAFE (*Sustainable Improvement in Safety of Tailings Facilities*), focus of the research was the safety of tailings facilities, including many areas of this field such as dam stability, dam condition monitoring and site investigation by different types of geophysical methods. Considering the whole operation cycle of ore processing and tailings production and handling, water management and advances of paste technology are of great interest. The present report represents the results of Subworkpackage 4.2 (*Implementation and improvement of water management and paste technology for reducing tailings facility hazards*) of the TAILSAFE project.

The superior objective of many efforts aimed at improving the safety and environmental characteristics of tailings disposal facilities is the reduction of water contents. Reducing the amount of the water at the disposal point results in more stable tailings facilities with decreasing risk and lower probability of pollution of the surrounding groundwater and soil. Reducing water quantities is possible in the mineral processing plant or inside the pond itself and can be carried out utilising different set-ups of technology, transport parameters and risk levels. The application of advanced paste technology, including paste thickening and transport, allows to raise the safety state of tailings deposits. In the frame of the present study, the following three aspects were considered:

1. **Water management:** The first part of this paper summarises the techniques and methods of keeping the water balance of tailings ponds at acceptable levels. Tools of water management both inside the pond and during mineral processing operation are discussed.

2. **Hydraulic paste transport experiments**, dealing with the evaluation of material and technical parameters governing paste transport in pipelines. Paste transport tests have been carried out with selected model materials and actual flotation tailings from one of the project's case study sites (Gyöngyösoroszi, Northern Hungary). One of the targets of the investigations was to determine the highest solid concentration still allowing to hydraulically transport the tailings. The main transport properties, both from material and technical point of view, were determined.

3. **Thickening process tests:** This part of the work gives a short introduction of our investigations on paste thickening. The section reports about the theoretical background of the design of continuous thickeners, about the principle of a prospective new thickener based on vibrating rods, about a pilot-scale new test equipment to carry out classical column settling experiment with or without vibration with the continuous measurement of the vertical concentration distribution in the test column, and about three series of settling experiments.

# 2 WATER MANAGEMENT OF TAILINGS FACILITIES AND PROCESSING PLANTS

Materials described as tailings are as varied as the type of minerals processed and the methods of processing. Tailings are generally defined as the solid waste product of a mineral concentration process. Regarding grain sizes they may range from coarse granular material to extremely fine colloidal slime. Examples of fine tailings are those generated by flotation, magnetic separation and agitation leach processes which require the original ore to be reduced in grain size to facilitate removal of the mineral values. However, two different tailings materials showing the same grain size distribution do not necessarily respond to the same method of disposal, because of different alteration of the particles and particle surfaces before or during the concentration or leach process, or because of different mineralogical composition.

As a waste material, tailings are usually removed from the processing plant for disposal. Their mass is often nearly as great as the original plant feed mass. The problem of disposal concerns both the solid materials and the accompanying liquids. In most operations these wastes are transported as a slurry to an impoundment area where the solid fraction is retained, and often a part of the liquid fraction is returned to the concentration process for reuse. Removal of non-recycled water is of first importance. Lack of water control can result in a lost structure due to embankment failure. Typically, the majority of fresh water required for processing is required to replace water losses from the system at the tailings storage facility. Water in the slurry is either retained in the tailings voids or lost to seepage and evaporation. Water usage efficiency can be improved significantly by thickening tailings, because the thickener overflow is retained within the plant circuit and is therefore not exposed to such losses.

Mudder et al. (2001) describe approaches to determining the overall water balance of a mining project and selecting options to deal with water management issues. The management issues usually involve having to cope with too little water, which requires supplemental supplies, or too much water, which requires storage and discharge with or without prior treatment. The development of a comprehensive site water balance and management plan should already begin during the feasibility analysis and permitting of a mining operation, with equal emphasis during start-up, in the operation period of the fully developed mine, and at its closure. The proposed methodology emphasizes the use of a systems approach which includes examining the entire mining operation, developing a complete flow network describing the various sources, sinks and flow pathways associated with both impacted and clean water. The natural rainfall/runoff cycle is superimposed on the mine flow sheet to determine the total quantities of water that must be dealt with on an annual basis, as well as during extreme dry and wet periods. Finally, value engineering is employed to determine the most cost-effective measures for providing make-up water supplies, storage of excess water that accumulates seasonally, reuse of water, and systems for treatment and discharge of water. Excess runoff and process water is stored in a storm water surge pond or flow equalization basin from which water is recycled in the processing circuit and/or discharged to the treatment facility in a manner to minimize flow fluctuations. The traditional storage area is the tailings impoundment or decant pond, but heap leach facilities have also been used for this purpose.

With an understanding of the site water balance and an identification of potential water sources and characteristics, attention can be focused on selection of an appropriate water treatment option. Water withdrawn from the pit (or underground mine) is used for dust suppression and as make-up water in the mill. During extreme storm events, excess pit water can either be temporarily stored in the pit, or pumped into a surge pond. Runoff and leachate from a low-grade ore stockpile area is also collected for possible reuse or treatment. In this example it is assumed that the quality of this water is impaired due to acid generation in the ore, and the pond acts as a zero-discharge holding facility. The water is ultimately pumped for reuse in the mill. Provision is made for an emergency overflow for extreme precipitation events which exceed the design storm event (where permitted). The emergency overflow option may be needed to maintain the geotechnical stability and integrity of slopes or embankments. Runoff from the mill site is directed either into the tailings pond or into a holding pond for treatment, reuse or discharge. Care is taken to ensure that all low pH solutions drain into this pond and not into a process pond in which hydrogen cyanide gas generation could occur. Process solutions can be combined with acidic drainage after it has been neutralized to a basic pH. Makeup water for the mill is derived from the pit, site runoff water, tailings pond and a fresh water reservoir. A portion of the makeup water used as boiler feed and gland seal water must be of a very high quality. In this case, it is drawn exclusively from the fresh water reservoir. Tailings slurry is discharged from the mill into the tailings impoundment. In the impoundment, water is lost by permanent entrainment in the tailings solids and by evaporation. Seepage through the embankment or dam is collected in a pond and pumped back into the impoundment, reused in the metallurgical circuit or is sent directly to treatment.

A useful method of approaching a water balance evaluation is to complete it in two stages. The first stage is a mean annual assessment of the long-term surplus/deficit situation. This involves developing mean annual values for each component, taking into account the leakages and those water sources that can be discharged, followed by determination of the overall water balance. The second stage involves performing short-term water balance assessments over periods of a few hours to a few months. These are required to determine the volumes of surplus water that can be generated over a short period during extreme wet conditions and storm events. For projects that have a water deficit, such assessments are required to determine the amount of storage that must be provided, or to determine possible short-term water discharge requirements. For projects that have a water surplus, the calculations are used to determine the maximum discharge rates or the amount of storage required to maintain a constant feed to the treatment plant. There is a trade off between the amount of storage provided and the capacity of a plant required to treat the water before it is discharged. Value engineering should be carried out to determine the best combination of storage capacity and treatment capacity. When evaluating the short-term surplus it is useful to use a statistical basis (i.e., select an extreme event of specified probability). Commonly used events include the 1 in

100-year and the 1 in 10-year storms, although some regulatory agencies are now requiring more extreme events up to the Probable Maximum Precipitation (PMP) event. It is important to note that each project will have its own unique critical duration which results in maximum storage and treatment requirements. As a general rule, the critical duration is short, typically on the order of a few days or weeks. In wet climates the critical duration may extend over several months. The critical duration can be determined by selecting one of several time periods such as 24-hour, 7-day, 30-day and/or 90-day periods and performing additional water balance computations. The critical duration is the one that requires the maximum storage volume and/or the maximum discharge rate.

The potential water sources are identified through development of a detailed and time-dependent site water balance. These sources include domestic sewage, acidic mine drainage, runoff from ore stockpiles and waste rock storage areas, leachate from disposal areas and tailings impoundments and turbid water originating as runoff from disturbed catchments within the mine site. Two sources of water are anticipated from an underground mine. The first source includes groundwater intercepted prior to exposure to the mine workings. The chemistry of the water is derived from the combined characteristics of the various groundwater sources entering the mine. The second potential source of underground mine water includes groundwater impacted by drainage from the mine backfill operation. The chemical characteristics of the water are derived from combination of the groundwater chemistry with the chemistry of pore water draining from the backfilled solids. The constituents of concern in contaminated mine water include trace metals and total suspended solids. The characteristics of the runoff from rehabilitated waste rock storage areas can be derived from the results of batch extraction and long-term column leachate studies conducted under natural conditions. The constituents of concern in the runoff waters from rehabilitated surfaces include trace metals and total suspended solids. The characteristics of leachate from ore stockpiles are generally derived from long-term field leachate studies. The constituents of concern in the leachate include elevated levels of trace metals and total suspended solids. Turbid waters, originating as runoff from disturbed areas, may require treatment for the removal of total suspended solids. The concentration of total suspended solids in the runoff varies with the intensity and frequency of precipitation impacting the catchments. Estimates of the total suspended solids concentrations in the turbid water are derived from either field or laboratory studies.

In summary, there are several ways to improve water balances in ore processing and tailings disposal facilities by the use of thickened tailings. These are:

1. Hydraulic transport of low or not too high concentration slurries into conventional tailings facilities. Water removal is necessary for return water, and for keeping the water balance at safety level.
2. Thickening tailings at the plant and transport paste or thickened tailings into the disposal area. Thickener water overflow is retained within the plant circuit.

3. Hydraulic transport of low or not too high concentration slurries to the disposal site. Thickening is conducted at the site before disposal. Return water can be recycled.

In this work, the processing plant is handled as a black box, and only the thickening process and the tailings transport and disposal are addressed.

## 2.1 Low Concentration Transport Method

Low concentration slurries are transported to the tailings disposal facilities through pipelines (Fig. 1). For safe and economic operation and to establish an optimal solids concentration the tailings are either dewatered before slurry transport or, in some cases, fresh water is added. Tailings transport is achieved through pipelines, using different types of slurry pumps. At the end of the pipeline the transported tailings-water mixture is charged into the tailings pond or series of ponds. These ponds are designed and built by conventional dam construction methods. In most cases it is necessary to reclaim the clear, supernatant water back from the surfaces of the ponds to keep the water balance in a stable state and to prevent dam failure as caused by, e.g., overtopping. The return water is pumped back to the plant through pipelines.

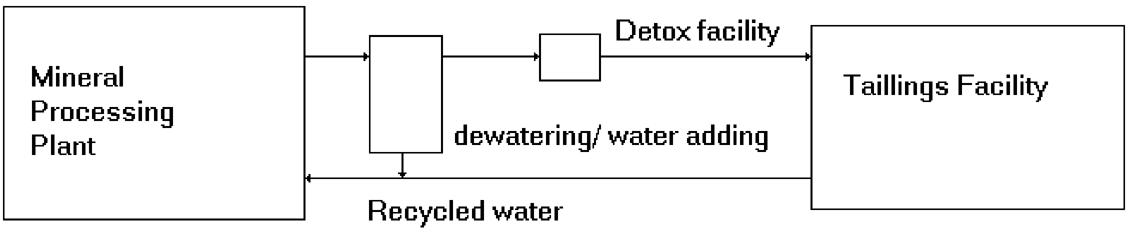



Figure 1: Schematic drawing of low concentration tailings transport.

Pipeline velocity is one of the most important factors in hydraulic system design. For this, we need to adjust the operational concentration of solids in the given particle size. During mineral processing operations, the concentration is set to facilitate the processing mechanism. After that, a suitable concentration has to be obtained for hydraulic transport. In some cases this could mean that additional water has to be added to the slurry or that it has to be dewatered. Generally, the dewatering process produces two different "products". One is the slurry which is ready for pumping into the tailings area. The other "product" is the reclaimed water which is ready to be returned back to the plant. This water usually has a low solids content, but might contain high concentrations of different chemical additives.

### 2.1.1 Hydraulic Transport Systems

Hydraulic transport can be conducted using slurry pumps or pressure feed systems. Applying slurry pump systems, the slurry is flowing directly through the pump. Slurry pumps are made for pumping fluids at relatively short delivery head and that is why these systems are only applicable for short distance slurry transport. Pressure feed systems consists of equipments which allow to insert slurry into high-pressure clean water flows. This technique needs a pump which is able to transport clean water at the needed pressure. Clean water pumps are available for very high delivery heads; therefore these systems are suitable for the transport of slurries over high or medium distances.

Positive displacement pumps and centrifugal pumps were developed for slurry transport purposes (Wills & Truscott 1978, Zarzycki 1983). For non-erosive materials, a high lifetime can be reached for both pump types. In a positive displacement pump, the slurry flows through the valves and is also in continuous contact with the walls of the equipment. Due of the valves, the transported material should contain solids with a predominant particle size below 1 mm and no particles with sizes above 3 mm. The development of positive displacement pumps drives the designers to select such construction solutions where it is not allowed for the solids to enter the pathway of the piston. Blocking of valves is tried to be prevented by rinsing with fresh water. In case of high concentrate slurries, it is necessary to use plain side valves moved by hydraulic cylinders instead of valves. Using of inflexible characteristic positive displacement pumps is quite favourable and safe in hydraulic transport. Hydraulic transport systems built for long distances (some time 100 km) and the transport of coal or ore use this type of pump without exception. Centrifugal pumps are applicable for much lower delivery head than positive displacement pumps, but the maximum transportable particle size depends only on the geometric properties of the paddle wheel of the pump. Centrifugal pumps used for sea excavation are capable to transport particles between 200-300 mm at high volumetric flow rates. Therefore, centrifugal pumps are used in such cases where coarse material needs to be transported over medium or short distances.

### 2.1.2 Dewatering

Operation ranges of various dewatering devices are shown in Fig. 2. Thickeners are generally of the continuous cylindrical type, but the advent of lamella thickeners could mean a changing emphasis as their potential for space and cost savings are realized. Sludge blanket feed thickeners, which use the sediment as a "filter", allow clearer overflows from dilute pulps than conventional clarifiers.

Disc filters and, to a lesser extent, drum filters, are the mainstays for most final dewatering because of their ability to remove most fine particles from a process stream. Fine particles, notoriously difficult to treat with vacuum filters, can now be handled with automatic filter presses. Alternative dewatering systems are possible in certain circumstances, particularly when fines recovery is not important. Under these conditions, screens, hydrocyclones, centrifuges, or a combination thereof can be used more cheaply than filtration.

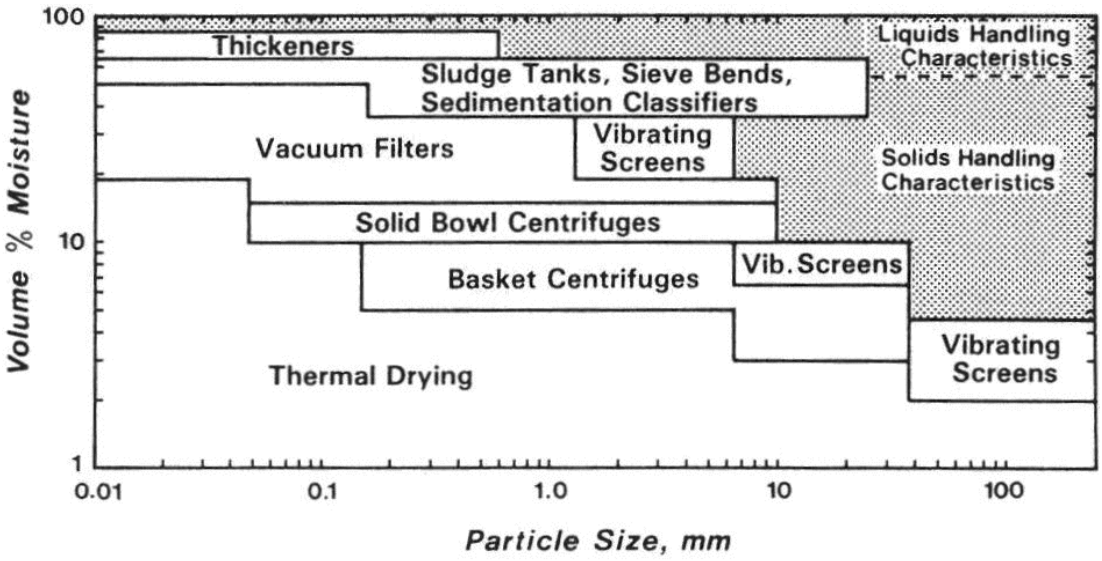



Figure 2: Particle size range handled and approximate moisture produced by dewatering equipments (Weiss 1985).

### 2.1.3 Sedimentation

Sedimentation is the removal of suspended solid particles from the liquid by gravitational settling. Such operations may be divided into thickening and clarification. Although governed by similar principles, these processes differ in that the primary purpose of thickening is to increase the solids concentration, whereas clarification serves to remove solids from a relatively dilute stream. Clarification separations are characterized by sedimentation without a clearly defined interface between clear liquid and sediment, and as a consequence capacity is limited by the amount of solids that can be accepted in the overflow. Performance therefore is characteristical of a wet classifier and can be best analysed as such. Thickening operations on the other hand are characterized by a clear liquid/sediment interface and capacity is limited by the underflow conditions.

### 2.1.4 Sedimentation Equipment

The most common type of sedimentation unit is the **cylindrical continuous thickener** (Fig. 3) with mechanical sludge raking arcos. The feed enters the thickener through a central feed well and clarified liquid overflows into a launder around the periphery. Thickened sludge (the sludge blanket) collects in the conical base and is raked by the slowly revolving mechanism to a central discharge point.

Tanks of sedimentation vessels are normally cylindrical, smaller units being constructed in steel or wood, and larger ones (> 30 m) of concrete. The tank base is a shallow cone to facilitate sludge removal at its apex and is generally constructed of the same materials as the walls, although large concrete thickeners may have an earthen base. The base slope is typicality 80-140 mm/m, but steeper slopes (up to 45°) may be used near the centre of very large thickeners or for "settling" pulps in very small thickeners where height is not much of a problem.

Figure 3: Cylindrical continuous thickener (www.solid-liquid-separation.com).

**Lamella thickeners** (Fig. 4) have been designed to reduce the space requirement of conventional thickeners. They are characterized by packs of sloping parallel plates in the settling area, resulting in very short settling distances (less than the vertical distance between the plates), and the effective settling area is the vertically projected surface of all the plates. The slope of the plates is a significant parameter, in that it must ensure an even flow of the settled solid down the plates to the discharge. In some cases vibrators may be used to ensure such a flow.

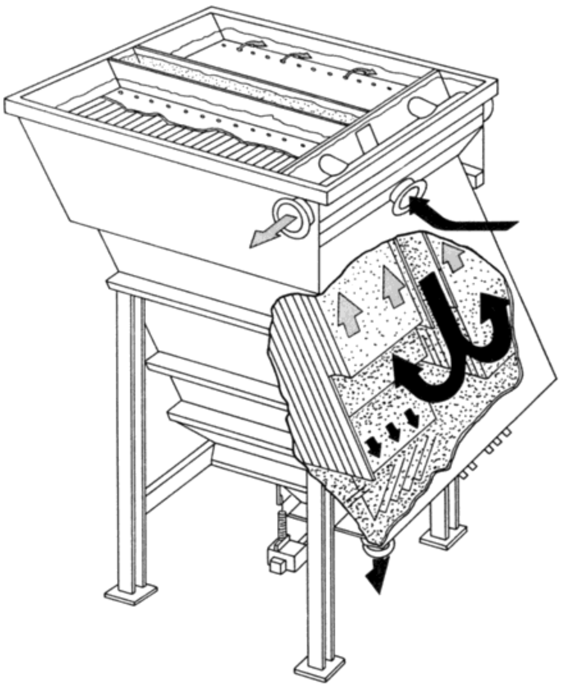


Figure 4: Lamella thickener (Kelly and Spottiswood 1982).

The **Deep Cone Thickener** (Fig. 5) uses comparatively high levels of flocculants. This, together with compression toward the bottom of the cone allows a plastic-like underflow discharge, which can be handled by conveyor. A 4-m diameter vessel has a capacity of about 80-110 $m^3$/hr, and produces a product with 60-70% solids (by volume).

Figure 5: Deep cone thickener (www.solid-liquid-separation.com).

### 2.1.5 Tailings Disposal

There are three common methods of operation when tailings materials are employed for progressive elevation of the dam. They are the upstream, downstream, and centreline methods. Dependent on local conditions other materials may be combined with the tailings to achieve a stable structure in all three schemes, but they must be compatible with the tailings to avoid differences in settling, compaction, and permeability which could result in instability.

#### 2.1.5.1 Upstream Method

The trend in recent years towards large tonnage operations has resulted in increasing the final height of tailings dams because of space limitations. This increased height requires more care in the placement of tailings in the embankment so that stability is built in from the ground up to the required final height. The upstream construction method consists of a starter dam erected at the outer toe with discharge of tailings material into the disposal area using spigots or cyclones for deposition of the coarser material rest the starter dam and flow of the finer particles toward a pond area for settling and water clarification. With this method the centreline of the tailings dam structure moves progressively upstream (toward centre of the disposal area) as the embankment elevation is increased, while the outer slope of the embankment is preserved for stability. Each successive lift is supported on top of the previous dike, with the upstream portion discharging over previously deposited sands. Ideally, the initial deposition into a new area should be such that a sand beach is quickly developed which extends far enough into the pond area to act as a firm base for the final structure. Beyond the beach, usually initially formed from cyclone underflow, the rest of the tailings find a channel into the pond area. The most common upstream feeding method is to place the main feed line (or spaced discharges from it) along the crest of the dike for spigot or cyclone discharge.

**Spigoting:** Using the spigot method, non-segregated tailings are delivered at regular intervals around sections of the periphery to allow natural segregation for continued development of a strong beach. The feed line has valves and spigot piping installed every 3 to 15 m (depending on the size of the installation) for even distribution. Material is fed into the pond in a sequence which raises the sand level equally along the dam to control the location of the open water surface of the pond. When the disposal area has been filled to the top of the dike, the feed flow is transferred to another section, which may be a separate dam or simply a division of a single large area. A suitable drying period follows, during which the spigot pipes are removed to provide a room for building the next dam level elevation.

As soon as the beach has dried and compacted sufficiently to hold heavy equipment, the dike is raised by a dozer pushing sand into position, or a carryall, or a dragline digging a borrow pit in the beach sand and depositing it on top of the dam. A combination of dozer and dragline may be used, with the dozer moving material to the dragline to reduce the size and depth of the borrow pit. Use of the dozer alone reduces the possibility of slime layer inclusions under future lifts by creating a relatively shallow borrow area, but involves moving a larger volume of tailings. The method of dike elevation is dependent on the height of impoundment required, the segregation characteristics of the deposited tailings, and the fill cycle schedule needed to maintain a continuous operation. Dozer edge erection is economical up to approximately 2.5 m while a dragline is practical to between 4.5 and 6 m per lift. Many operations use a dozer to erect dividing partitions from the dam toward the slime area during the erection procedure to prevent slimes from travelling laterally and thus weakening the structure. These partitions also accelerate water accumulation in the pond and reduce evaporation loss.

Upon completion of the lift, spigot pipes are either reinstalled without moving the feed line, or the feed line is transferred to the top of the newly erected dike with reinstallation of the spigots. Location of the elevated embankment level in relation to the previous lift is dependent on the outer overall slope required for structure stability, and the width of the dam is governed by equipment needs.

Except where the tailings pond is the principal water reservoir, the water pool should be held only large enough to allow ample settling time to get a clear overflow, or to allow any necessary chemical changes to be completed. The water pool should be herded around the tower very carefully to keep it far from the berm. Mismanagement of this operation can result in deposition of an impervious slime layer across which the future drainage could be forced laterally through the dike, with resulting weeping, piping, and possible failure. The beach is thus raised successively all around the dam so that the pulp flows directly from the spigot to the water pool and does not meander excessively or run parallel to the dike.

Typical operations in the southwest of the USA building upstream tailings dams by spigoting are Cyprus Pima, Asarco Mission, Asarco Silver Bell, Duval-Sierrita, and Phelps Dodge Morenci, all in Arizona (Keener 1973, Salter 1957, Janes 1975, Ludeke et al. 1974). Berm erection at these properties is mostly done by dozer or dragline, and all have sufficient tailings area available to withdraw one section at a time during berm building and thus achieve the desirable compaction. Compaction is evidenced in the pond area by shrinkage cracking at the perimeter and an actual decrease in elevation of the previously deposited tailings.

**Cycloning:** Berm erection using cyclones is less commonly used in the upstream construction method than direct spigot discharge. The use of cyclones may be a necessity at least for a while if the tailings materials will not classify by natural segregation because of excessive slime; this occurred at Asarco's Silver Bell Inspiration Christmas 7 and Phelps Dodge's Tyrone operations. Operations with cyclones at Tyrone and San Manuele are very similar and generally typical. A main feed line, valved as in a spigot system, feeds cyclones mounted on tripods. These units are assembled to a height of 4.5 m with positioning brackets for cyclones at 1.5 m vertical intervals. The spacing of the assemblies, like that of the direct spigot discharges, which they replace, depends upon parameters earlier mentioned, all with the sole objective of providing a solid beach without slime accumulations. Cyclone overflows are piped about 15 to 23 m into the settling area and continue toward the pond for settling. The slope of the outer face of the embankment is about three horizontal to one vertical overall, slightly flatter than is established for direct spigoting. Tyrone and San Manuel build 10 and 12 m of dike before moving the feed line, which remains on the previous berm between moves. Where the dike is formed directly from the spigot discharges of the cyclones, more sand is usually produced than is necessary, so this method is used in the cold countries to build enough freeboard to allow direct feeding of tailings into the main area part of the year. Otherwise the cyclone may discharge its sands upstream of the berm and subsequently a part of the temporary beach will be used to build a berm. In many cases, cycloning is used only in the initial stages, or intermittently elsewhere around the dam.

#### 2.1.5.2 Downstream Method

In this system, the centreline of the top of the embankment shifts outward as its elevation increases; in other words, the embankment is built up on its outer face. The starter dam is constructed of impervious materials usually consisting of compacted borrow materials containing significant quantities of silt and clay. Each subsequent stage of berm construction is supported on top of the outer or downstream slope of the previous section. If tailings are used for dike construction only the coarse fraction can be utilized. Prior to each downstream extension the previous under-drain system (if used) must be extended to conform with the new profile.

Material for berm erection, using tailings, is normally obtained by cycloning, with the slime fraction moved into the pond. The underflow sands are continuously placed downstream to enlarge the outer dike which grows wider as the height is raised. The seepage water line remains well behind the outer dike face in this type of construction. Several methods are currently in use to obtain the necessary compaction for erection of a stable structure. Coarse tailings may be piped to the embankment and spread in thin layers, or they can be hauled from a central cycloned stockpile, spread, and compacted.

If the volume of coarse tailings is not sufficient for dam construction, local borrow materials can be incorporated for part of the structure. To minimize seepage through embankments constructed with tailings sands, the inside face is normally progressively sealed with impervious soil or covered with slimes from the cyclone overflow product. There are several variations of the downstream construction method in use for sand deposition. With one, the cycloned sand is placed in layers and mechanically compacted, resulting in fairly steep slopes, and a good under drainage system is required. A second variation uses clean, free-draining sand, resulting in a relatively flat downstream slope with a good under drainage system. A third uses cycloned sand deposited between an inner starter dam and an outer waste rock toe dam. Underdrains are required but need not be as extensive since the lower rock-fill toe dam is quite pervious.

One major disadvantage of all methods of downstream embankment construction is the large volume of sand required to raise the embankment. In the early stages of operation it may not be possible to produce sufficient volume of sand to build the berm at the required rate. In this case, either a higher starter dam is required or the sand supply must be augmented with borrow fill. Both procedures add to the cost of the early stages of dam building. Another disadvantage is that the outer face is constantly changing, so planting of vegetation on that face is impractical.

#### 2.1.5.3 Centreline Method

This method of tailings dam construction is a variation of the downstream method, the only difference being that the crest of the dam, instead of moving downstream as the dam is built, rises on the centreline. The major advantage of the centreline method over other methods of downstream construction is that it requires a smaller volume of sand fill. Thus the dam can be raised faster and there is less trouble staying ahead of the fine tailings and water pond during early stages of construction. A disadvantage of the method is the tendency of forming a temporarily unstable inner face. This can happen if the berm grows too steeply; however, even under these conditions a slump into the pond would ordinarily be of little consequence. Breaching of the dam and a serious discharge of effluent and tailings can occur in extreme cases.

#### 2.1.5.4 Other Methods

Due to economics, terrain, or need of sand for mine fill purposes, some tailings dams have been built entirely of borrow material. Some mining developments produce large volumes of overburden and rock from open pit operations which, if a short haul is involved, may be more economical to use for tailings dam erection than the tailings materials themselves. In some instances, particularly where the ore is ground fine and a relatively low sand production is expected, dam construction entirely of waste rock and overburden may be the most practical. Others may find that a combination of waste rock and tailings sands is best. Some plants which use a combination of metallurgical processes, such as vat leaching and flotation, have used vat leaching tailings as the basic berm material, with flotation tailings deposited on the inner face.

The major advantage of using open-pit waste materials for embankment construction is that the method permits the use of conventional water storage dam techniques to provide a structure which can resist large seismic movements. Since the tailings are not required for berm construction they may be deposited without concern about separation of slime from sands.

### 2.1.6 Pond Water Removal Systems

The removal of excess surface water from a tailings pond, during temporary and current operations and after final suspensions of operations, is second in importance only to the retention of solids. Of several criteria for judging one system against another we favour the degree of dependence upon labour, supervision, supplies, and energy. The main question is: During a suspension of plant operations lasting months or even years, must these services continue or, on the contrary, with the mill shut down will the tailings dam take care of itself?

#### 2.1.6.1 Systems Dependent on Continuous Service

Under this heading there first comes to mind the circumstances where the clear water that accumulates on the tailings – and in some cases this may include runoff from the surroundings as well as rainfall upon the tailings – is removed by ditches, siphons, or pumps floated on the water pond or moved from one pond to another as needed.

**Ditches:** In small operations of several hundred up to 1000 to 2000 tpd of ore, and where the tailings disposal area is on a steady slope of 6 to 10%, the dam can be built in a horseshoe pattern so that control of the clear water pond up against the hillside is natural and simple. Ditches of temporary nature can draw off clear water with the help of a weir, and further up the hillside permanent ditches can be built to keep runoff from entering the tailings pond. Unfortunately, this system cannot be depended upon to maintain itself over long periods. The ditches may collapse, and the weir has to be tended. However, services are minimal. While the mill is operating this system is more labour-intensive than many others.

**Siphons:** Some large operations – specifically Kennecott, Garfield, and White Pine, all in the US – have used siphons, usually of variable head. A small pump is used at the high point of the line to prime the siphon from time to time, and a valve controls the quantity of outflow. The use of the siphon is limited by height and distance of flow. The water pond must be reasonably close to the dike, and this entails the risk of slime layer formation at the most dangerous point. This system is cheap to operate, but cannot operate without attention over long or permanent suspension periods.

**Pumps:** Another common method of removing ponded slurry water is the use of pumps mounted on floating barges or on skids at the upstream shoreline. Barge-mounted pump systems are relatively inexpensive to install and maintain and, if the disposal area is lower than the mill site, have the advantage of a decreasing static head on the return water line to the plant site. Once installed, the only construction requirements are those of pipe length changes of the return line. A floating or land-based pumping system is not subject to failure due to decant line problems, but a power failure cancels water removal, and excess water storage can possibly result in overtopping of the dam. Considerable attention to the system is required to maintain pumps at the proper position in the pool as the pond level rises. Removal of floodwaters is limited by pump capacity, and an alternate method of water removal due to surface drainage is required after abandonment of the tailings site. In cold climates freezing water can be troublesome. Compressed water circulation may be required to maintain the barge as a free-floating unit. It is obvious that this system is the least satisfactory and flexible for long suspensions of activity and must be converted (at considerable expense if not provided for initially) to some kind of gravity system in that eventuality. It will be seen under the next subheading that floating or shoreline pumps can be very convenient in the early months or even years of a decant system.

#### 2.1.6.2 Systems Not Dependent Upon Continuing Services

**Decant Lines:** The ideal system from this point of view uses permanent pipelines installed in the ground which carry clear water from the pond to a storage system below the dam or series of dams. Water removal by the decant method is fully by gravity from the water pool. Steel, reinforced concrete, or reinforced precast concrete pipelines are common, but all must be designed to withstand the total anticipated static and dynamic load of the tailings, should rest on a firm base, and should be provided with two or more seepage collars if it passes through the embankment rather than beneath it in its own channel. Such a system is often known as underdrain, because the permanent decant pipelines lie beneath the tailings, in or on the ground. The underdrain lines must be securely anchored for their entire length to prevent the possibility of failure due to flotation or to the tremendous forces that can result from the shifting of semi-fluid tailings in great tonnages from one point to another beneath the surface to maintain isotropic equilibrium. The occasional opportunity to examine the reason for underdrain failure in old dams has shown that these forces are of unbelievably great magnitude.

**Decant Towers:** In the initial stages of underdrain dam building the proper location of the decant towers is of prime importance. In several instances the tower has been placed so close to the berm that by the time the tailings dam grew sufficiently to use the tower, the water pond had left it far behind, buried in a sand beach. For this and other reasons it is usually wise, when tailings are to be deposited on sloping ground (commonly 2-5%), to defer construction of the towers and to use the decant line itself to draw off clear water as the shoreline advances. In small operations the underdrain line is used without a tower and simply extended as needed, with a screen box at the end.

Decant towers have been used for many years and range from simple vertical steel pipes to reinforced round, square, or horseshoe shaped concrete sections. Towers up to 3 m diameters have been used and offer very good skimming properties. Most decant towers have holes cast into them at 100 to 300 mm vertical spacing for water inflow, but same are constructed for use of weirs (wood, concrete, steel) for level and volume control. As the elevation of the pond rises, the holes are plugged or weirs are raised to conform to the clear water level at the tower. Towers must be designed with a substantial foundation to minimize settling, provide entrance for underdrains, and withstand the wind load as well as any movement of the tailings surrounding it. It is important that every part of the decant line and tower be strongly built in order to remain workable even after the deposition of tailings has been completed and the area abandoned. Decant towers generally require little attendance and therefore have a low operating cost, but initial installation costs may be high. Decanted water flows to a collection sump where it is pumped to a concentrator reservoir for reuse. If the water is not required for plant operation, it is usually allowed to flow to a stream or watershed area, although for environmental control purification or cooling may be required prior to release.

Some advantages of a decant line and tower installation are:

1. The structure is relatively simple to design and construct.
2. The tower can be erected to its full height initially, or its height can be increased in stages or sealed off as the tailings dam rises.
3. The system requires only intermittent attention to maintain the proper water level by plugging decant holes or installing weirs.
4. The discharge capacity increases with increase in water depth for quicker runoff removal.
5. The decant system provides an essentially maintenance-free permanent drainage facility following abandonment.

Disadvantages of the decant line-tower system are:

1. The installation cost is high.
2. The pumping head for reclaimed water does not diminish with the age of the system since the pumping system is usually at the lowest elevation of the disposal structure.
3. Any major damage to the system by earthquakes, or settlement or misalignment from other causes, may rupture or block the decant line resulting in possible temporary loss of the dam pending repair or quick conversion to an alternate water recovery system.
4. The possibility of a piping failure along the decant line at the outer toe of the dam.

#### 2.1.6.3 Combination Methods

When starting a new dam temporary pumps are often used because of uneven terrain. That part of the tailings are blocked from the pond area. After a few months this situation usually corrects itself so that the pumps are no longer needed.

### 2.1.7 Seepage Water Control

Seepage from a tailings dam is usually closely controlled. Controlled construction of the starter dam, the decant line through it, drains in and around it, and control of the pond are important factors in seepage control. Most tailings ponds will seep some water vertically through the bottom and horizontally to the downstream face. The amount of seepage at the downstream face determines the stability of the entire embankment. A starter dam constructed of good local materials and well compacted may require upstream filters to drain off the water so that the dam does not become saturated; predevelopment studies may indicate that seepage pressures through the dam will have to be minimized. These filters will keep the phreatic water surface down near the toe of the dam und not at the crest. This increases the stability of the sand structure considerably.

If a drain or filter system is deemed desirable for seepage control, a pipe drain, blanket drain, or a combination of drains will maintain the phreatic surface as low as possible. A perforated pipe parallel to the upstream toe of the dam with a drain beyond the downstream toe will drain the water off. The pipe is perforated only on its lower half und is bedded in washed graded gravel protected by sand filters. The size und gradations of this filter material are critical und must be selected for the particular problem. Upstream from und connected to the perforated pipe drain may be installed a blanket drain for better water removal. The blanket drain thickness may vary between 300 and 900 mm, depending on the expected seepage.

Chemical tests are usually performed on embankment, drain materials, und seepage water to ensure compatibility. For example, the drains cannot be made of carbonate rocks if the seepage water is acidic.

Careful design of the drainage layers ensures satisfactory long-term operation. Over-designed capacity will be helpful if leakage develops. Where the embankment contains zones of material having significantly different size gradation, or where the gradation of the foundation and embankment materials differs markedly, zones of markedly different gradation must again be separated by filter zones to prevent piping and subsequent subsurface erosion. The filter must meet two requirements: it must be more permeable than the adjacent finer soil so that it will drain freely, and it must have a gradation to prevent passage of soil particles into the drainage layer. Particular care must be taken that segregation does not occur during construction. Commonly used rules for sizing filter and drain materials include. Where the gradation differences are great, two or more filter layers may be required to meet the foregoing criteria. In areas where the soil layer is thin or in relatively narrow mountain valleys, the filter blanket should extend upslope several tens of meters and up the abutments, as at Morenci and Climax (Keener 1973). In relatively flat topography the filter blanket need to be only 3 to 6 meters wide along the inner toe of the starter dam.

When the starter dam to be built is long and the desired materials are available in sufficient quantity, a zoned rock, gravel, sand starter dam is superior and less costly. Any water that reaches the inner face will have free access through the outer toe of the dam, the dangers of piping will be reduced, and the phreatic water level will remain low. In some cases small amounts of water will be visible on the outer toe, but if it must be collected or treated, it can be directed into ponds or picked up in wells und pumped out for reuse or treatment. This type of dam must also be well-compacted so that it has high shear strength.

If artesian pressures are known to exist in the pond area itself, removal of water is controlled by installing a filter blanket and drains as previously described. If possible, starter dams should not be erected over or near existing artesian water. If pressures develop in the foundation or below the toe of the dam after it has been built, there is a danger of piping und instability. This can usually be controlled by the installation of pressure relief wells. The spacing, depth, design, and monitoring information for these wells is documented well by the US Army Corps of Engineers (1963) for hydro and storage dams.

Seepage through thick pervious deposits is somewhat controlled in gently sloping areas when a blanket of slimes is deposited on the ground surface as the water pool progresses upslope. There is considerable water loss at the clear water/natural soil interface at the upslope end. Cyclone overflow slimes can be deposited on the ground surface first. With the additional slimes deposited with ordinary spigoting, a very impervious blanket is eventually formed on top of the ground. When the slime layer develops a height of 3 to 4.5 m the vertical seepage rate may approach zero.

In summation, control is the key to seepage. Even in the best-built dam seepage is impossible to eliminate. To keep it at a minimum where necessary, such as where poisonous or radioactive effluent escapes, making the dam impervious is probably warranted, with controlled seepage that allows the water to be monitored, removed, and treated if necessary. Nearly all other tailings dams should have a pervious starter dam (or impervious material with drains) in which the seepage is controlled for reuse, treatment, or escape to the watershed. Many tailings dams with a clay core in the starter dam have been erected without drain provisions at the dike and operate successfully by maintenance of a stable sand beach.

## 2.2 Paste or High Concentration Transport Method

In this case, the tailings slurry is thickened in the processing plant (Fig. 6) before the paste or thickened tailings are transported to the disposal area. The thickener water overflow is retained within the plant circuit. Concerning water management, it is easier to design and operate this system than conventional systems, because retained process water is driven back to the plant in shortcut, within the plant. The operator only needs a pumping station on the plant side of the system. There is no need to pump back water to the plant from long distances.

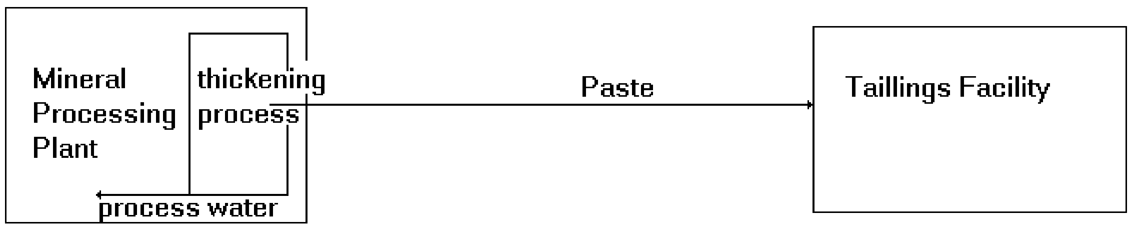



Figure 6: Schematic drawing of the high concentration transport method.

### 2.2.1 Hydraulic Transport Systems

Transport of paste and thickened tailings through pipeline systems is quite different to low concentration transport. Paste mixtures are generally Bingham plastic fluids and have a significant yield stress but have a relatively constant viscosity as flow rate increases. Viscosity can either increase or decrease with time or flow rate, depending on the characteristics of the paste. Many pastes exhibit pseudo-plastic features. Positive displacement pumps and centrifugal pumps were developed for slurry transport purposes. Applying to low-erosion materials, high lifetimes can be reached for both pump types. Positive displacement pumps provide high discharge pressures. They are generally used for long-distance hydraulic transport and for highly viscous slurries. The most common types of reciprocating slurry pumps are screw, piston, plunger and piston diaphragm pumps. Capacities range from 30 $m^3$/h to 800 $m^3$/h at pressures of up to 45-60 MPa. Horizontal paste transportation distances may be in excess of 1 km. Pipeline diameters range from 100 to 200 mm and flow velocity is less than 1 m/s while typical dilute slurry velocities are around 3 m/s. This reduces friction losses and energy consumption by nearly one order of magnitude due to proportionality to the square of velocity. Piping and fittings are also a bit different in paste transport. Due to high friction value expensive friction proof piping is necessary. Fittings and valves are also more valuable and specialised for operation of "extreme" conditions such as high friction, high yield stress, and high viscosity.

### 2.2.2 Dewatering/Thickening

Paste is a high-concentration mixture of water and fine solid particles. It has a relatively low water content (10-25%) Therefore, dewatering of the tailings slurry is usually the first step. Fine particles, “slimes”, must not be lost during the dewatering operation. Dewatering process is more or less the same as it was described for the low concentration tailings transport but an additional thickening process is added to reach ultra-high solids concentrations. These processes can be cycloning, filtration and centrifuges.

**Filtration:** Filtration is the removal of solid particles from a fluid by passing the fluid through a filtering medium on which the solids build up. Industrial filtration is analogous to laboratory filtration and differs basically in the bulk of material handled and the necessity of treating it at low cost. Many factors can be important in selecting a filtration process, but since mineral processing operations are concerned primarily with recovering the solids at large throughputs, the selection of equipment is considerably narrowed. Filtration equipment size is specified by the surface are necessary to produce the required product. As with sedimentation, particle properties can not be adequately measured, and small-scale filtration tests must be carried out to obtain basic data.

Filters can be operated in two basic modes. **Constant pressure filtration** maintains a constant pressure, so that the flow rate falls slowly from maximum at the start of the cycle. Most continuous filters can be considered to operate on this principle, using a vacuum to provide the pressure difference. (Some additional variable pressure may come from the hydraulic head of the system.)

**Constant rate filtration** requires gradually increasing pressure as the cake builds up and increases the resistance to flow. A common approach is to use a constant flow rate until the pressure builds up to a certain level, and use constant pressure filtration for the remainder of the time. This cycle can conventionally be achieved using centrifugal pumping and has the advantage of forming a rather loosely knit initial cake, which minimizes the quantity of solids forced into and through the medium. Equipment can be divided into three classes: **drums, discs**, and **horizontal filters** such as the belt type. Although substantially different in design they are all characterized by a filtration surface that moves by mechanical or pneumatic means from a point of slurry deposition under vacuum to a point of filter cake removal. Their continuous nature is somewhat deceptive because in reality these filters operate with an endless series of batch events that only approximate a continuous pattern.

A typical **drum filter** (Fig. 7) consists of a horizontal cylindrical drum that rotates while partially immersed in an open tank, the bottom of which is curved to match the drum. In this zone most filters have some means of agitating the slurry. Drum diameters vary from about 1 to 4.5 m, with filtration areas of 1-80 $m^2$. The drum shell itself consists of a series of shallow compartments about 20 mm deep covered with a drainage grid and a filter medium. The interior of each compartment is in turn connected by a conduit to a valve mechanism on the central drum shaft, which allows vacuum or pressure to be applied to the compartment at various stages of the cycle. By the action of the automatic valve on the drum shaft, vacuum is applied to the immersed sections, which results in cake build-up on the filter medium surface. As the drum revolves, the cake is raised above the liquid level and wash water, if required, is sprayed on the surface. The vacuum is maintained at this stage and beyond, when cake dewatering occurs. Before the cake can re-enter the slurry on the opposite side of the drum, some form of discharge is used. This can be achieved with a comparatively simple doctor blade close to the filter cloth. If the solids are fast settling and might not form a satisfactory cake, a drum filter can be top fed onto the ascending face of the drum. Since the cake has to be discharged at the bottom of the drum, only half the drum is effectively used.

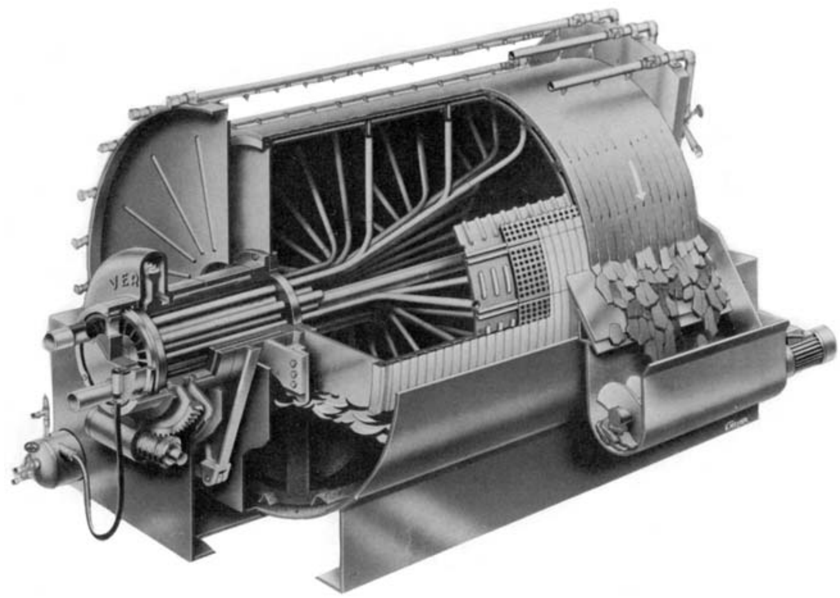


Figure 7: Schematic set-up of a drum-filter (Kelly and Spottiswood 1982).

Other variations of the drum filter are the internally fed filter and the single compartment filter. The former has the filter medium on the inside of the drum, and is well suited to heterogeneously sized materials in that the coarse particles can be made to form cake first. The latter filter has the entire inside of the drum subjected to vacuum, except for a small discharge zone. Although more expensive and inflexible in operation, it is suitable for very slow-filtering slurries.

A typical **disc filter** (Fig. 8) consists of a number of discs partly immersed in slurry, and mounted at a regular interval along a hollow shaft. Each disc is divided into segments and is ribbed on both sides to provide support for the filter medium. Again, the central shaft is connected by a set of valves to a vacuum and pressure system to allow cake formation and discharge. As the disc sections submerge during rotation, vacuum is applied to form a cake on both sides of the disc. As the segment emerges from the slurry, vacuum is maintained to provide cake dewatering, but a wash stage may be applied in between if necessary. Before the cake-carrying segments reach the slurry again, a light air blast is applied, which causes the cloth to inflate slightly and discharge the cake. If necessary the discharge may be assisted by the use of a scraper.

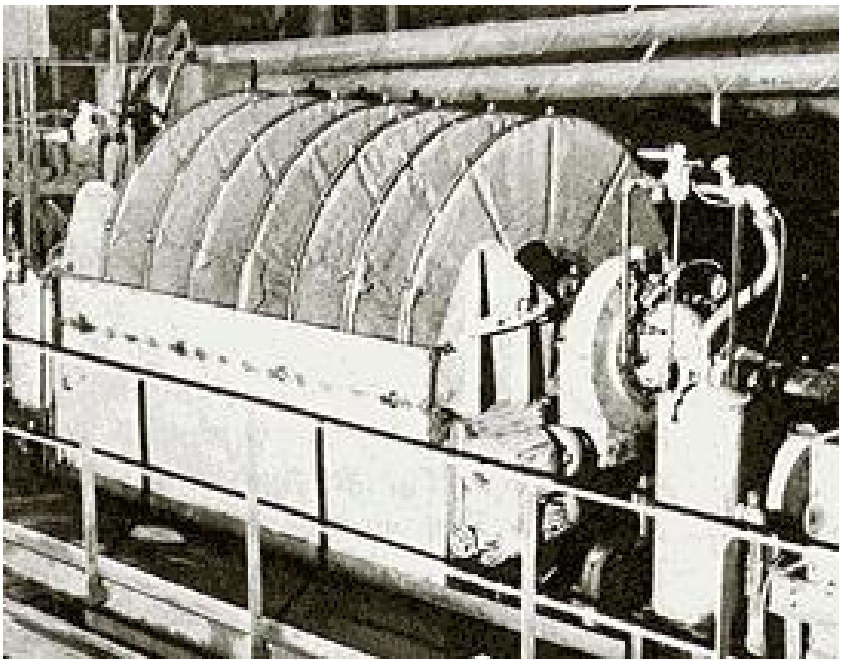


Figure 8: Disc filter (Kelly and Spottiswood 1982).

Disc filters may have 1 to 12 discs, which can be up to 5 m in diameter, thus resulting in about 30 $m^2$ of filter surface per disc. Discs may have 12 to 30 segments; the larger number provides better performance. Although one common trough is applied to all discs with an agitator to maintain the suspension, better cake formation can be achieved by having a separate trough for each disc and allowing about 10% of the slurry to overflow and be recycled. These measures ensure a continuous, even slurry flow close to the filter media. Disc filters are the cheapest and most compact of the continuous filters. Their major disadvantage is their inability to be effectively washed, but this is relatively unimportant in concentrate filtration.

The **horizontal continuous vacuum filter** is characterized by a horizontal filtering surface in the form of a belt, table or series of pans in a circular or linear arrangement. In spite of their varied forms, they have common advantages and disadvantages. The advantages are independent choice of cake formation and wash and drying times, effective filtration of heavy dense solids, effective filtration of sledges, the ability to flood wash, and adaptability to counter current washing or leaching. However, horizontal continuous vacuum filters are more expensive to install and operate than drum filters, and they require relatively large floor areas for a given filter area. In general, this restricts these filters to hydrometallurgical operations where their washing ability justifies their higher costs. The belt filter has the additional disadvantage that only half the filtering surface is effectively used, although this may be offset by the opportunity to back-wash the idle filter medium.

**Centrifuges:** Centrifuges are expensive but versatile units that can be used as classifiers, clarifiers, thickeners, and filters. In mineral processing they are used mainly for dewatering when gravitational settling rates would otherwise be too slow, or when comparatively low residual water levels are required. Two basic types of centrifuge can be distinguished: solid bowl and perforated basket (Kelly and Spottiswood 1982). Both types use a high speed (1000-6000 rpm) rotating bowl or basket that has a slurry centrifuged against its inner surface. The heavier solids settle through the liquid to the inner surface of the bowl in the machine, a helical scroll inside the bowl rotates at a slightly slower speed, and conveys the solids out of the liquid pond up to the discharge opening. Once above liquid level, further drainage of the solids occurs and washing, if required, can be carried out. Drained solids and liquids leave the machine at opposite ends. Solid bowl centrifuges are well suited for the treatment of dilute feeds and finer particle sizes.

The vibrating form of perforated basket centrifuges is the one most commonly seen in the mineral industry, generally for the dewatering of coal. Material transport through this centrifuge is achieved by vertical vibrations in the basket. These vibrations also loosen the bed of particles, aiding drainage and allowing lower speeds (550-750 rpm) than are necessary with solid bowl centrifuges. Because of the perforations in the basket, these centrifuges are not suitable for feeds having a significant proportion of fines.

There are many situations in the thickening process where it is necessary to use ***flocculants/coagulants***. Settling of tailings, filtration of solids and concentrated tailings, centrifugation of tailings and the nature of the refuse slimes, the need for clear water will rarely permit operation without addition of chemicals for flocculation. Many types are employed, both organic and inorganic, such as alum, lime, iron salts, sulphuric acid. In the organic group, the natural or modified starches have long been employed. Raw starch is sparingly soluble and quite ineffective. Many types of synthetic organic polymers are also on the market and have become prominent in coal and refuse clarification.

Three forms are available:

- **Non-ionic polymers** - exhibit neutral behaviour in solution
- **Anionic polymers** - attract positively charged ions in solution
- **Cationic polymers** - attract negatively charged ions in solution

Polymer chains act to attract the fine particles suspended in a liquid, forming larger particle groupings, called "flocs". If these flocs develop sufficient density, they will precipitate during settling, leaving behind a clear liquid. Alternatively, low density flocs may be used to separate undesirable particulates from the water and be skimmed from the surface, leaving clear water behind. The nature of the target particles, the properties of the liquid, including pH, electrical conductivity, hardness, and added chemicals, will determine the most effective polymer type to use. They are used in small amounts - polymers are added in 0.01-0.05% solutions which will usually form a better floc structure. Most polymers are purchased in either a powder or a liquid form. Alum has been used for many years as a coagulant aid to municipal waste water clarification. A gelatinous precipitate will form when it is added to slurries and this precipitate tends to collect fine non-settling solids, resulting improved settling. Alum is also used on many coal refuse waters, primarily for pH adjustment. Most refuse waters, particularly flotation tailings, will flocculate best at a pH of about 7.0. Where the water is alkaline, alum or sulphuric acid may be used. Acid waters will normally be treated with lime.

The **ultra-high-rate** and **ultra-high-density thickeners** evolved in the 1990s. They encompass a deep thickener tank and use specially designed dewatering centre cones. These features plus the cylinders rigging the outside of the feed well produce a good clear overflow. The rapid removal of water from the feed collapses the hindered settling zone of the thickener taking solids from the free settling zone into the compaction zone. The deep tank plus the 60° cone provide a deep compression zone that results in high-density underflow solids These thickeners require a high degree of instrumentation and automatic control plus variable speed controlled pumps.

### 2.2.3 Paste Deposition

Prepared and transported paste or thickened tailings, due to their heavy consistency and thus high viscosity, will flow great distances without segregation. Eventually the flow stops at a gentle slope. The slope is controlled by the degree of thickening.

Significant benefits can be gained from depositing tailings into an above ground storage area. The formation of steeper planar beach slopes permits tailings to be stacked above ground rather than needing to be stacked in an impoundment. Discharge of the paste can be through single or multiple point discharge. Single point discharge will create a low conical hill. Multiple point discharge systems are good for creating different configurations or overlapping cones. Tailings disposal areas may consist of valleys or flat terrain somewhere in the vicinity of the processing plant. To form a sloping tailings deposit in a valley, the thickened tailings would be discharged at the head of the valley or along one of the side hills. The heavy slurry will flow down the valley until it encounters a slope flatter than its own, or alternatively, until it is stopped by a small dam. On flat terrain, thickened tailings would be discharged from an artificial ramp or tower. It is also possible to discharge high concentration tailings or paste into conventional embankments. But in this case, some advantages of segregation will be lost. Paste can also be released into mined open pits. Thickened tailings and paste are good for underground mine backfill purposes. For this application, physical properties of the backfill material can be engineered according to the goal of the backfilling.

### 2.2.4 Costs And Financial Risk (Jewell et al. 2002)

Some economic benefits and costs of the Paste and Thickened Tailings (P&TT) technology are easily quantified. Social and environmental benefits may be more difficult to quantify but can be valuable nonetheless. For example, Alcoa (1999) introduced dry stacking of very high density tailings between 1986 and 1992 across its three Western Australian aluminium refineries, which handle 13 Mt per annum of residue. To fully evaluate the benefits of P&TT technology, an economic study based on "full life-cycle costs" is needed. Capital costs; operating costs, timing and the time value of money need to be applied across the full spectrum of the mine plan, including closure. Often complete mine and/or tailings plants need to be run to closure to truly understand the benefits of this technology. For example, paste tailings incorporated into mine backfill can lead to cost savings across the whole operation despite the extra costs of tailings thickening facilities. Non-monetary benefits like improved public perception should also be evaluated, using appropriate qualitative measures. Savings are possible in water, energy and reagent conservation, reduced impoundment needs, more rapid closure, reduced financial provisions and backfill savings. Additional capital and operating expenses for thickening equipment, pumps and piping may partially offset these savings.

#### 2.2.4.1 Capital Costs

The Paste and Thickened Tailings (P&TT) technology can significantly reduce the cost of containment structures but will require some offsetting expenditure on thickening equipment:

***Tailings Containment***

Reductions in capital cost will result from the reduced need for engineered embankments around tailings disposal facilities, particularly in the early years of project development. Further capital and operating savings accrue from the reduced need for underdrains, toe drains and recovery bores to protect ground and surface waters from contaminated leachate. In most mines with conventional tailings, tailings must be contained by high specification engineered embankments: These embankments are required to contain the slurry pumped from the plant and the sand, slimes and decant process water that separate from it: Engineered drainage or liquor recovery systems that add to the costs are also often required. The embankments are usually constructed before production commences and consequently represent a large up-front capital cost. Since high-density tailings can be discharged as a cone or stacked, they do not need high specification embankments comparable to those required for "conventional" tailings; hence substantial savings can be realised through reduced embankment volume for comparable stack heights. The ability to use lower specification material for embankment building and reductions in the cost of placement is also benefits. In most cases, high density thickened tailings will be centre discharged, thereby resulting in a "cone" structure rather than the conventional "dish" structure. The "dish" structure requires more containment than a "cone" and consequently more embankment material; resulting in extra operating and/or capital costs. The “cone" has its lowest point at the outside of the structure and hence only requires a small wall for the toe of the deposited material and water redirection: In comparison the "dish" has its highest point at the outside of the structure, requiring further engineering to contain and manage water that falls on the structure. However, the central thickened discharge (CTD) technique will require a greater footprint than stacked tailings and conventional TSFs: The Peak Hill Gold Mine at Cobar in New South Wales, Australia, is an underground mine producing 300,000 tonnes per annum of fine tailings. Thickening and CTD discharge has allowed the tailings to be deposited without the need for extensive dam construction: This eliminated the need for large borrow areas for construction materials, signifycantly reducing the environmental impact. This method also leads to considerable economies in initial and ongoing construction costs: Operating costs have also been reduced because there is no requirement to keep rotating the spigot outlets or to raise and relocate the tailings delivery line (EPA 1995).

***Thickening Pumping and Piping Costs***

Additional capital costs for operations adopting P&TT technology stem from the equipment necessary to thicken and transport the tailings, such as thickening equipment, the flocculent addition system, pumps and pipelines.

Thickening equipment will usually consist of thickeners and/or filters. If filters are used, a re-slurry system is usually needed to reduce the consistency of the filter cake to a pumpable level. The cost of the thickeners and pumps required to produce and move the product can run from a few hundred thousand dollars to millions of dollars depending on size and application: Placement of P&TT will usually require less piping than conventional tailings: For example, central discharge of thickened tailings eliminates the need for a long ring main and spigots. However, transport of high-density slurries and pastes generally involves higher pipeline pressures and hence more expensive pressure rated pipes and fittings.

The pumping requirements may also require the installation of positive displacement pumps that are significantly more expensive than conventional centrifugal pumps. Additional operating costs for thickened tailings stem mainly from flocculation reagent, thickener/cyclone/ filter maintenance and piping and pumping operating costs. Where filters are used, the cost of transportation of the dry material may be a factor. Of these costs, the flocculation costs can be the most unpredictable and onerous.

#### 2.2.4.2 Water Conservation

In areas where water is expensive or scarce, the water savings can more than pay for the cost of thickening tailings. This is particularly relevant if operations are disrupted by water shortages. There are mines in arid areas or areas of restricted water access as in Australia that have lost production due to a lack of water in dry or low rainfall years. In other areas, companies may have to pay for water or pump from extreme distances, incurring large capital and operating costs. For example, the Murrin lateritic nickel mine near Leonora in Western Australia is examining the feasibility of pumping process water more than 400 km from the Officer Basin to the northeast. Recirculating process water in such situations provides savings in both capital and operating costs. The capital costs not only involve pipe and pump costs, but may also involve the purchase of water rights, establishment of bore fields, access rights to land and government usage or license fees. Thickened tailings can be produced and transported at a consistency containing little excess water. In this situation, there is no free water to form a decant pond at the end of a beach and the only recirculation pumping system needed (if at all) is for precipitation run-off and consolidation release water. In underground mines, the lack of bleed water results in savings associated with not having to collect it and return it to the surface.

#### 2.2.4.3 Energy Conservation

In operations where process water temperature is required to be above ambient, energy savings can be realised if tailings are thickened in the plant and the water is recirculated without being sent to the tailings pond where it cools. In industries located in colder climates that require above ambient process temperature (such as oil sands), these energy savings can be substantial. Any reduction in beating or pumping requirements will also clearly reduce energy usage. These savings may be partially offset by the increased energy needed to operate dewatering equipment.

#### 2.2.4.4 Reclamation/Closure Costs

P&T'T will usually provide a stable consolidated surface that can be landscaped earlier and more easily than conventional tailings. Consequently, reclamation and closure is usually less expensive than for conventional tailings. For example, at the Macrae's gold mine in New Zealand conditions attached to mine approval required the position of tailings to cease after 15 years of mining, with the monitoring of leachate treatment and site rehabilitation to then continue for a further 20 years. Financial risks associated with conventional tailings disposal can be large and enduring. While early reclamation/rehabilitation may be seen as negative because of the time value of money, early rehabilitation can, however, facilitate the treatment of expenditures as an operating cost while there is still a cash flow coming from the project. Early rehabilitation can also close out contingent liabilities and associated provisioning and insurance costs.

#### 2.2.4.5 Reduced Financial Provisions and Insurance Costs

Conventional tailings management facilities often leak through the embankments or the base as a result of ongoing consolidation. Enabling the transport of contaminants into ground or surface waters such leakage can continue over many decades, requiring the allocation of financial provisions to cover this contingency, as well as ongoing insurance cover to offset future claims for damage. Thickened tailings contain less excess water and consolidate quickly. Hence, they do not have the same tendency to leak and can be rehabilitated early, mitigating the need for long-term financial provisions and insurance. P&TT technology also offers the potential for improved long term predictability of deposit performance, which facilitates early planning with a degree of confidence for decommissioning: This affords the opportunity to more accurately allocate financial provisions for rehabilitation and decom-missioning with greater certainty such that there will not be an unpredictably long period of deposit rectification and leachate recovery after closure. Surface rehabilitation can be com-pleted quickly and with a high degree of confidence that the surface will not deteriorate due to ongoing consolidation, capillary rise of contaminants, slumping or erosion.

#### 2.2.4.6 Time Value of Money

When considering costs, the time or net present value (NPV) of money also needs to be taken into account: Since construction of tailings embankments is minimised, this saving is accrued prior to and/or in the early years of the mine. On the other hand, thickeners, filters and auxiliary equipment must be installed before mining commences and result in up-front capital costs that reduce savings made on the TSF. Where reclamation/rehabilitation and closure are accelerated, the impact may be negative by bringing forward expenditures that would normally be discounted out into the future. The main uncertainty inherent in taking a NPV approach is that by the time remedial measures are contemplated, the deposit may well not be amenable to relatively low cost rehabilitation and a much more complex, time consuming and costly process may have to be adopted.

### 2.2.5 Process Integration

Where P&TT technologies are introduced into a mining operation the process must be integrated into the total mineral processing and tailings management scheme. This integration involves establishing a process base for four functions:

- The total mine water balance.
- The volumetric balance of the mine and tailings plans.
- The chemistry of the circulating water as it passes from makeup source(s) through mineral processing, tailings processing and tailings water recovery.
- The thermal energy balance where mineral processing is carried out at elevated temperatures.

The key considerations for each function are set out below.

#### 2.2.5.1 Total Mine Water Balance

Considerations:

- Loss to groundwater.
- Evaporation and precipitation balance.
- Residual water content of the tailings deposits.
- Surface drainage and water recapture management.

#### 2.2.5.2 Mine and Tailings Volumetric Balance

Considerations:

- Minimum design angle of repose and material strength.
- Space requirements for in-pit operations.
- Timing for in-pit tailings/volume requirements for out-of-pit disposal.
- Consolidation rates and ultimate chemistry of tailings.

#### 2.2.5.3 Circulating Water Chemistry:

Water chemistry (dissolved solids, pH) will be influenced by factors present in each of the process steps:

- Source water for makeup from surface sources (rivers, lakes), groundwater or surficial aquifers.
- Mineral extraction – interaction between the ore and any chemicals used in processing.
- Tailings treatment chemicals.
- Total water balance – concentration effects resulting from recycling and evaporation.
- Impact of dissolved solids on the thickening process.

#### 2.2.5.4 Thermal Energy Balance

Where the mineral process operations are conducted at temperatures greater than ambient, thermal energy must be supplied by heating the process water. For these operations, the thickened tailings process may assist in energy efficiency by returning heated process water to the operations from the thickeners. Considerations:

- Density of thickened tailings.
- Use of cold dilution water.

## 2.3 Low Concentration Transport Method Combined with Paste Thickening at the Discharge Area

The third case combines the first two cases. First, there is hydraulic transport of low or not too high concentration slurries to the disposal site. The thickening is then conducted at the disposal site, before disposing the so produced thickened tailings (Fig. 9). Return water can be recycled. Advantage: reducing energy and capital costs of case 2 by low concentration transport system.

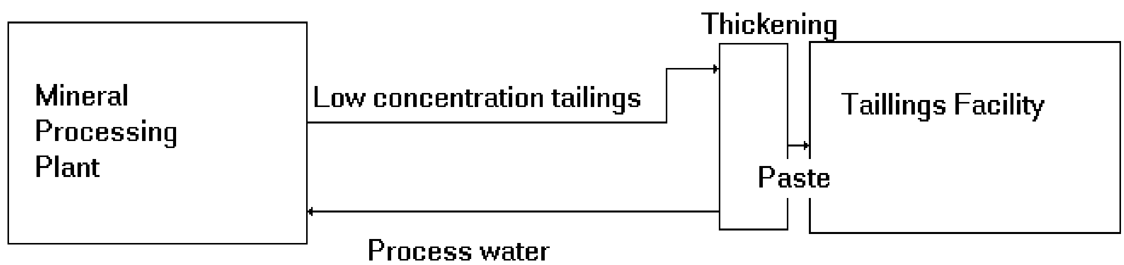



Figure 9: Schematic drawing of the combined method.

# 3 HYDRAULIC TRANSPORT INVESTIGATIONS

One of the targets of the Miskolc University group within the TAILSAFE project was to establish those parameters that govern the hydraulic transport of thickened tailings at the highest possible solids concentration. For the experimental work, one main test material was chosen and characterised. Additionally other selected materials were also tested for comparison. The main test material was flotation tailings from Gyöngyösoroszi, Hungary. The Gyöngyösoroszi tailings facility was one of the TAILSAFE case study sites. This material was chosen because it was available at large amounts for the investigations, and it is a typical tailings material with properties that could meet paste technology requirements.

Figure 10: Sampling of the main test material at the Gyöngyösoroszi site.

The material deposited in the Gyöngyösoroszi tailings pond is comprised of tailings after flotation technology. Especially in the upper level of the tailings body also other waste materials can be found. The Gyöngyösoroszi tailings facility was built by the well known and often used cycloning technique, where the embankment was built from coarse material separated by the cyclone while the fines were washed into the lagoon to settle down. Samples were taken by the Miskolc University research group from boreholes drilled by Mecsek-Öko Rt., Pécs, current owner of the site. The sampling points were located at different places, covering the whole tailings facility. For this reason, the sampled test material was not homogenous. Depending on the place where they were taken (from the former embankment, from settled material or from nearby the open water surface) the samples differed in appearance and characteristics. Sampling is shown in Fig. 10.

In the lab the samples were mixed and a homogenised composite sample was produced to be used for the experiments. Several laboratories of the University of Miskolc were involved in examining the composite sample material to determine the relevant material properties.

## 3.1 Basic Material Properties

Basic material properties have to be analysed to determine the connection between those material properties and hydraulic transportability and rheology of mixtures of the materials with water. It has to be examined also if the test material is actually suitable for paste production. The following material properties were analysed:

1) Particle size distribution
2) Consistency parameters
3) Density
4) Rheology
5) Column settling tests
6) Slump cone test
7) Mineralogy

### 3.1.1 Particle Size Distribution

The particle size distribution defines the relative proportions of particles of different sizes in a mass. In paste technology it has a relatively high importance, because small sized particles (below 20 µm) have a determinative role in paste production. The particle size distribution of the test materials was analysed by sieve analysis and laser diffraction analyser, and the data were confirmed by microscopic analysis (Fig. 11).

In the Gyöngyösoroszi tailings, the fraction of particles with a size of more than 100 µm is low (Fig. 12), so it can be said that the maximum particle size is around 100 µm. The fraction of particles with a size of less than 20 µm is about 75% and of particles below 2 µm about 22%. The slime content is about 52%.


Figure 11: Picture of 40-63 mm fraction of Gyöngyösoroszi tailings through microscope.

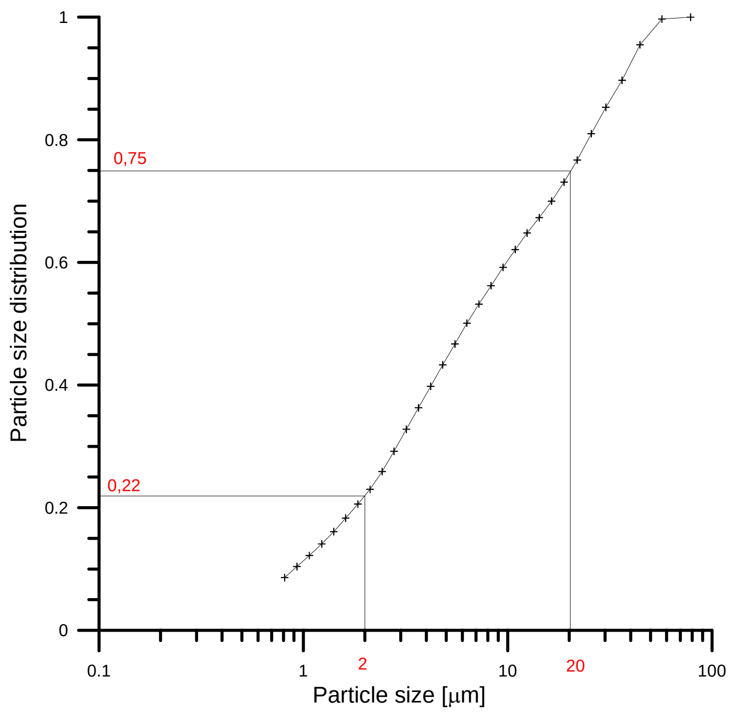



Figure 12: Particle size distribution function of the Gyöngyösoroszi flotation tailings material.

### 3.1.2 Plasticity Parameters

This particle size distribution of flotation tailings material allowed us to determine the plasticity parameters of the samples:

Liquid limit $W_l = 37.5\%$
Plastic limit $W_p = 21.7\%$
Plasticity index $I_p = W_l - W_p = 15.8\%$

The value of plasticity index corresponds to the value of a lean clay (15-20%), but it is close to silt values (10-15%; Table 1). Plasticity properties and particle size distribution data show very similar results.

Table 1: Soil classification based on the plasticity index.

| $I_p$ [%] | Name |
|---|---|
| 10 – 15 | Silt |
| 15 – 20 | Lean clay |
| 20 – 30 | Clay |
| > 30 | Fat clay |

### 3.1.3 Particle Density

The particle density of the test material from Gyöngyösoroszi was determined as 2701 kg/m$^3$. This data was necessary for concentration calculation of the transport and rheology experiments.

### 3.1.4 Rheology

The rotational viscosimeter and the capillary viscosimeter are the two most important devices for measuring rheological properties of fluids. However, the application of both of them for investigating slurries is limited. In rotational viscometers, particles settle to the bottom of the device, causing unfeasible measurements. In capillary viscosimeters, the size even of very small particles can be comparable to the diameter of the tube. Therefore, for the present investigations a tube viscosimeter was used (Fig. 13) which is actually a big capillary viscosimeter with a large tube diameter. The principle of the measurement is that the mixed slurry is circulated through a pipeline by a screw pump, at low flow rates to enable laminar flow. Pseudo shear curves can be obtained by measuring the pressure loss and flow rates under known geometric and physical properties. Flow parameters and rheology of the fine suspension can be determined using such pseudo shear diagrams. Every investigation was started at $c_v = 0.1$ volumetric transport concentration, going up to the maximum concentration which could still be transported by the pump. At least 6 different flow rates were measured, controlled by a frequency controller from 60 Hz to 15 Hz. Higher flow rates were applied after every lower flow rate between two measurements to avoid settling of particles and obstruction of the system. Pseudo shear diagrams could be obtained during the experiments. Flow properties of the slurries have also been determined at every measured volumetric concentration.

The flow properties of the Gyöngyösoroszi tailings material showed typical behaviour of slurries: Newtonian at low, mostly under $c_v = 0.2$ concentration, becoming Bingham-plastic radically at concentrations above that value (Fig. 14).

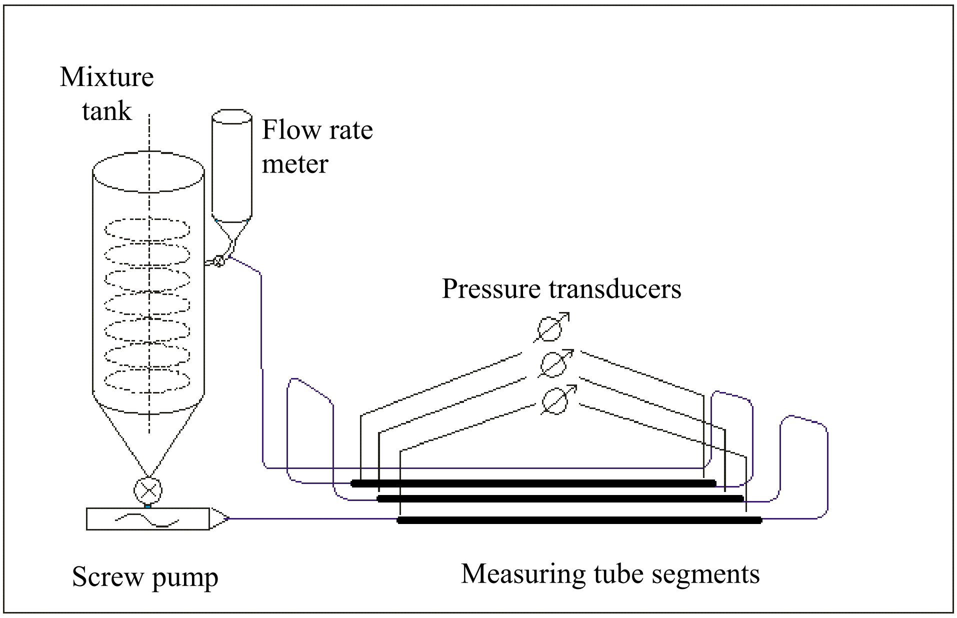



Figure 13: Schematic drawing of tube viscosimeter.

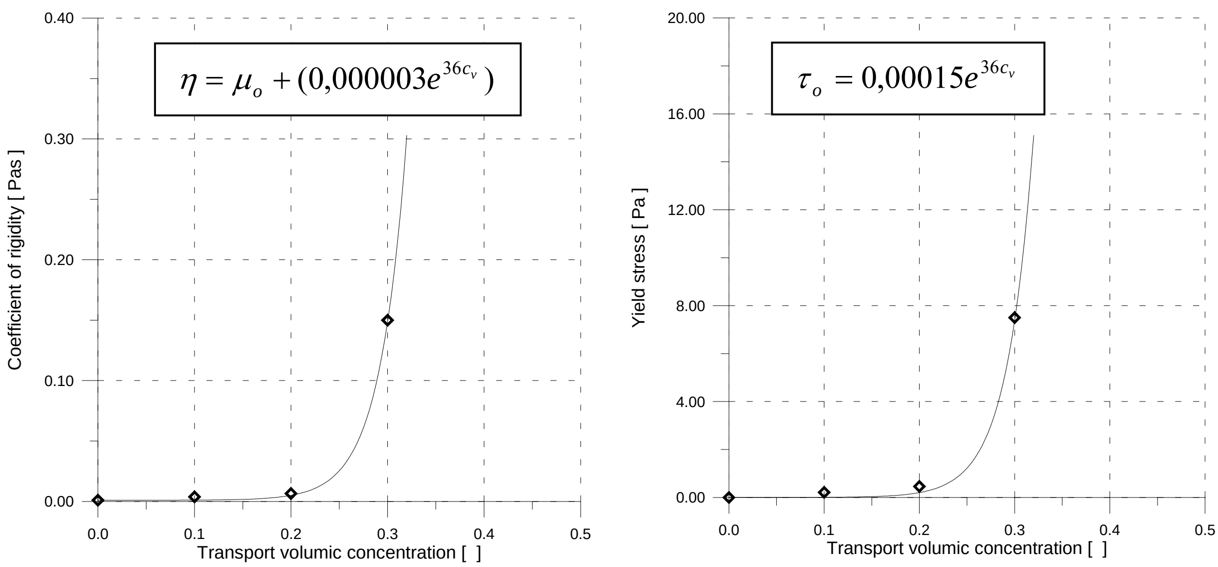



**Figure 14**: Shaping of coefficient of rigidity and yield stress of Gyöngyösoroszi tailings as a function of volumetric transport concentration.

The results show that the rheology of many material/water mixtures can be described by the same function with different parameters as:

$$\eta[Pas] = \mu_o\left(1 + K_1 c_v + K_2 c_v{}^{K_3}\right) \qquad \text{(a)}$$

and

$$\tau_o[Pa] = K_4 e^{K_5 c_v} \qquad \text{(b)}$$

where:

| | |
|---|---|
| η[Pas] | = coefficient of rigidity |
| $\mu_0$ [Pas] | = dynamic viscosity of water |
| $\tau_0$ [Pa] | = yield stress |
| $c_v$ | = transport volumetric concentration of solids |
| $K_1 \ldots K_5$ | = constants |

### 3.1.5 Column Settling Tests

The column settling test is applicable for determining the stability of a slurry during transportation and after deposition. This type of test is also suitable to predict the maximum concentration which can be achieved with the analysed material.

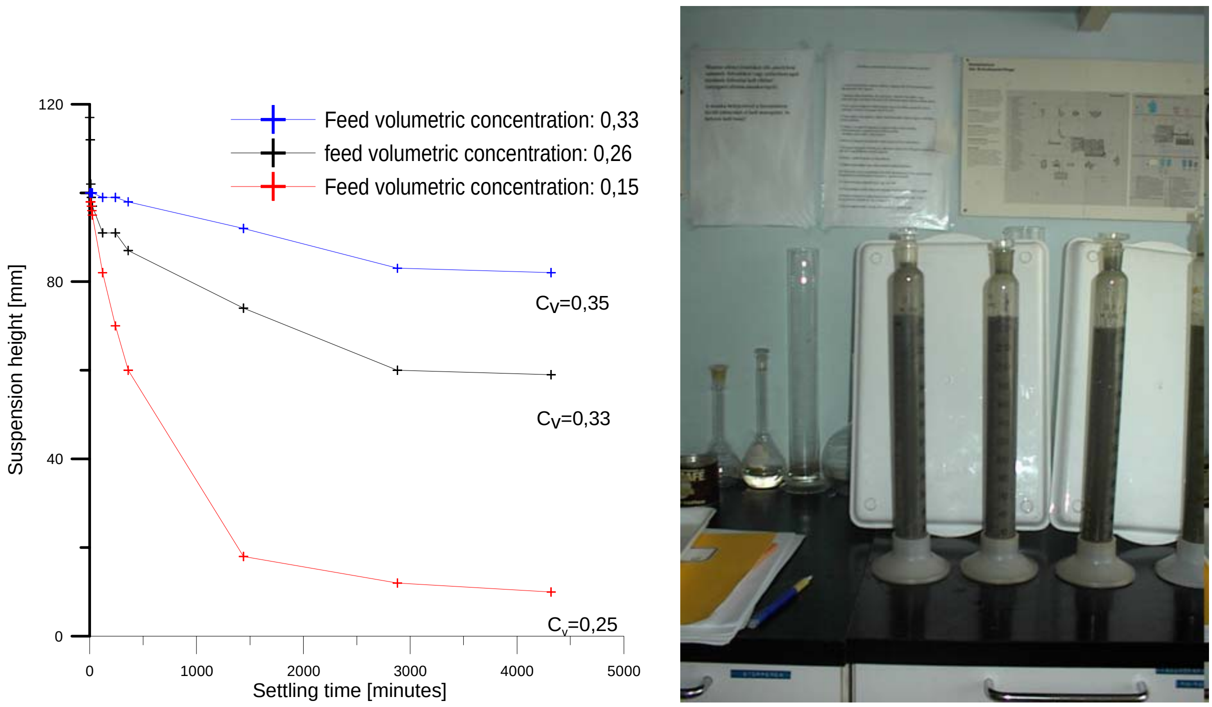



Figure 15: Column settling tests of Gyöngyösoroszi tailings material (without flocculating reagent).

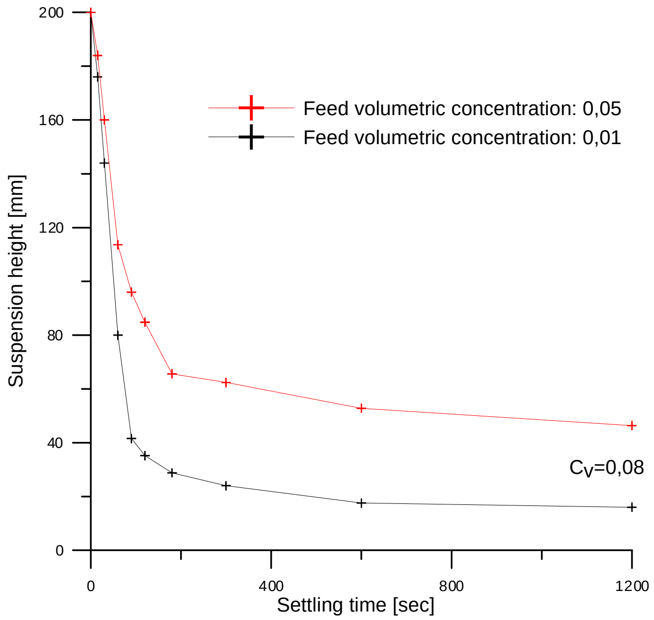



Figure 16: Column settling tests with polyelectrolyte flocculating reagents.

Settling tests of Gyöngyösoroszi tailings material (Fig. 15) showed that the final concentration was reached after 50-60 hours at the maximum of 25-32% by volume. This means that the final concentration was dependent of the initial concentration.

Settling tests with Gyöngyösoroszi tailings material were also carried out with polyelectrolyte flocculent reagents. The results (Fig. 16) show that the maximum solid concentration that could be reached was around 10% by volume. This low value results from the fact that with higher initial concentration there was no space for flocs to start up. Higher initial concentrations yielded no significant effect on the final concentrations.

### 3.1.6 Slump Cone Tests

The rheology of a paste is controlled by a number of different variables. The most determinative properties are the rate of particles with a size below 20 µm, surface properties of the particles and properties of the carrier liquid. There are many minor variables as well, therefore instead of prediction, measuring is crucial for every different material. A common means to measure rheological properties of a paste is the ASTM slump cone test, which uses a truncated cone 305 mm (12 inches) in height. The cone is filled with paste and then removed allowing the contents to assume a pile shape with a natural slope (Fig. 17). The distance of the top of the pile from 305 mm is the slump. A slump measurement reflects the yield stress of a paste. Dense paste would have low slump of 50-200 mm (2-8 inches). Materials at the transition between a slurry and paste would have high slump, up to a maximum of 305 mm. It was determined by the slump cone test that at least 40% solid concentration by volume is needed to reach paste state for Gyöngyösoroszi flotation tailings material.

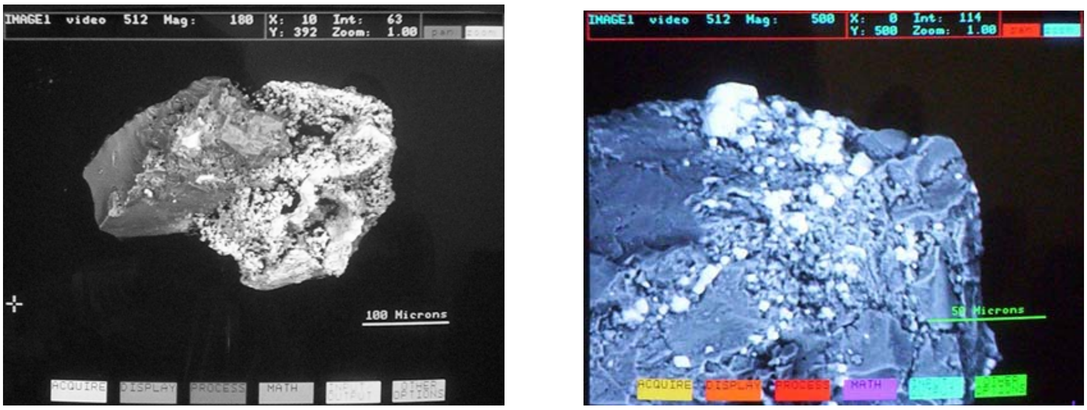
Figure 17: Slump cone test of Gyöngyösoroszi flotation tailings at solid concentration of 50% (V/V).

### 3.1.7 Mineralogy

The mineralogy of the tailings test material was of great importance to be analysed because it was supposed that the clay mineral content has a significant effect on the thickening and hydraulic transport behaviour of the pastes. The mineralogy of the Gyöngyösoroszi tailings material was determined by X-ray diffraction analysis. The results are shown in Table 2. The results were confirmed by SEM and microprobe analysis (Fig. 18).

Table 2: Mineralogy of the Gyöngyösoroszi tailings material.

| | [%] |
|---|---|
| Clay content (montmorillonite, illite, kaolinite) | **14** |
| Chlorite | **2.75** |
| Quartz | **54.75** |
| Potassium feldspar | **5.5** |
| Plagioclase | **3.5** |
| Calcite | **11.25** |
| Pyrite | **4.5** |
| Gypsum | **0.25** |
| Amorphous | **3.5** |




Picture 18: SEM images of Gyöngyösoroszi tailings material.

### 3.1.8 Other Materials Tested

According to the slump cone tests, the Gyöngyösoroszi tailings material is able to reach the paste state. To allow a correlation between material properties and paste behaviour, different model materials were also tested. The analysed materials are listed below. Paste state was not achieved with any of these materials. The reasons for this behaviour are connected to the different mineralogical and physicochemical properties of these materials. The following materials were also tested:

a) Power Plant Fly Ash Type I from India (PPFA t1)
b) Power Plant Fly Ash Type II from Dorog, Hungary (PPFA t2)
c) Quartz sand
d) PPFA t2 / bentonite mixture at 5% and 10% by mass
e) Quartz sand / bentonite mixture at 5% and 10% by mass

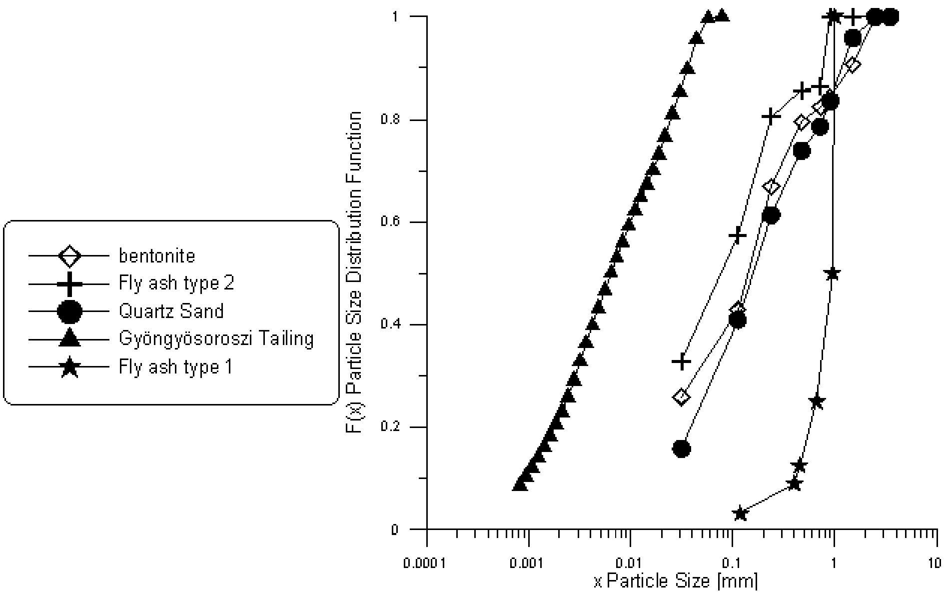



Figure 19: Particle size distribution of the materials.

As shown in Figure 19 the **particle sizes** of the alternative test materials are much coarser than those of the Gyöngyösoroszi material. The most significant difference can be observed for the Type I fly ash.

The **particle densities** of the tested materials are shown in Table 3. The determination of these data was necessary for concentration calculations in the transport and rheology experiments.

Table 3: Particle densities of test materials.

| Material | Density [kg/m$^3$] |
|---|---|
| PPFA t1 | 2492 |
| PPFA t2 | 2355 |
| Quartz sand | 2720 |
| Bentonite | 2574 |

The **rheology** of slurries of the model test materials has been examined at every measured volumetric concentration. The flow properties showed typical behavior of slurries. Newtonian at low, mostly under $c_v = 0.2$ concentration, becoming Bingham-plastic radically at concentrations above that value (Figs. 20 and 21).

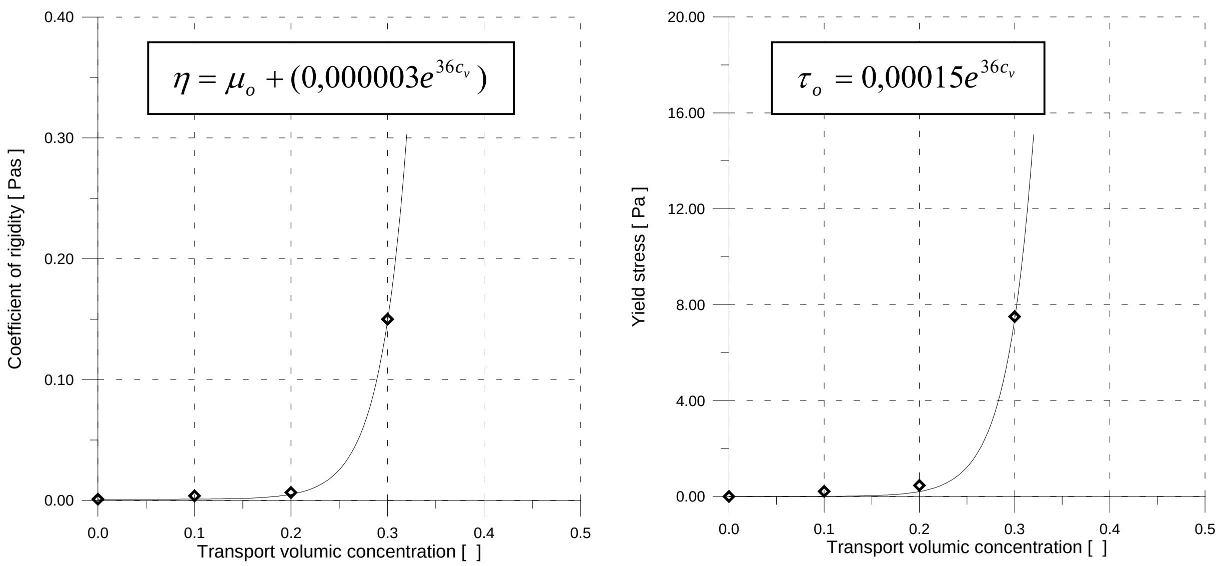



Figure 20: Shaping of coefficient of rigidity and yield stress of fly ash (Type 2) as a function of volumetric transport concentration.

Suspensions of fine fly ash (Type 2) powder in water behave as a Bingham-plastic fluid at lower concentrations (Fig. 20). However, at concentrations lower than 20% volumetric solids content they exhibit only low viscosity, with a coefficient of rigidity only slightly higher than for the viscosity of water (1 mPas). At higher concentrations, the coefficient of rigidity then increases significantly. The coefficient of rigidity is 150 mPas at a volumetric concentration of 30%. The experiments showed that the suspensions behave as a thixotropic fluid at concentrations of 30% and higher.

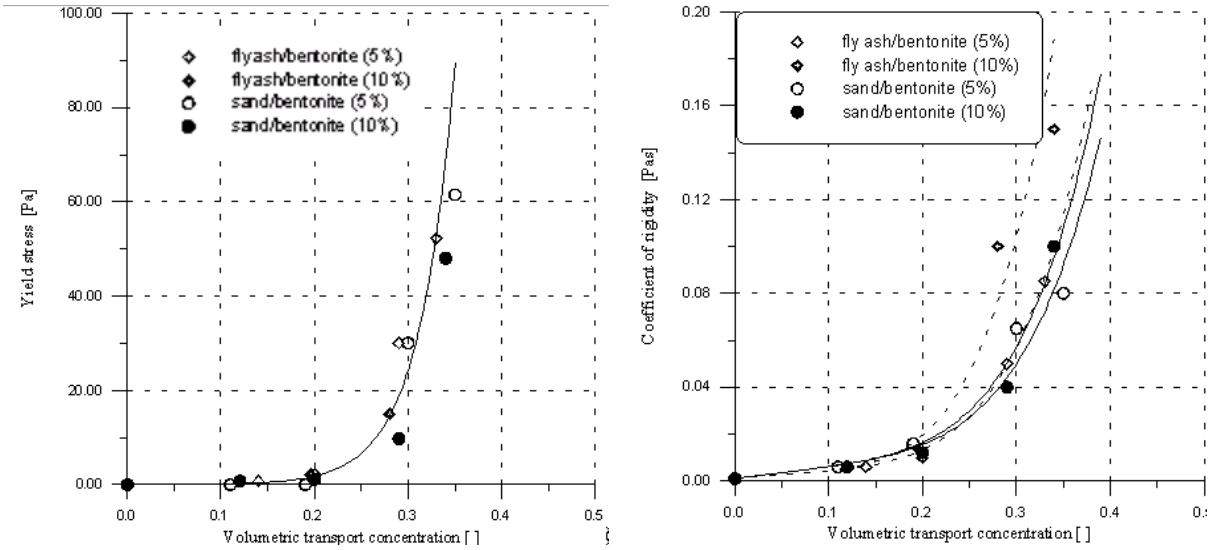



Figure 21: Shaping of coefficient of rigidity and yield stress both of fly ash (Type 1)/bentonite and sand/bentonite mixtures as a function of volumetric transport concentration.

Rheology experiments were also carried out on other mixtures. The results are summarized in Figure 21 and show that all the investigated suspensions behave as a Bingham-plastic fluid, except the mixture of sand/bentonite (5%) at concentrations lower than 20%. Mixtures containing higher amounts of bentonite increase the coefficient of rigidity faster. Function fitting has been done for the measured points, and the following function were received for the coefficient of rigidity:

$$\eta = \mu_o \left(1 + K_1 C + K_2 C^5\right)$$

The values of the constants of the mixtures are:

| | $K_1$ | $K_2$ |
|---|---|---|
| Fly ash/bentonite (5%) - water | 30 | 19500 |
| Fly ash/bentonite (10%) - water | 30 | 39000 |
| Sand/bentonite (5%) - water | 50 | 14000 |
| Sand/bentonite (10%) - water | 50 | 17000 |

The yield stress ($\tau_0$ [Pa]) shows no significant difference between the four measured mixtures, therefore only one function has been fitted.

The investigation showed that the flow properties of most type of Bingham-plastic slurries can be determined by a function like function (a) and (b). It has to be clarified how constants and the exponents depend on the material properties, and if there is a dependence, what kind of material properties are dominating the general functions.

Another important fact is that these functions are applicable only within the concentration interval of the measurements, where solid particles had not reached compacted state. More experiments are needed to determine the type of function that can describe the paste flow properties over a wider range of concentrations.

The settling tests showed that the alternative mixtures (d. and e. – with bentonite) could reach a maximum concentration of 35% by volume. The materials with lower initial solid concentrations reached a close-to-final concentration after 50 hours. The fly ash from India settled faster and reached the maximum concentration of 52% by volume after 7 hours independently from the initial concentration. The reason might be the coarser particle sizes of the fly ash and the lack of clay minerals. Again it is important to note that these materials could not reach the paste state in the face of high solids concentrations.

## 3.2 Hydraulic Test System

When hydraulic transport of solids was first time used, operation safety was the most important factor. This means that mainly low concentrated slurries were transported through the pipelines. Although this procedure was relatively simple and failure proof, it was expensive because of the high amounts of water that had to be transported too. The need evolved to develop other, more economic and environment-friendly transport methods. The overall objective of the development activities is to increase the solids concentrations of the slurries and to determine the concentration that can still be transported at a given flow rate without settling of particles. Advantage of the use of high-density slurries is that smaller dams are sufficient. The load on the embankment is also smaller which means less risk to failure. There is little or no water which has to be treated, and the deposited material cannot be liquefied. In case of a failure it spreads only within a limited area causing less damage than conventional tailings.

To study the hydraulic transport of high-density tailings slurries (pastes), an experimental system was built at the University of Miskolc to conduct pilot-scale experiments for the determination of the technical parameters of paste transport. The scheme of the pilot-scale experimental set-up can be seen in Figure 22. The equipment was designed to allow the use of different types of pumps. The system has a mixing tank with a volume of 400 litres (see also [1] in Figure 26) and a built-in mixer driven by a hydraulic engine. Continuous mixing helped to reach the right homogenisation state. Homogenisation was more important at low concentrations than at higher concentrations since homogeneity of a slurry in paste state is permanent even without mixing.

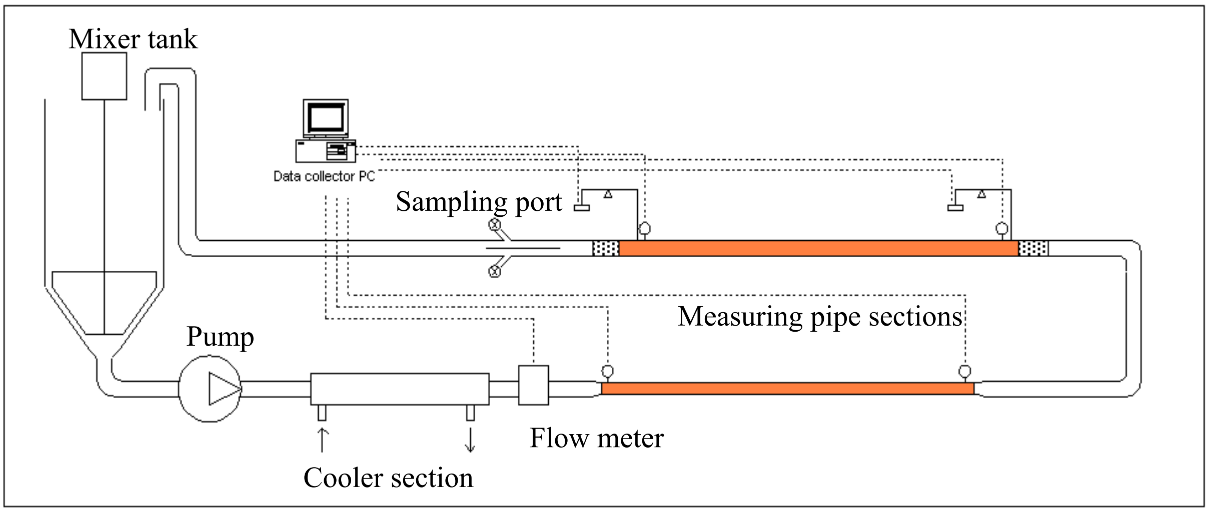



Figure 22: Schematic of the hydraulic transport test system.

Three types of pumps were used for the investigations. Figure 23 shows the centrifugal pump used in the experiments (see [2] in Figure 26). The hydraulic transport tests were initially commenced using this pump type, to determine the maximum volumetric concentration that can be achieved using centrifugal pumps. The centrifugal pump used was a WARMAN type slurry pump, where the pump case and the impeller are coated with rubber.

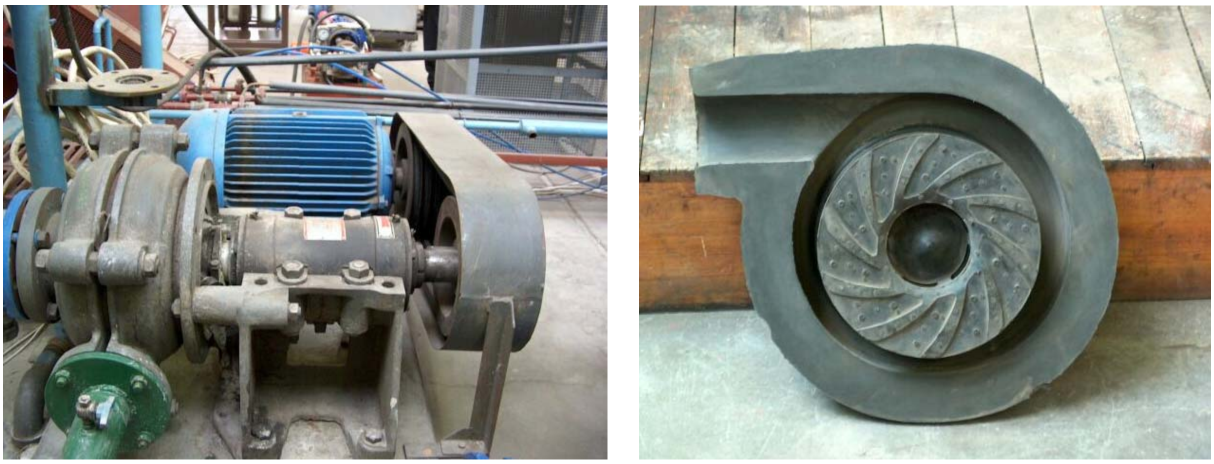


Figure 23: WARMAN type centrifugal pump.

The second pump used was a screw pump which enabled us to transport much higher volumetric concentration slurries than with the centrifugal pump because of its strict characte-ristics. The screw pump was a British made, MONO type pump (Fig. 24) with a 15 $m^3$/hour flow rate and with the ability to produce a 15 bar pressure head.

Figure 24: Mono type screw pump.

A piston and membrane pump was also available for the experiments (Fig. 25). The piston and membrane pump used was a German made, ABEL type special slurry pump with a 11 $m^3$/hour flow rate and with the ability to build up a 15 bar pressure head. Advantage of this pump type is that during slurry transport the abrasive material does not come into direct contact with the piston or its sealing. The transported material and the piston are separated by a membrane, and the piston is only contacting an oil and water emulsion agent. The second sensitive part of the slurry transport piston pump is the valve. Ball valves are used in this type of pump. The diameter of the valve balls was approximately 100 mm and the balls were coated with an elastic, wear-resistant material.

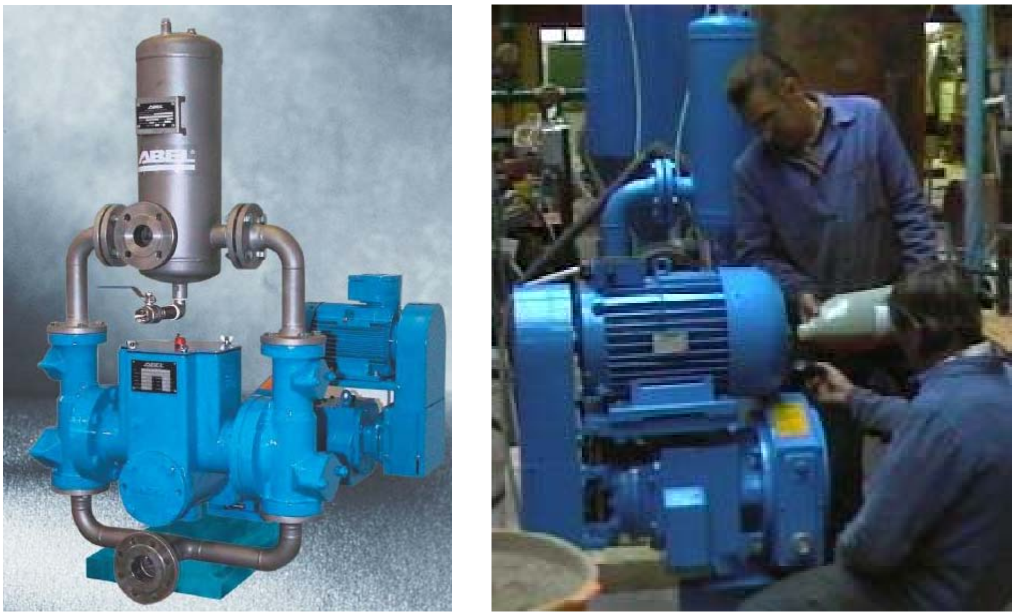



Figure 25: Abel type piston and membrane pump.

Right after the pump outflow a cooling section was built to keep the temperature of the slurry at a constant value. After the cooling section, an induction flow meter was installed (see [3] in Figure 26).

The system contains two measurement sections (Figure 22). The first measurement section comprises a 6 meters long 6/4” (40 mm) pipe (see [4] in Figure 26), where the pressure loss of transported slurry over a given pipe section can be measured. A Hottinger-Baldwin-type differential pressure meter was used to measure the pressure differences. After this first measuring section, the pipeline is turning back and a second measuring section follows, consisting of a 6 meters long 2” (50 mm) pipe (see [5] in Figure 26). Pressure loss is measured again here as the difference between the two measured pressure values at the beginning and the end of the pipe section, and additionally the weight of the pipe section is measured by a scale system (see [6] in Figure 26) to determine the solids concentration of the slurries. Following the measuring sections a sampling port allows sampling of the transported slurry from the bottom and the top of the pipe to determine any concentration differences within the pipe. All measuring devices are connected to a PC where the experimental data are stored and processed.

## 3.3 Experiments

The volumetric solids concentration in the slurries was increased step by step during each experiment. All experiments were started at low solids concentrations which were gradually increased, if possible up to the paste state. Transport properties were measured at different flow rates after a concentration was set.

### 3.3.1 Properties Measured in the Experiments

- **Flow rate**
  Flow rates were measured using an induction flow meter. Transport velocity in the test pipe sections can be calculated when the flow rate and the diameter of the pipe is known.
- **Volumetric concentration**
  Volumetric concentrations were adjusted by the amount of solid material mixed into water. This value was just an approximate value because of continuous sampling and the intrinsic moisture content of the solid materials. Actual volumetric concentrations were measured using continuous measurement of the weight of a pipe section where the volume of the pipe section is known. If the density of the transporting fluid (water), the density of the solid particles and the weight and volume of the slurry are known, the volumetric concentration can be calculated.
- **Hydraulic losses (pressure drop)**
  The flow resistance of a pipe section whose diameter and length are known has been determined by measuring pressure differences. By this way the changes of resistance of

different pipe sections could be determined as a function of transport velocity and volumetric concentration of solid particles.

- **Temperature of transported slurry**
  Flowing slurry is continuously warmed up through friction losses. Temperature was continuously measured during the experiments. There was a pipe section where external water cooling was used to prevent slurry warming.
- **Sampling**
  Samples were taken from the bottom and top sections of the flowing slurry at the sampling point installed behind the measuring pipe sections.


Figure 26: Picture of the hydraulic test system.

### 3.3.2 Results

- **Centrifugal slurry pump (WARMAN)**
  The centrifugal slurry pump was used to investigate the hydraulic transport of Gyöngyösoroszi flotation tailings. We could increase the volumetric concentration up to about 30%, where the transport stopped. The reason was not the power limitation of the pump, flow rate decreased to zero by rotation pump. High concentration hydraulic transportation was reached, but paste state was not achieved (for paste state it is necessary to reach at least 48% volumetric concentration for this material).
- **Piston and membrane pump (ABEL)**
  **A - First** investigations were carried out with Power Plant Fly Ash. We could increase the volumetric concentration to up to 51%. High concentration hydraulic transport was achieved. The particle size distribution of this material did not allow paste state because of the lack of fine particles.
  **B -** The s**econd** investigation series was carried out with flotation tailings from Gyöngyösoroszi. The ball valves blocked and the transportation stopped at approximately 20% volumetric concentration. It was concluded that the ABEL type piston and membrane pump is unsuitable to transport materials that contain high amounts of fine particles such a clay minerals.
- **Screw pump (MONO)**
  Investigations were carried out with flotation tailings from Gyöngyösoroszi. The volumetric concentration could be increased up to 48%. Transport losses drastically increased even until mechanical failure of the measuring system, and pressure meters reaching the limit of their measuring range. We have approximated the limit of pipe strain (15 bar). Paste state has been successfully reached (Fig. 27).

Figure 27: Gyöngyösoroszi tailings material at paste state.

In conclusion, both material and technical requirements of paste transport have been analysed. Concerning the material properties, it was found that the particle size distribution of the transported slurries is a significant parameter in paste technology. However, whether a suspension is able to form a paste depends on many variables, and it is necessary to perform tests on every material to determine if the material is suitable for paste producing.

From technical point of view it is clear that for paste transport operators and designers should choose pumps not only with high capacity and pressure head, but their construction is also important to be “paste proof”. During tests with the WARMAN pump the continuous material flow was interrupted, and during the tests with the ABEL type pump the valves blocked. Only the use of the screw pump proved to be feasible for paste transport in our tests.

## 3.4 Pressure Loss Calculation of Tailings Paste Pipe Transport

### 3.4.1 Introduction

For the design of a paste tailings deposition system, the method how to calculate the pressure loss of a slurry flow in a horizontal, straight pipe section is probably the most important. The main technical parameters of the transport are either required as an input to this calculation or they are results of it. There are many models and equations in the literature how to calculate the pressure drop for the different cases, but still it is necessary to carry out experimental work in specially designed hydraulic test loops to validate the models. Due to the historical development of hydraulic transport – initially only the dilute slurry technique was applied, and as the environmental aspects of this fairly hazardous industrial activity had been becoming stricter the concentration of solids was increased and the dense slurry technique and later the paste technology had been introduced – the pressure loss calculation of pastes in pipes was not so thoroughly investigated as the parameter estimation for the previous two techniques. Therefore, in the frame of the TAILSAFE project, a pilot-scale hydraulic test loop has been built and systematic paste transport experiments have been carried out with different materials and different types of pumps.

### 3.4.2 Experimental Series

In the pilot-scale hydraulic test loop, hydraulic transport experiments were carried out with two test materials (1. tailings from Gyöngyösoroszi, Hungary; 2. coal power plant fly ash) and with three different types of pumps. In Table 4 the different experimental series and the achieved volumetric transport concentrations of each experiment are listed.

As discussed in Section 3.3, piston displacement pumps are generally applied if high pressures are needed. Paste pipe transport requires really high pressures, therefore the membrane piston displacement pump was tested with both the Gyöngyösoroszi tailings and the fly-ash. The membrane piston displacement pump worked perfectly with the fly-ash, but the Gyöngyösoroszi tailings with its high clay content blocked the ball valves of the pump, so only the first mixture with a solids concentration of 10% could be measured for this material. For the Gyöngyösoroszi tailings the screw displacement pump worked, even in the case of high concentration where extreme high pressures were needed. The centrifugal pump was not capable to handle the excessive high pressures at high concentrations.

Table 4: Achieved volumetric transport concentrations of each experiment.

| Experiments with **Gyöngyösoroszi tailings** | | | | | |
|---|---|---|---|---|---|
| *Type of pump* | *Transport volumetric concentration $C_T$ [%]* | | | | |
| Piston displacement pump | 10 | - | - | - | - |
| Screw displacement pump | 10 | 21 | 30 | 40 | 48 |
| Centrifugal pump | 10 | 15 | 19.6 | 28 | 35 |
| Experiments with **coal power plant fly-ash** | | | | | |
| *Type of pump* | *Transport volumetric concentration $C_T$ [%]* | | | | |
| Piston displacement pump | 10.66 | 20.55 | 30.7 | 39.7 | 50.05 |

### 3.4.3 Theoretical Background of the Pressure Drop Calculation

The Department of Process Engineering of the University of Miskolc has been dealing with two-phases pipe flows for a long time. One of their most important results is the fine suspension/coarse mixture flows model that was introduced by Tarján and Faitli at the end of the last century (Tarján & Faitli 1998). The most important parameter of a solid granular material system is the particle size in respect of the type of the two-phase flow. The effect of the particle size can be examined if all the other parameters are kept constant, only the size of mono-disperse solid fractions is changed and examined systematically. Such experiments with many different materials had been carried out, where narrow particle fractions were sieved and tested in a tube viscometer with three different pipes.

The effect of the particle size is summarized in Figure 28. Typical pressure drop curves are shown for two different particle sizes. The transport concentration and the pipe and all the other parameters are the same in both cases. In the case of small particles (2) the pressure loss curve is similar to that of clear fluids. This kind of slurry flow is called *fine suspension flow*. In this case the solid-liquid mixture behaves like a clear fluid with its own density and rheological properties in the given pipe. This is like a continuum, and the continuum model can be well applied. The pressure loss can be calculated on the basis of the rheological properties of the suspension. Unfortunately the rheological properties of suspensions of fine powders and water are more non-Newtonian as the concentration increases. At this time the general tensor level non-Newtonian constitutive equation does not exist but the artificial one-dimensional rheological models such as Bingham-plastics, Exponential-Law, yield-pseudo-plastics, etc. can be well applied in the engineering practice. In a given practical case the rheological model and the parameters in it should be known as a function of the concentration. The pressure loss can be calculated by the methods given in the literature for each rheological model.

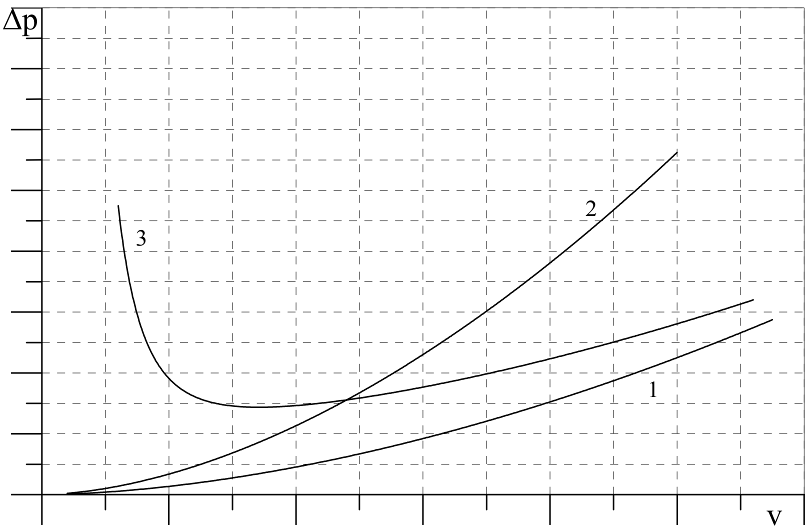



Figure 28: The effect of the particle size (1 - water, 2 - fine, 3 - coarse particles) (Faitli 2000).

In the case of coarse particles (curve 3) the pressure loss curve has a minimum point. This case had been extremely widely examined in the literature and there is a fairly big mis-understanding in this respect as well. If the flow rate is sufficiently high the concentration distribution is symmetrical; this is called pseudo-homogeneous flow and sometimes rheological models are used for calculating the pressure loss. This is evidently wrong. In this case the continuum model is hard to be applied or, in our opinion, it cannot be applied. In this real two-phases flow the phenomenon can be imagined as fluid flows between big rocks. This point of view is logically supported by the two-layer model, where pressure loss of the sliding bed is calculated merely as mechanical friction between the pipe wall and the particles. What is surprising here: the transport of the same amount of coarse particles requires less energy than transporting fine particles in the same pipe. This can be shown in experiments. In a given case the pressure loss curve should be measured and a function should be curve-fitted to it. The first model to this type of flow was the Durand equation. Evaluating our experiments it was determined that the Durand equation was well applicable with changing the K multiplier and the n exponent as parameters in it for a given material. This kind of flow with coarse particles was called as *coarse mixture flow*. The transition as a function of the particle size from fine suspension flow into coarse mixture flow is not sharp. However, a transitional or critical particle size can be interpreted or measured by this way. Smaller particles than the critical size can be transported in the given pipe in a fine suspension flow. The conclusion of these thoughts is that primarily the particle size determines the quality of the solid-liquid mixture flow. The actual flow patterns are then determined by the flow rate.

In reality, hydraulically transported solids generally contain particles in a wide spectrum of different sizes, shapes and densities. For this case the *coarse in fine suspension flow* model can be applied. The particles smaller than the limit particle size and the carrier liquid (generally water) form a fine suspension with its own physical properties such as suspension density and rheological behavior. The coarse particles are transported by this fine suspension. An experimental proof for this model is the measurement carried out in 1998 at the University of Miskolc with fly-ash from the Mátra Power Plant. The limit particle size for the fly-ashes was determined beforehand by sieving narrow particle fractions. Suspensions of them were tested in a tube viscometer. That research showed that the limit particle size of fly-ashes is about 160 µm. Two experiments were carried out: one test series with a tube viscometer (16 … 27 mm pipe) with the fine (< 160 µm) fraction only to measure the rheological behavior and parameters of the fine suspension part, and another in the pilot-plant scale test loop (53 … 120 mm pipes) with the actual material. Figure 29 shows a representative example, where the volumetric transport concentration was 33.8%. The calculated pressure loss curve of the clear carrier water is signed by 1. This curve was measured as well. From the data of the rheological measurement and the particle size distribution the pressure loss curve of the fine suspension part was calculated, signed by 2. Determining the parameters in the modified Durand equation the pressure loss was calculated for this coarse in fine suspension flow. The calculation fits the measured data really well.

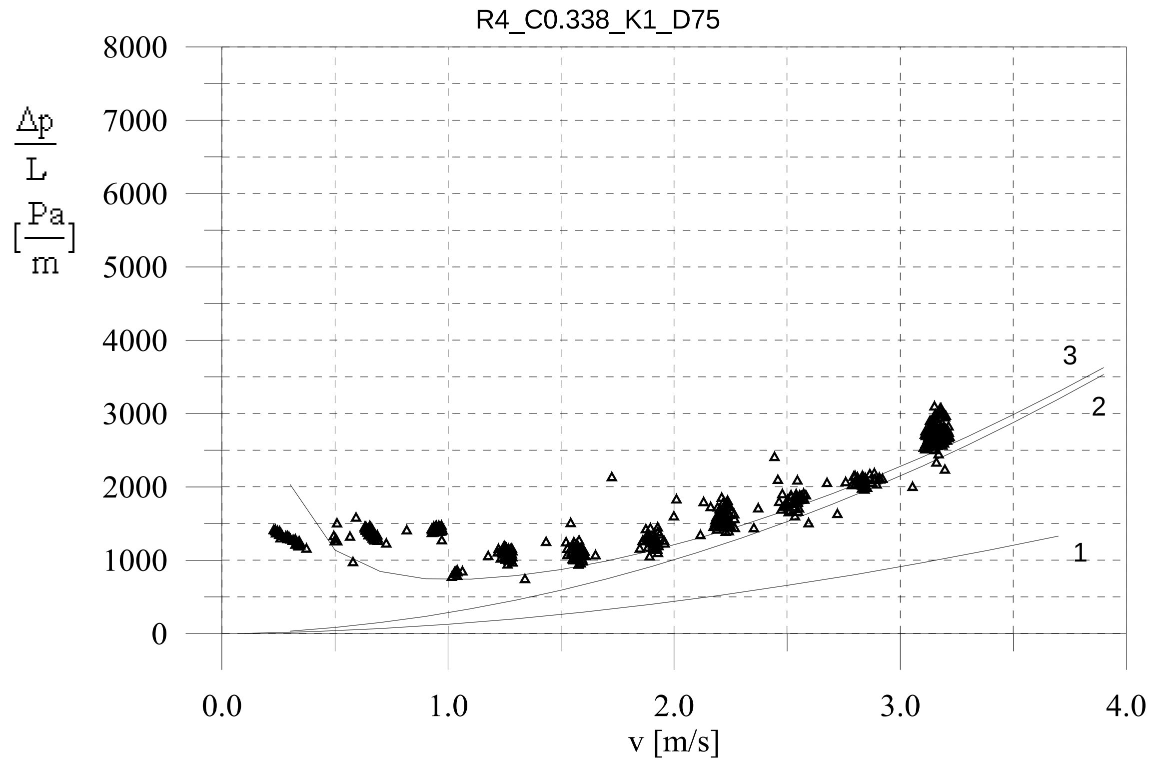



Figure 29: Pressure loss curve of a fly-ash, slag and water mixture.

### 3.4.4 Calculation of the Pressure Loss

The pressure loss is calculated using the fine suspension flow, the coarse mixture flow and the coarse in fine suspension flow models.

#### 3.4.4.1 Calculation of the Pressure Loss of the Fine Suspension Flow

If only fine particles (smaller than the limit particle size) are transported in the pipe a new fluid is formed which can be characterized by its density and rheological behavior. The pressure loss is calculated based on the applied rheological model and parameters. The Hersel and Bulkley model (Govier et al. 1972) as the constitutive equation for Yield-Pseudoplastic fluids is:

$$\tau = \tau_0 + K\left(-\frac{du}{dr}\right)^m \tag{1}$$

There are three rheological parameters in this equation, respectively the yield stress ($\tau_o$), the consistency index (K) and the flow behavior index (m). The application of this model is quite practical, because if the yield stress is set to zero we get the rheological model of Power Law fluids, if the exponent is set to 1 we get the Bingham plastics rheological model and if we set the yield stress to zero and the exponent to 1 we get the Newtonian model. Today, personal computers have enormous calculation potential, so the calculation with the very complicated rheological model for Yield-Pseudoplastics fluids is no problem. The constitutive equation can be integrated to obtain the well known expression for laminar flow:

$$\frac{8v}{D} = \frac{4}{K^{\frac{1}{m}}\tau_w^3}\left(\tau_w - \tau_0\right)^{\frac{1+m}{m}}\left[\frac{\left(\tau_w - \tau_0\right)^2}{\frac{1+3m}{m}} + \frac{2\tau_0\left(\tau_w - \tau_0\right)}{\frac{1+2m}{m}} + \frac{\tau_0^2}{\frac{1+m}{m}}\right] \tag{2}$$

This equation can be expressed in terms of the usual Fanning friction factor:

$$f = \frac{16}{\psi \, \mathrm{Re}_{PL}} \tag{3}$$

The '$\mathrm{Re}_{PL}$' term represents the Power Law Reynolds number defined by Equation (4). For taking into account the effect of the flow behavior index it is necessary to modify the Hedstrom number. The modified Hedstrom number '$\mathrm{He}_m$' is:

$$He_m = \frac{D^2\rho}{\tau_0}\left(\frac{\tau_0}{K}\right)^{\frac{2}{m}} \tag{4}$$

'Ψ' in Equation (3) can be expressed as:

$$\psi = (1+3m)^m (1-x_0)^{1+m} \left[ \frac{(1-x_0)^2}{1+3m} + \frac{2x_0(1-x_0)}{1+2m} + \frac{x_0^2}{1+m} \right]^m \tag{5}$$

where '$x_0$' is the quotient of the yield and wall shear stresses:

$$x_o = \frac{\tau_o}{\tau_w} \tag{6}$$

To obtain the value of '$x_0$' at an actual shear rate the next equation has to be solved:

$$\mathrm{Re}_{PL} = 2He_m \left( \frac{m}{1+3m} \right)^2 \left( \frac{\psi}{x_0} \right)^{\frac{2-m}{m}} \tag{7}$$

These equations cannot be solved directly. First by substituting Equation (5) into Equation (7) '$x_0$' can be calculated. If '$x_0$' is known 'Ψ' can be determined from Equation (5) and then the friction factor by Equation (3).

For the case of transitional and turbulent flow in smooth pipes, Hanks (1971, 1975) introduced a complete model for designing pipe transport systems using a modification of the Prandtl-Van Driest mixing length theory. The constitutive equation can be written taking the effect of turbulence into account by the mixing length component as:

$$\tau = \tau_0 + K\left(-\frac{du}{dr}\right)^m + \rho L^2 \left(-\frac{du}{dr}\right)^2 \tag{8}$$

'L' is a modified mixing length given by:

$$L = 0.36\frac{D}{2}(1-x)\{1-\exp[-\phi(1-x)]\} \tag{9}$$

$$\phi = \frac{R-R_C}{\sqrt{8}B} \tag{10}$$

The parameter R is an appropriate modification for this type of rheology of the turbulence parameter used for Newtonian fluids ($R = \mathrm{Re}\sqrt{f}$). '$R_C$' is the value of 'R' at the laminar-turbulent transition. 'R' is defined for this rheology as:

$$R = \left(\frac{1+3m}{m}\right)\left[\mathrm{Re}_{PL}\left(\frac{f}{16}\right)^{\frac{2-m}{m}}\right]^{\frac{1}{m}} \tag{11}$$

The parameter 'B' is an empirical parameter which accounts for the influence of the wall on 'L'. If 'B' is depend upon both the modified Hedstrom number and the flow behavior index the next relation can be suggested.

$$B = \frac{22}{m}\left[1 + \frac{0.00352He_m}{\left(1 + 0.000504He_m\right)^2}\right] \tag{12}$$

The modified constitutive equation can be expressed in a dimensionless form if we define the dimensionless shear rate as:

$$\xi = \frac{\left(-\frac{du}{dr}\right)}{\left(-\frac{du}{dr}\right)_w} \tag{13}$$

'x' is the dimensionless radial coordinate, (x = 2r/D). The 'w' sub-index means the value at the pipe wall of the indexed variable. The dimensionless modified constitutive equation can be written as:

$$x = x_0 + \left(1 + x_0\right)\zeta^m + \frac{1}{8}R^2\left(1 - x_0\right)^{\frac{2}{m}}\lambda\zeta^2 \tag{14}$$

'λ' is the dimensionless mixing length (λ=2L/D). Equation (14) can be expressed after considerable algebra as

$$\mathrm{Re}_{PL} = \left(1 - x_0\right)^{\frac{2-m}{m}}\left(\frac{m}{1+3m}\right)^m R^2\left[\int_{x_0}^{1} x^2\zeta\left(x, x_{0,}R\right)d\zeta\right]^{2-m} \tag{15}$$

The '$\zeta(x,x_0,R)$' function is implicit, therefore in the numerical evaluation process, Equation (14) must be solved for each 'x' to determine the value of '$\zeta(x,x_0,R)$'. For solving this equation system one more equation is necessary. After some manipulation of the definitions of 'f', '$\mathrm{Re}_{PL}$', 'R' and '$He_m$' the next equation can be expressed:

$$R^2 = \frac{2He_m}{x_0^{\frac{2-m}{m}}} \tag{16}$$

The calculation design procedure to be used is as follows, after Hanks:

1. Determine the values of 'm', '$\tau_0$', 'K' and 'D' to be used in a particular design problem.
2. Compute '$He_m$' from Equation (4).
3. Compute '$x_{oc}$' from Equation (18).

4. Compute '$(Re_{PL})_C$' from Equation (17), and '$f_C$' from Equation (3) and Equation (1.28).
5. Compute '$v_C$' the transitional average velocity from the Darcy and Weisbach equation. If the desired working velocities are smaller than this value (the flow is laminar), use Equation (3-7) to design the pipeline. If the desired working velocities are greater than '$v_C$' (the flow is turbulent) use the following procedure.
6. Compute '$R_C$' from Equation (10) using $Re_{PL} = (Re_{PL})_C$; and $f = f_C$.
7. Choose a series of values of $R > R_C$ ('R' is the working calculation parameter and does not appear in the final results).
8. For a given value of 'R', calculate '$x_0$' from Equation (17).
9. Evaluate the integral in Equation (16) by a numerical integral process.
10. Compute '$Re_{PL}$' from Equation (16).
11. Compute 'f' from Equation (3).
12. Repeat steps 8-11 for each of the values of 'R' chosen in step no. 7. This generates a curve of 'f' vs. '$Re_{PL}$' for the 'm', '$He_m$' combination of the desired design problem. This curve is then used in place of the usual Moody diagram to complete the hydraulic resistance portion of the pipeline design.

However, due to the development of the speed of the digital computers, this problem can be solved if the flow rate is given as well. An iteration loop has been built, where the 'R' working parameter is in the centre of this loop.

To correlate the laminar-turbulent transition the equations of Hanks – based on his transitional theory – can be applied. The value of the Power Law Reynolds number at the transition can be determined from the next equation:

$$\left(\mathrm{Re}_{PL}\right)_C = \frac{6464m}{(1+3m)^m}(2+m)^{\frac{2+m}{1+m}} \frac{\left[\dfrac{(1-x_{0C})^2}{1+3m} + \dfrac{2x_{0C}(1-x_{0C})}{1+2m} + \dfrac{x_{0C}^2}{1+m}\right]^{2-m}}{(1-x_{0C})^m} \tag{17}$$

Where '$x_{0C}$' is the value of '$x_0$' at the transition and can be obtained from Equation (18):

$$He_m = \frac{3232}{m}(2+m)^{\frac{2+m}{1+m}}\left[\frac{x_{0C}}{(1-x_{0C})^{1+m}}\right]^{\frac{2-m}{m}}\left(\frac{1}{1-x_{0C}}\right)^m \tag{18}$$

To carry out calculations by the iterative solution of the presented Hanks method a computer program called “Rheology” has been written.

### 3.4.4.2 Calculation of the Pressure Loss of the Coarse Mixture Flow

The literature gives a lot of examples for the calculation of the pressure loss of solid-liquid pipe flows when the particle size distribution of the solids is not very wide and the size range is considered as coarse. However, since a universally applicable equation does not exist, still experimental work has to be carried out. The modified Durand equation is suggested for this case:

$$\Delta p = \Delta p_w \left\{ 1 + C_{TD} \frac{K\left(\frac{\rho_S}{\rho_w} - 1\right)^{1.5}}{C_D^{0.75}} \left(\frac{\sqrt{gD}}{v}\right)^n \right\} \tag{19}$$

In this equation K and n are material dependent parameters which are determined by curve fitting to the data of the pilot-scale experiments.

#### 3.4.4.3 Calculation of the Pressure Loss of the Coarse in Fine Suspension Flow

For being able to calculate the pressure loss of hydraulic transport for wide granular solid material, the particle size distribution and the rheological behavior of suspensions as function of the concentration of the fine fraction (smaller than the limit particle size) have to be known. The solids fraction is divided into two parts. The transport volumetric concentration of the fines ($C_{TF}$) can then be calculated, and the concentration of the coarse material ($C_{TD}$) in this fine suspension is calculated as well. The pressure loss of the fine suspension ($\Delta p_F$) is calculated based on the rheological model and the parameters of the $C_{TF}$ concentration fine suspension. Then the additional pressure loss caused by the coarse particles is calculated by the modified Durand equation applied for this coarse in fine suspension model:

$$\Delta p = \Delta p_F \left\{ 1 + C_{TDF} \frac{K\left(\frac{\rho_S}{\rho_F} - 1\right)^{1.5}}{C_D^{0.75}} \left(\frac{\sqrt{gD}}{v}\right)^n \right\} \tag{20}$$

### 3.4.5 Evaluation of the Pilot-Scale Hydraulic Experiments with Gyöngyösoroszi Tailings

During the evaluation process it was established that the *fine suspension model* is well applicable for these measurements. It is quite expedient if we take into account that the particle size distribution of the material shows 90% particles smaller than 40 μm. Results of the tube viscometer measurements were as follows: rheological behavior is Bingham plastics, the yield stress and the coefficient of rigidity can be calculated from the following equations:

$$\eta_F[Pas] = \mu_o[Pas]\left(1 + 80C[] + 600000C[]^9\right)$$
$$\tau_F[Pa] = 0.7e^{10.4C[]} \tag{21}$$

The measured particle density is 2701 kg/m$^3$. From this data the main parameters into the pilot-scale test loop can be calculated:

| $C_T$ [%] | $\rho_F$ [kg/m$^3$] | $\eta$ [mPas] | $\tau_0$ [Pa] |
|---|---|---|---|
| 10 | *1170* | *9* | *2* |
| 21 | *1357* | *18* | *6* |
| 30 | *1510* | *37* | *16* |
| 40 | *1680* | *190* | *45* |
| 48 | *1816* | *850* | *103* |

Using this data the pressure loss curve of each measurement can be calculated by the Hanks method described earlier. Three examples are given in Figures 30-32.

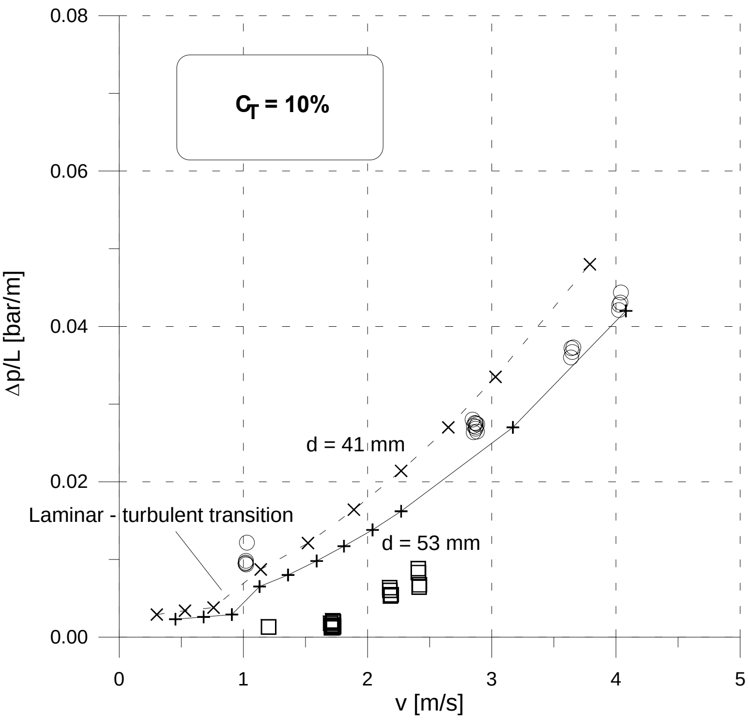



Figure 30: Pressure loss curve (measured/calculated) for the 10% transport volumetric concentration of a Gyöngyösoroszi tailings/water suspension.

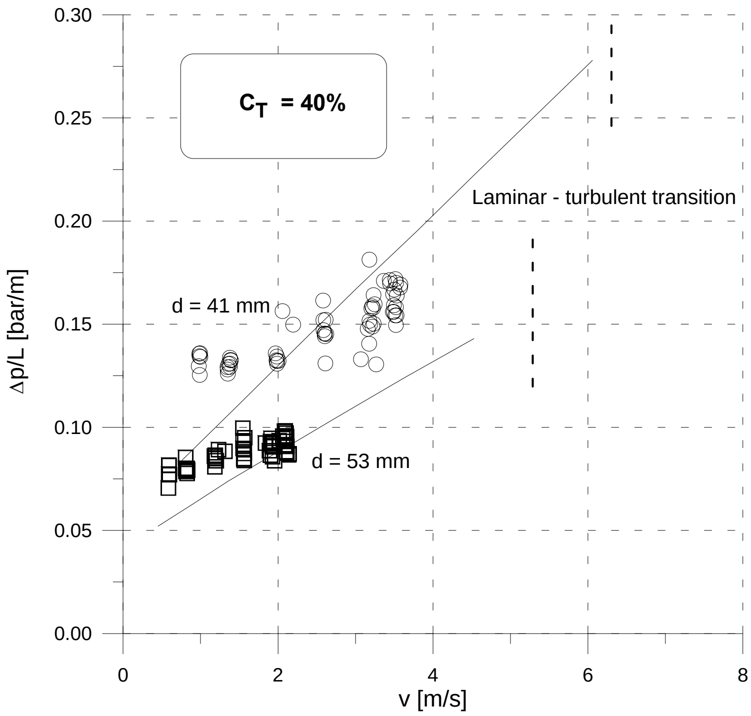



Figure 31: Pressure loss curve (measured/calculated) of the 40% transport volumetric concentration of a Gyöngyösoroszi tailings/water suspension.

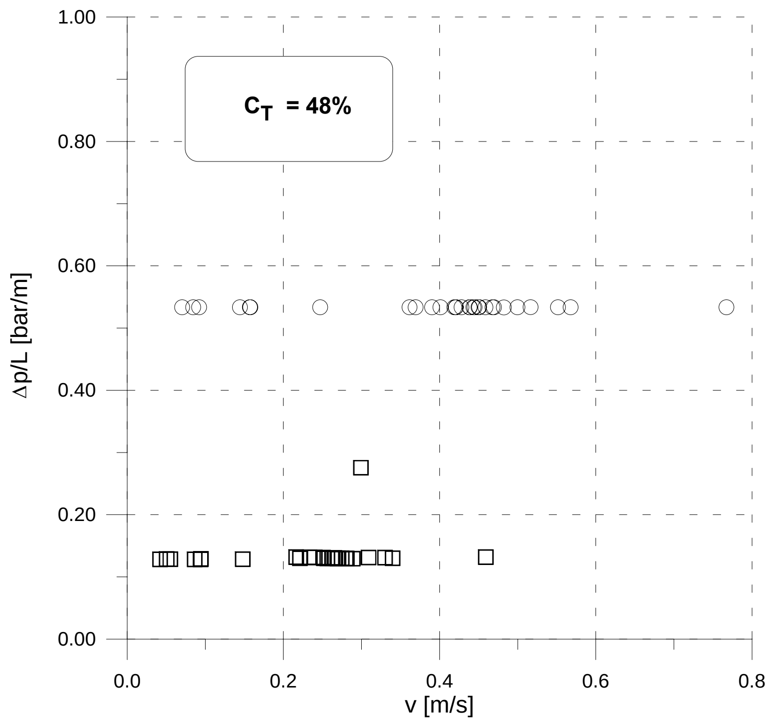



Figure 32: Measured pressure losses for the 48% transport volumetric concentration of a Gyöngyösoroszi tailings/water suspension: overload of the pressure sensors.

In the measurements carried out with 10% transport volumetric concentration suspensions (Fig. 30), there are mainly turbulent points (due to the low concentration). The laminar-turbulent transition was calculated by the Hanks method as well (Fig. 30). In the measurements carried out with 40% transport volumetric concentration (Fig. 31), all the measured points are laminar, even at the highest flow velocities (3.8 m/s). The concentration is very high, and the suspension is very viscous (Fig. 31).

The results obtained for measurements carried out with 48% transport volumetric concentration suspensions (Fig. 32) are not satisfactory. The reason is the extreme high pressure loss, leading to an overload on all pressure sensors. During the measurement also a small accident happened when a rubber pipe section was blown away due to the high pressure and the slurry was sprayed into the lab. Another conclusion of these data is that the measured flow velocities are very low. The screw pump has a rigid characteristic, but it was suppressed by the paste in the pipe.

As it was described earlier that only the 40% and 48% volumetric concentration suspensions – from these pilot scale tests – accomplished the "slump cone test"; only they are paste. The calculated pressure loss curves based on the rheological behavior in accordance with the fine suspension model fit well into these measurements.

### 3.4.6 Evaluation of the Pilot-Scale Hydraulic Experiments with Power Plant Fly Ash

To correlate the pressure loss of these measurement the *coarse in fine suspension model* was applied because the particle size distribution of the fly-ash is wide, 50% is finer than the earlier determined limit particle size of 160 µm. The "particle" density of this fly-ash is 2492 kg/m$^3$. By this data and by the results of the separately carried rheological tests the concentrations and rheological parameters of the fine suspension portion can be calculated.

The measured rheological behavior of the fine fly-ash/water suspensions were Bingham-plastics as well and the yield stress and the coefficient of rigidity can be calculated as function of the volumetric concentration from the following curve fitted equations:

$$\begin{aligned} \eta_F[Pas] &= \mu_o[Pas]\left(1+25C[]+360000C[]^{9.2}\right) \\ \tau_F[Pa] &= 0.01e^{18.5C[]} \end{aligned} \qquad (22)$$

| $C_T$ [%] | $C_{TF}$ [%] | $\rho_F$ [kg/m$^3$] | $\eta_F$ [mPas] | $\tau_{oF}$ [Pa] | $C_{TDF}$ [%] |
|---|---|---|---|---|---|
| 10.66 | 5.6 | 1084 | 2.4 | 0.03 | 5.3 |
| 20.55 | 11.4 | 1170 | 4 | 0.08 | 10 |
| 30.7 | 18 | 1270 | 5.6 | 0.3 | 15 |
| 39.7 | 25 | 1369 | 8 | 1 | 20 |
| 50.05 | 33 | 1498 | 24 | 5 | 25 |

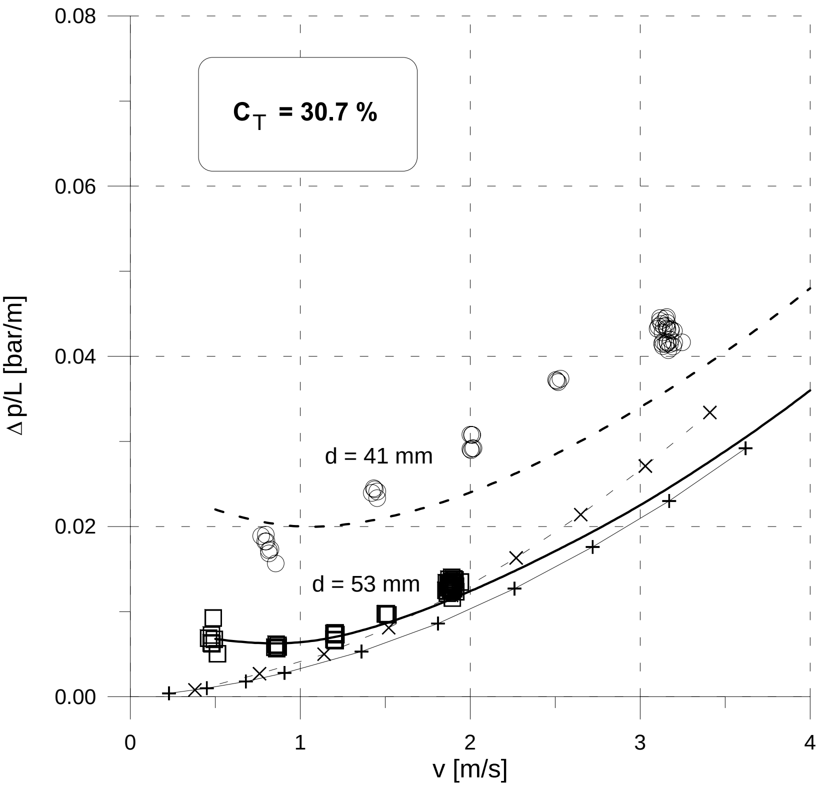



Figure 33: The pressure loss curve (measured/calculated) of the 30.7% transport volumetric concentration coal power plant fly-ash/water suspension.

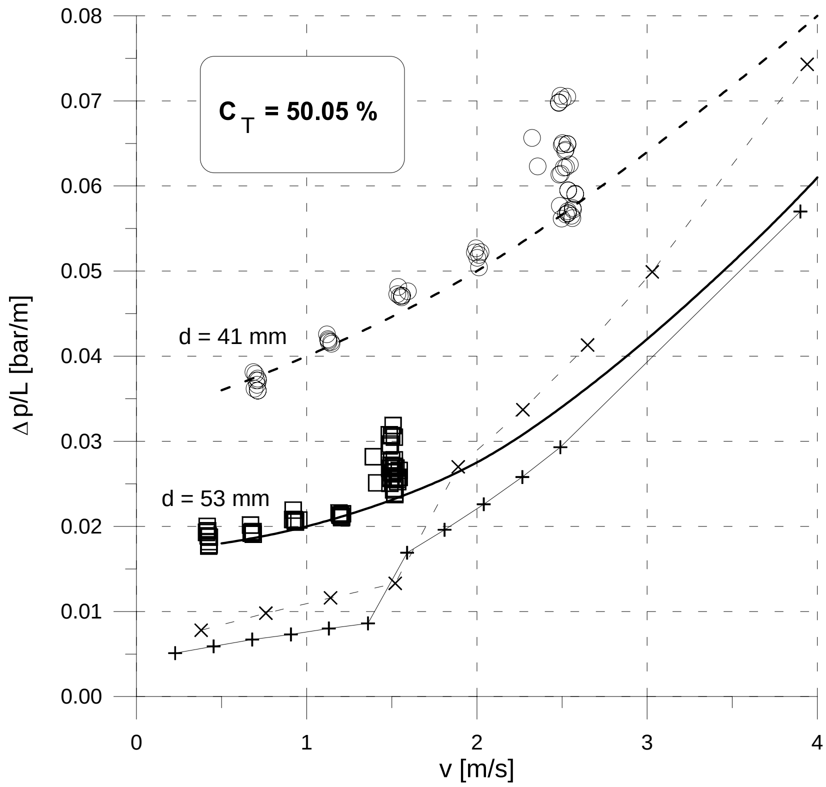



Figure 34: The pressure loss curve (measured/calculated) of the 50.05% transport volumetric concentration coal power plant fly-ash/water suspension.

The pressure loss of the fine particulate portion is calculated by the Hanks method and the additional loss due to the coarse particles is calculated by the modified Durand equation. Two examples are shown in Figs. 33 and 34.

What is quite interesting from these measurements is that although the achieved 50.05% transport volumetric concentration is very high, stable hydraulic transport was realized in the laboratory. Another striking aspect is that this high concentration fly-ash suspension was still not a paste because in the slump cone test this material flowed away. The reason might be the coarser particulate structure. The *coarse in fine suspension model* predicts the measured pressure loss curves very well.

## 3.5 Conclusion

➢ The *fine suspension flow* and the *coarse in fine suspension flow* models gave good results for the hydraulic experiments with tailings and fly-ash.

➢ Pilot-plant-scale **paste pipe transport** was realized for Gyöngyösoroszi tailings at 40% and 48% volumetric concentration.

➢ A pressure loss calculation model and computer program for paste technology parameters calculation have been presented: Fine suspension flow → calculation on rheological behavior → laminar range: solution of the material equation; turbulent range and laminar-turbulent transition; iterative solution of the Hanks method for Yield-pseudoplastic fluids.

➢ The main technological parameters and costs of paste technology can be determined by the presented model. The necessary experiments to be conducted beforehand to determine required input parameters are the following:
- Particle size distribution analysis
- Measuring the particle density
- Rheological tests using the tube viscometer

# 4 DESIGN OF SLURRY THICKENING, DEVELOPMENT OF A NEW TYPE OF THICKENER BASED ON VIBRATING RODS

## 4.1 Introduction

The safety of tailings facilities can be considerably increased by the application of high concentrate deposition of the particulate solid material. This is the application of the so called paste technology. A key aspect in designing, building and operating a paste transport and deposition system is the slurry thickening to produce the high concentration paste. This section reports about the theoretical background of the design of continuous thickeners, about the principle of a perspective new thickener based on vibrating rods, about a pilot-scale new test equipment to carry out classical column settling experiment with or without vibration with the continuous measurement of the vertical concentration distribution in the test column as well as about three series of settling experiments. The first settling experiment series has been carried out with a selected reference material (glass sand) to investigate the effect of vibration, the second one focused on the effect of clay materials and the third one investigated the effect of vibration into the usage of flocculating reagents.

## 4.2 Design of Continuous Thickeners

The basis of any design method for continuous thickeners is the steady-state thickening experiment carried out in a settling column. The height of the surface of the settling solids is measured as a function of time. All the following considerations and calculations are based on this function. A careful execution of the settling column test as well as the correct frequency and accuracy measurement of the data is essential. The settling of different slurries is considerably different depending on the materials and circumstances. Categorized types of settling are shown in Figure 35.

The shape of the H(t) curve indicates different types of settling, namely single particle settling, zoned settling and compression. As experience shows, one single function, such as a linear or exponential functions, cannot be applied to describe the total phenomena of settling. The application of two functions is evidently unfavorable. Therefore, the design method developed by our department, published by Csőke et al. (1994), based on the application of a third order spline function, can be suggested for the design of continuous thickeners. It is not necessary to know the exact mathematical expression to describe the complete H(t) function. If the following can be determined by the applied mathematical method, the thickener can be reliably designed:

1. The estimation of the H(t) value at $H(t_i) < H(t) < H(t_{i+1})$ any time (interpolation).
2. Determination of the first and second derivative of the H(t) function to determine the minimum and maximum and inflexion points.
3. Numerical determination of the $\int H(t)dt$ integral, which is the area below the curve.

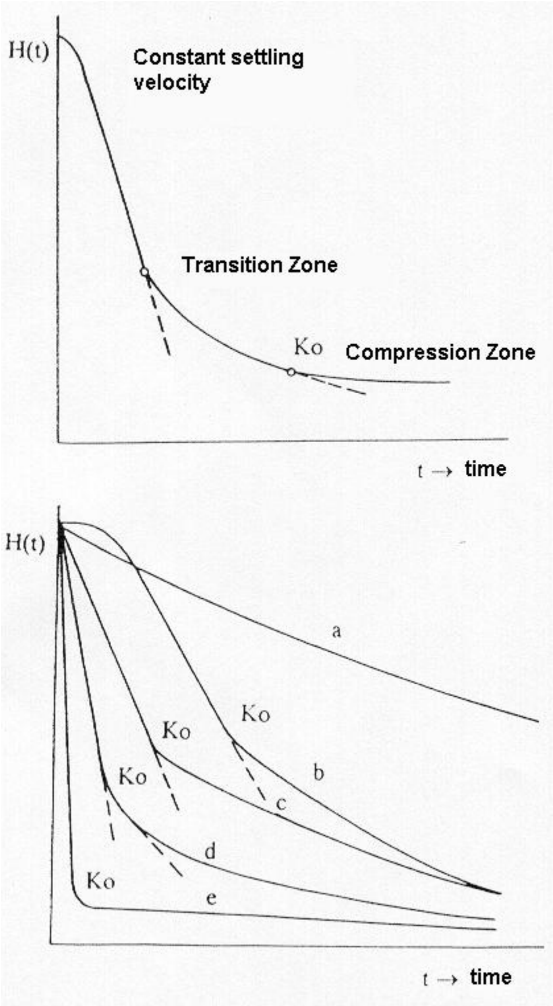



Figure 35: Typical settling types of different slurries: a = dense slurry (flocculated or not flocculated), b = medium dense slurry (flocculated or not flocculated), c = like b, d = flocculated dilute slurry, e = not flocculated dilute slurry, $K_o$ compression point (after Schubert 1984).

The applied function is the local spline smoothing, using the $\left(p_3(x) = a_3x^3 + a_2x^2 + a_1x + a_o\right)$ polynom, called third order spline function. This function can be two times continuously derived. By this numerical method the following characterizing functions of the continuous thickeners can be determined. The following relationships are set:

The slurry height as a function of time:

$$H = f(t) \tag{23}$$

Transformed slurry height as a function of time:

$$H'(t) = \frac{H(t) - H_\infty}{H_o - H_\infty} \tag{24}$$

The settling velocity as a function of time:

$$v(t) = \frac{dH(t)}{dt} \tag{25}$$

The solids concentration as a function of time:

$$C(t) = \frac{C_o H_o}{H(t) + tv(t)} \quad (26)$$

where $C_0$ is the initial or mixed concentration. The flux of the settling solid material (flow rate of the settling solids through a unit area):

$$S = vC \quad (27)$$

The minimal value of solids flux:

$$\frac{dS}{dC} = 0 \to S_{\min} \quad (28)$$

If the concentration of the outflow slurry at the bottom of the thickener $C_u$ is constant, the effective thickening area is as follows:

$$A_e = \frac{\dot{m}}{S_{\min}} \quad (29)$$

where, $\dot{m}$ is the mass flow rate of the solid material. The effective thickening depth can be obtained from the following formula:

$$h_e = \frac{M}{A_e}\left[\frac{1}{\rho_f}(t_k - t_u) + \frac{1}{\rho_m}\int_{t_k}^{t_u} D(t)dt\right] \quad (30)$$

where, $\Delta t = t_k - t_u$ is the residence time of the slurry in the thickener, $\rho_f$ is the density of solids, $\rho_m$ is the one of fluid and D(t) is the diluting, mass of fluid divided by the mass of solids. The numerical method of the application of the third order spline function makes it possible to determine the acceleration of settling as a function of the time:

$$v'(t) = \frac{dv(t)}{dt} \quad (31)$$

as well as the velocity of concentration change:

$$C'(t) = \frac{dC(t)}{dt} \quad (32)$$

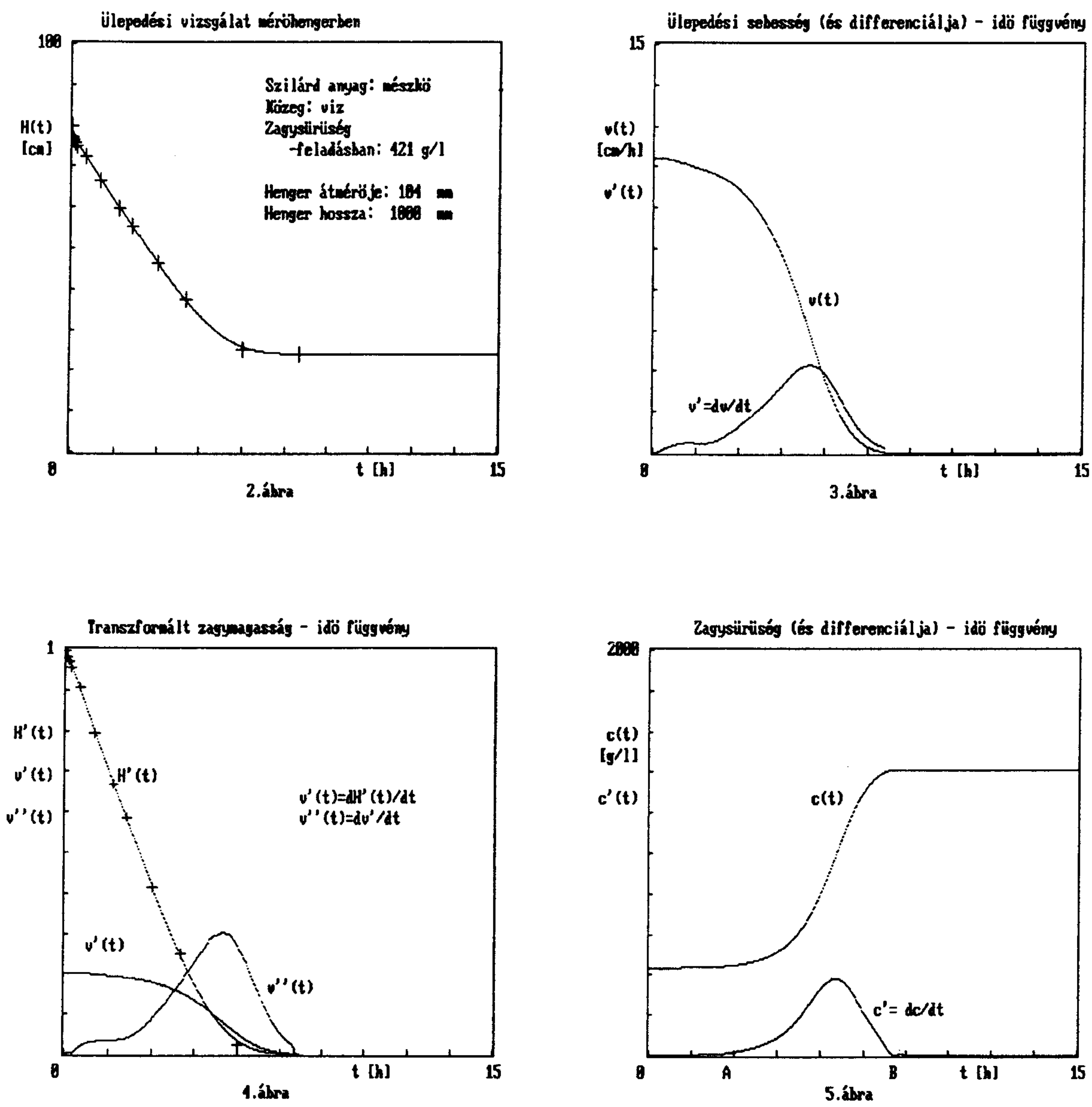
The first and second derivative of the transformed H'(t) curve can be determined as well. The transformed settling velocity as a function of time:

$$v_{Tr}(t) = \frac{dH'(t)}{dt} \tag{33}$$

and the transformed acceleration of settling as a function of time:

$$v'_{Tr}(t) = \frac{dv_{Tr}(t)}{dt} \tag{34}$$

As an example to this numerical design method, Fig. 36 shows the calculated curves for a test column settling.




*2-3-4-5. ábrák*

Figure 36: The measured and calculated characterizing functions of settling (Csőke at all, 1994).

## 4.3 Principle of a New Slurry Thickener Based on Vibrating Rods

In the mineral industry where the waste materials have to be deposited in tailings facilities, there are generally very fine particles in the process residues. The thickening of the suspensions containing such fine particles is quite difficult, and often flocculating reagents are added. These reagents are fairly expensive and might cause additional environmental problems. It is expedient that mechanical thickening processes are generally better for the environment compared to the chemical or chemically supported processes. Also in economical respects mechanical processes might be competitive. Sándor George Kiss (US Patent, 1974) presented an idea about a new type of thickener where settling are supported by vibrating rods. The principle of this equipment is shown on Fig. 37.

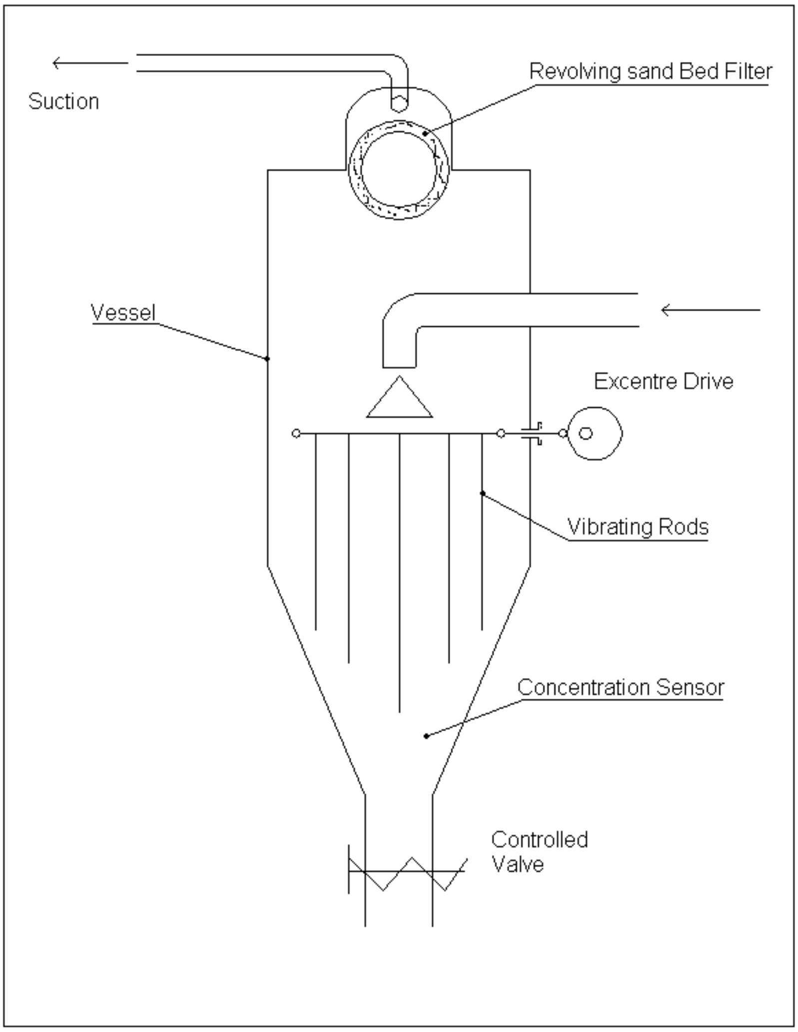



Figure 37: The principle of the new slurry thickener with vibrating rods.

The slurry is fed into the middle of the thickener by a pump. A structure made up by a number of rods is installed in the tank. Through a sealed shaft this rod structure is vibrated by means of a driver unit, e.g. an eccentric drive. At the bottom of the tank there is a controlled valve.

The signal controlling this valve is fed in from a concentration sensor. Another solution to control the downward flow is the application of a transporting screw. The upper part of the thickener is a revolving sand bed filter. There are two cylinders with small holes on it or two cylinders made by wire mesh and narrowly separated sand is filled in between them. This sand bed filter revolves and by this way cleaning is possible by a cleaning blade.

## 4.4 Pilot-Scale Batch Instrument for the Design of Slurry Thickeners (for Traditional as well as for the Vibrating Thickener)

So far it has been pointed out that the traditional column settling test is very important for the design of continuous thickeners. Even the size and the measuring protocol systems of the column settling test should be further improved. In addition, an investigation of the effect of vibration into settling is evidently necessary to be able to design a new thickener based on vibrating rods. Therefore, a new pilot-scale settling column has been developed (Fig. 38).

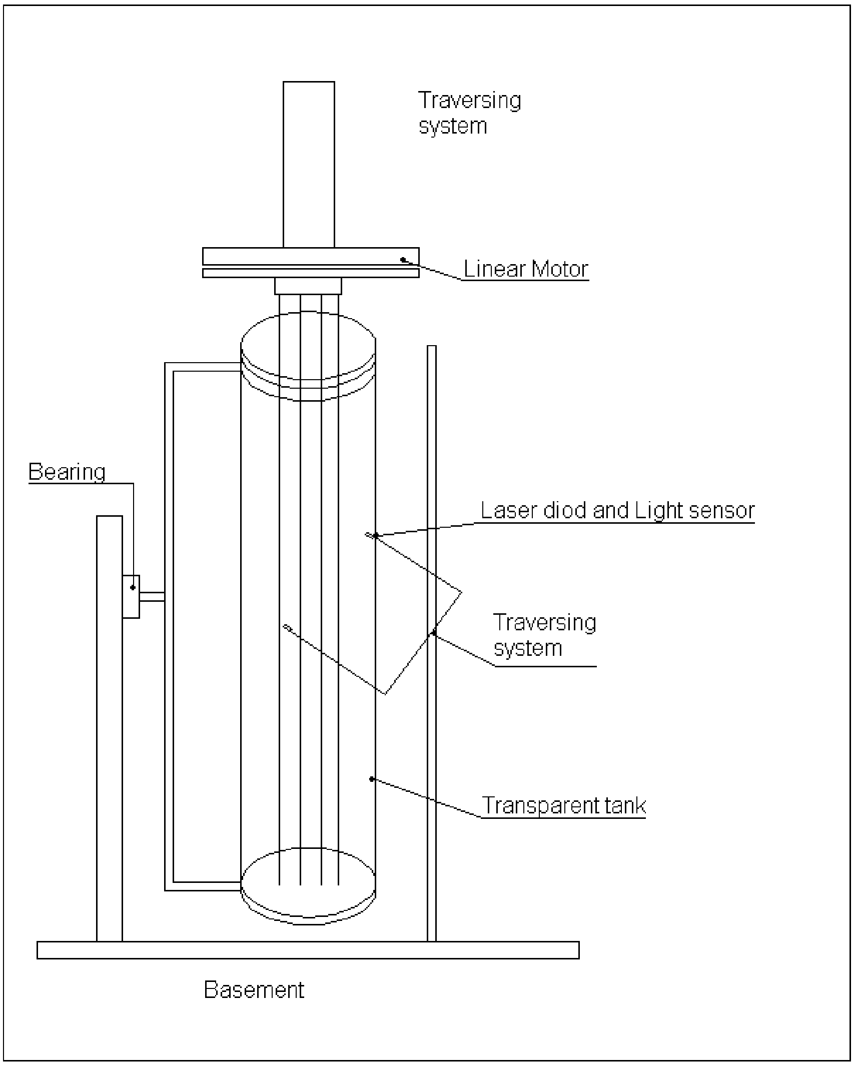



Figure 38: Schematic drawing of the new pilot-scale settling column.

The settling column is made by transparent plastic material. Its diameter is 24 cm, the height is 100 cm, and the effective volume is about 40 l. Since manual feeding, emptying and mixing of a volume of this size is difficult, the column is mounted via a bearing on a steel fork. By this way the column can be turned around the bearing. The height of the settling slurry as a function of time is measured by a computer data acquisition system. At one end of a second steel fork a laser diode is mounted. At the opposite side of the fork a light sensing device is mounted. The output voltage of the electronic system of the light sensing device is proportional to the concentration of the slurry in the settling column. The function between the measured light and concentration has been calibrated previously by a test vessel of the same size and material as the settling column. Discrete concentration suspensions have been mixed and the output voltage was measured. Up to a maximal concentration the light-inducted voltage is proportional to the concentration, but above it is not. The concentration measuring system with laser light is continuously moved up and down by a traversing system to scan the concentration distribution. In the compressed zone the concentration distribution is assumed as homogeneous. So by this computer aided measuring system not only the settling slurry height can be measured as a function of time but also the concentration distribution at discrete moments during the experiment. Because of its increased size and sophisticated measuring system this pilot-scale settling column is a suitable instrument to carry out the tests necessary to obtain the data needed for the design of continuous thickeners.

Since the pilot-scale settling column is equipped with a test system to investigate the effect of vibration into settling, it is also necessary to determine the main vibrating parameters for each material investigated, i.e. the frequency, the amplitude and the rod structure, to be able to design the thickener with vibrating rods. The test equipment should be very flexible in this respect. The application of the high-tech linear motor is the solution into this problem. It is a question of software parameters to set different amplitude and frequency vibration or any kind of motion. The resolution of the linear motor is 0.01 μm. The linear motor is mounted on a traversing system, because during feeding and emptying the motor is lifted up removing the rods as well.

The data acquisition and control software has been written in C++ in LabWindows CVI. Figs. 39 to 41 show photos of the pilot-scale settling column.

Summary of the main features of the built pilot-scale settling column with vibration:

- 40 l volume batch.
- Continuous measurement of the H = f(t) function.
- Measuring the concentration distribution.
- Sophisticated batch experiment without vibration.
- Suitable to determine the optimal vibration frequency, amplitude and rod construction for the examined material.
- Up to date technology: Linear motor

Figure 39: The pilot-scale settling column with vibration.

Figure 40: The laser light concentration measuring device.

Figure 41: The linear motor's traversing system

## 4.5 Settling Experiments

### 4.5.1 Settling Experiments with a Reference Material with and without Vibration

In the introduction it was emphasised that the settling behaviours of suspended materials differ mainly depending on the material composition and the solids concentration in the suspension. To examine systematically the effect of vibration into settling it was necessary to select a reference material for the investigations. This material should be inert, very fine granular and easily reproducible. The material selected was glass sand from Fehérvárcsúrgó, Hungary. Two types of this powder were used:

| | |
|---|---|
| a30 | 90% smaller than 30 µm |
| a75 | 90% smaller than 75 µm |

The initial or mixed concentration is very important in respect of settling. Three suspension densities were investigated: 10%, 20%, 30% by volume. The necessary amount of water and glass sand was precisely measured, then they were mixed before the column settling test. The first experimental series was performed to investigate the effect of vibration, to examine whether the vibration has any effect on the settling. The rod structure installed consisted of four 6-mm diameter rods. The vibration frequency was set to 50 Hz. Concerning the vibration parameters, only the amplitude was tested in this initial series and four different values were set. In the following table the adjusted amplitude parameters are shown as well as the marking of them used in all the following graphs.

In this first experimental series, 24 column settling tests have been carried out plus tests for checking the reproducibility (Fig. 42-47). The legend for all figures is as follows:

| Marking | Amplitude [mm] |
|---|---|
| ○ | No vibration |
| ◊ | ~ 0.2 mm |
| Δ | ~ 0.5 mm |
| □ | ~ 0.8 mm |

In the first free settling zone vibration almost does not have any effect on the settling, however, after the compression point, the highest amplitude vibration resulted in a much more compressed slurry. This phenomenon can be well seen in Fig. 42, but it could also be observed directly in the experimental column. After the tests using the high-amplitude vibration the removal of the slurry from the bottom of the column was very difficult. After free settling without vibration, the slurry could be easily poured out. This phenomenon of better compression through vibration is widely used in the industry. For example there are machines using vibration to remove water from wet soil, and vibration is used during the moulding of concrete for better compression.

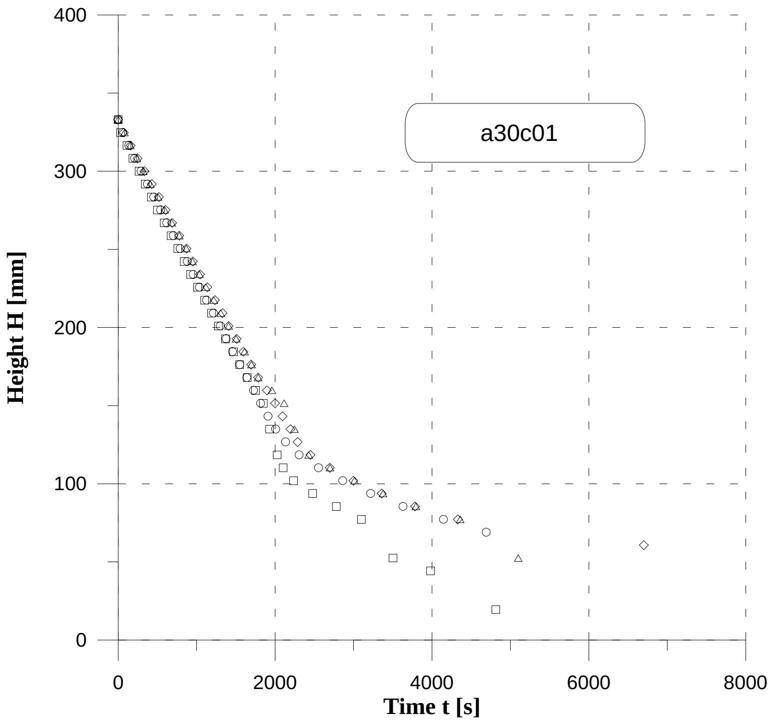



Figure 42: Settling curves of material a30, initial concentration 10%.

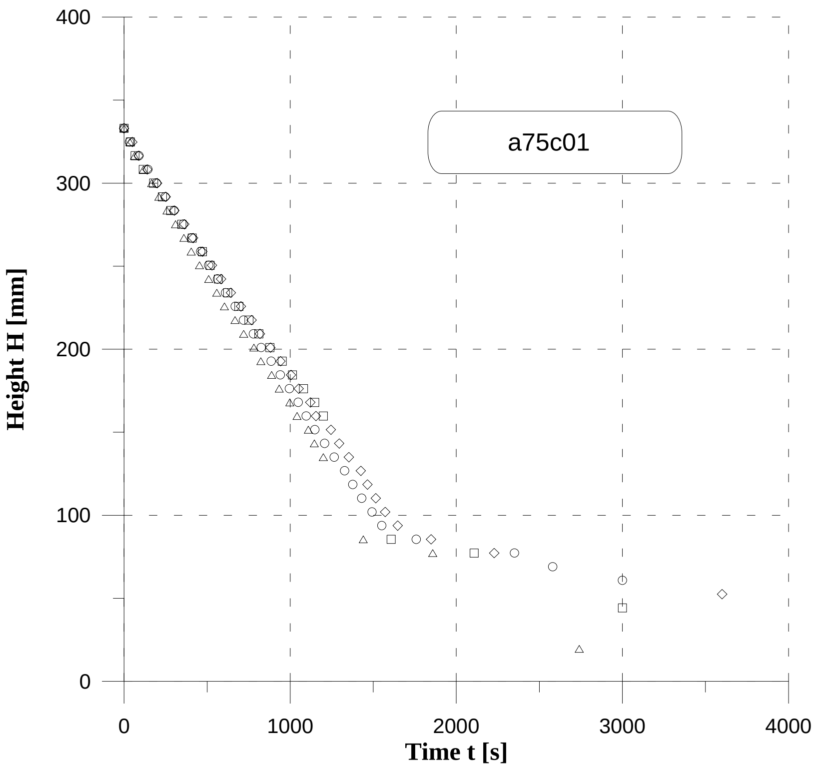



Figure 43: Settling curves of material a75, initial concentration 10%. Situation is very similar as in the case of the a30 material. A medium amplitude vibration slightly makes settling faster and vibration makes compression much stronger.

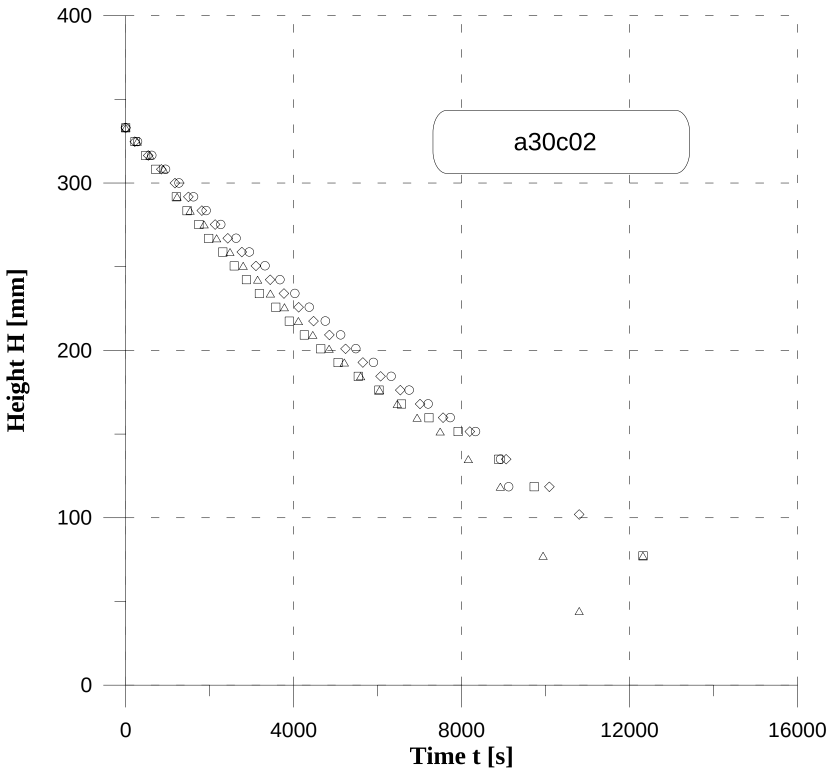



Figure 44: Settling curves of material a30, initial concentration 20%. In the free settling zone the effect of vibration increases. For example the slurry needs less time by 10% to reach the 200 mm level. In this zone the medium amplitude vibration is the best.

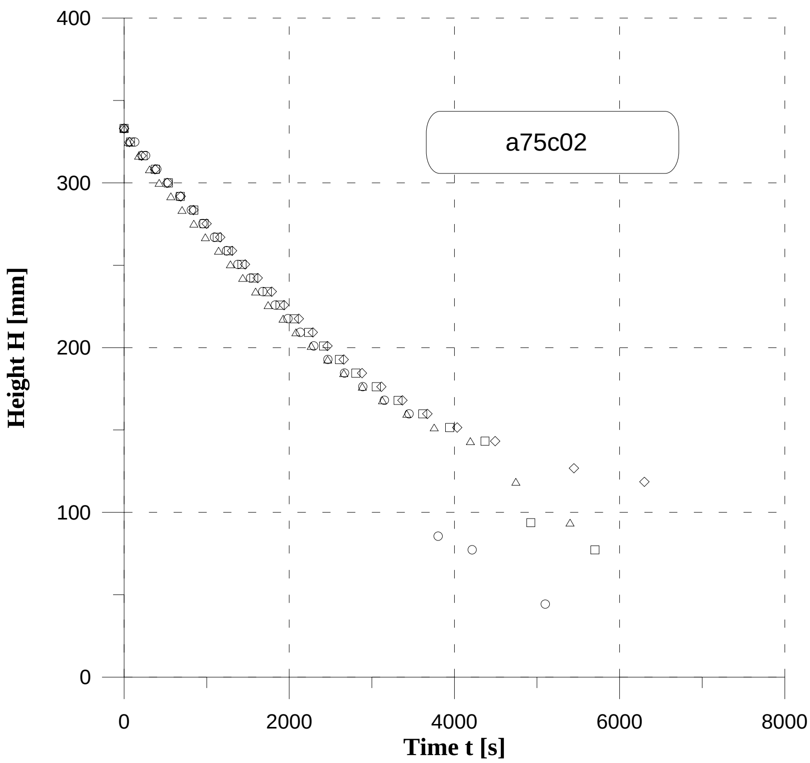



Figure 45: Settling curves of material a75, initial concentration 20%. For this coarser material a75, the vibration has less effect on settling in the free settling zone comparing to the finer a30 material.

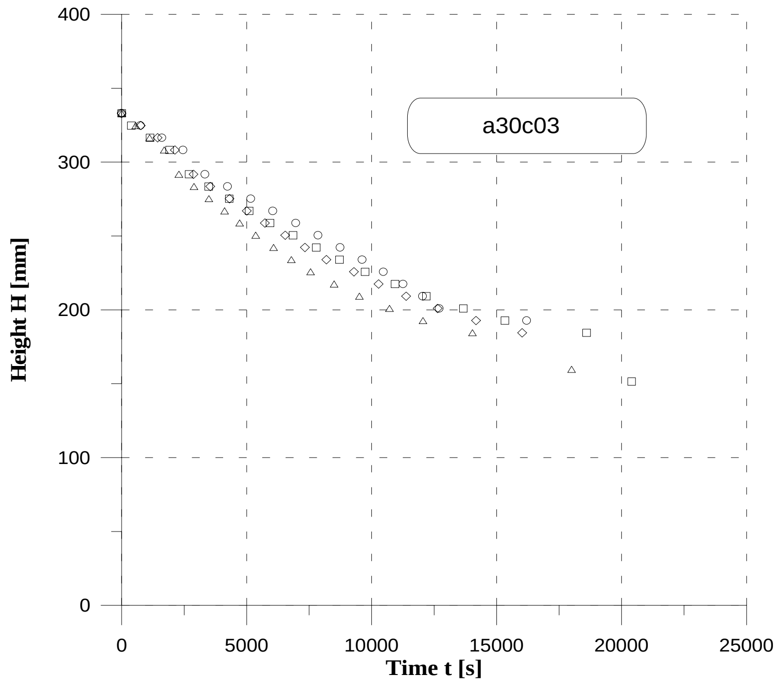



Figure 46: Settling curves of material a30, initial concentration 30%. Medium amplitude vibration has great effect on settling in the free settling zone. Settling is faster by about 30%. A thickener should design into similar circumstances. At this stage it is a conclusion that pre-thickening is necessary before the thickener with vibrating rods if the slurry is quite dilute.

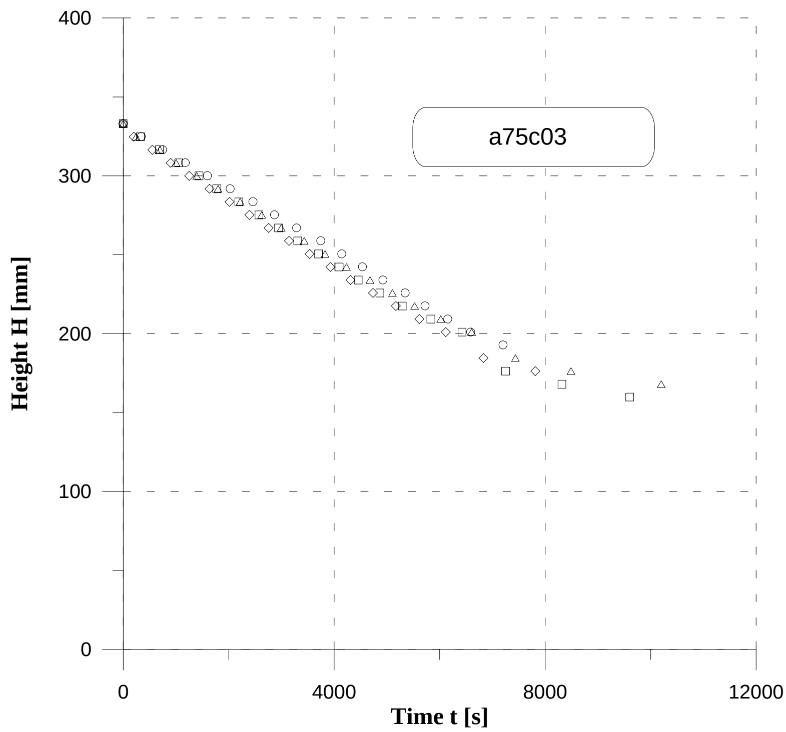



Figure 47: Settling curves of material a75, initial concentration 30%. For this coarser material vibration has less effect in the free settling zone.

### 4.5.2 Settling Experiments with Glass Sand and Bentonite

When the safety of tailings facilities is examined, questions about clay minerals have to be considered. The aim of this second experimental series is the examination of the effect of vibration into the settling behaviour of suspensions containing clay minerals; therefore, bentonite was mixed into our reference material. From the previous investigations it was concluded that the vibration increases the settling velocity of the finer particulate glass sand. Therefore in this investigation only the a30 material was applied and only the 20% of solids by volume concentration was set as initial concentration. Before mixing the solids with water, the following three discrete mixtures were produced:

| No. | Mass ratio of glass sand [%] | Mass ratio of bentonite [%] |
|---|---|---|
| 1. | 97 | 3 |
| 2. | 95 | 5 |
| 3. | 90 | 10 |

The parameters of vibration and the rod structure were the same in the previously described tests. In this second experimental series, 12 column settling tests have been carried out plus the tests for checking their reproducibility (Figs. 48-50). The legend for all figures is as follows:

| Marking | Amplitude [mm] |
|---|---|
| ○ | No vibration |
| ◊ | ~ 0.2 mm |
| Δ | ~ 0.5 mm |
| □ | ~ 0.8 mm |

As it was expected the bentonite reduced the settling velocity (Fig. 48). Vibration seems to have almost no effect on settling in the free settling zone. After the compression point vibration makes the compression stronger but the final concentration is lower than it was without bentonite.

As a conclusion of the experimentals with bentonite-containing mixtures it can be stated that if the clay content is high a thickener based on vibrating rods cannot be suggested.

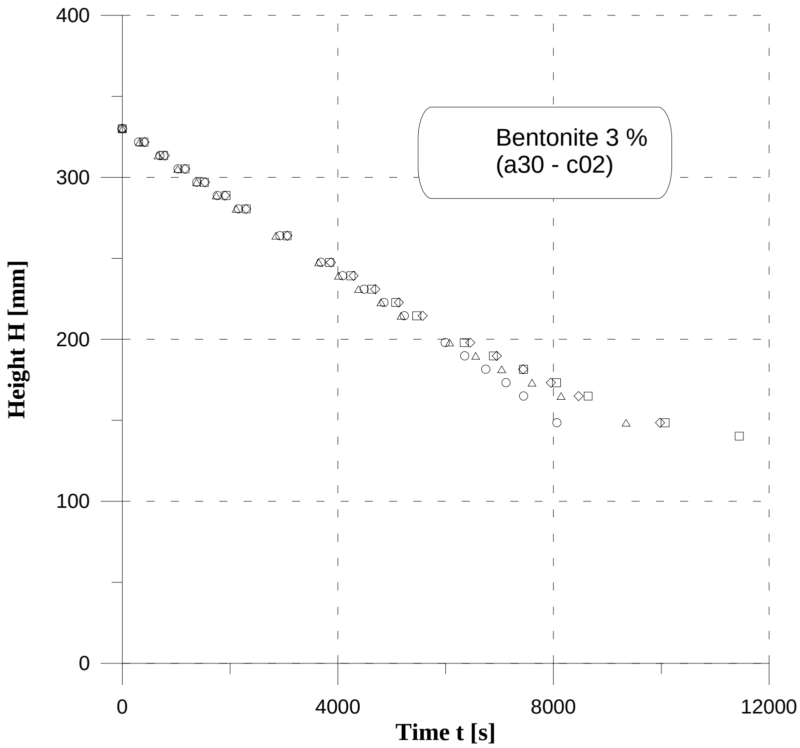



Figure 48: Settling curves of glass sand (a30)/bentonite (3%) mixtures, initial concentration 20%.

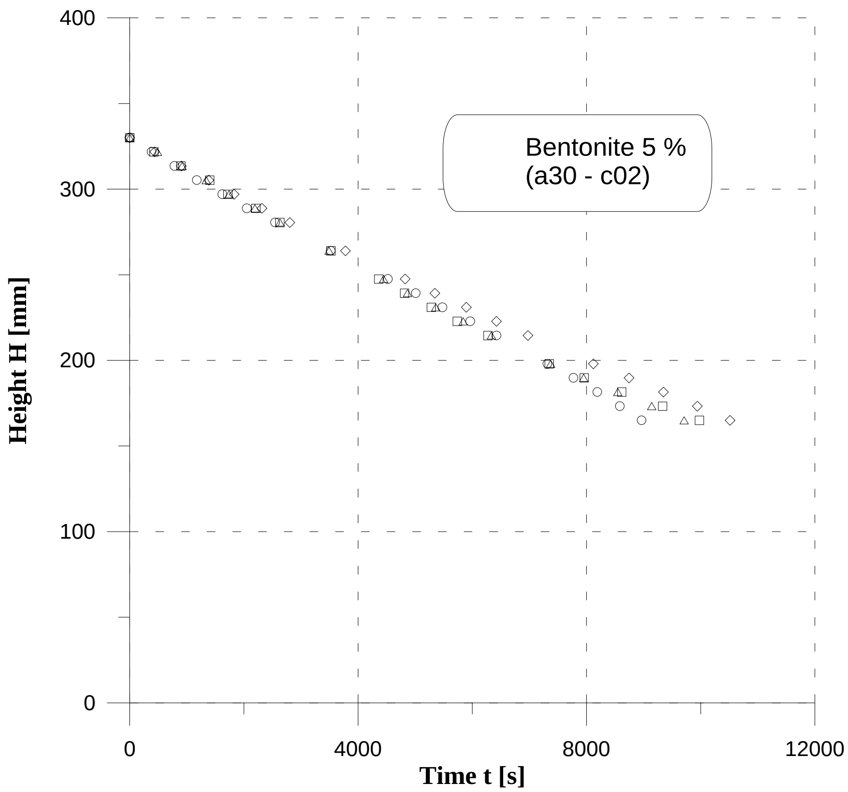



Figure 49: Settling curves of glass sand (a30)/bentonite (5%) mixtures, initial concentration 20%. The higher bentonite content makes settling even slower, but the effect of vibration is very similar to the previous case.

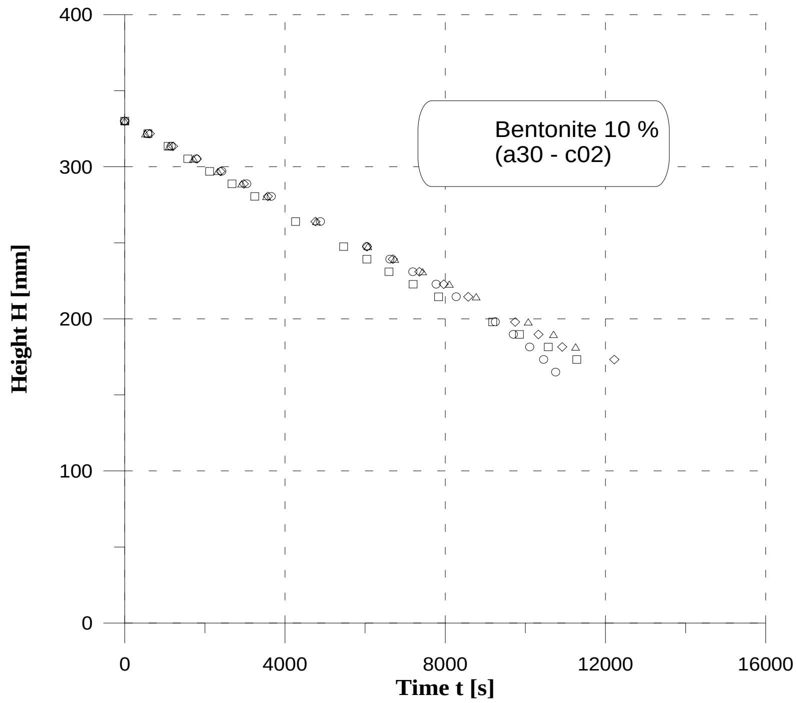



Figure 50: Settling curves of glass sand (a30)/bentonite (10%) mixtures, initial concentration 20%. The even higher bentonite content means slower settling, and the same situation with vibration.

### 4.5.3 Settling Experiments with Flocculating Reagent

In the introduction it was emphasized that the questions of flocculating additives have great industrial, technical and economical importance. Therefore it is quite interesting to examine what happens if vibration influences a settling system containing flocculating additives. The third experimental series investigated this phenomenon. During the settling column tests the reference material, glass sand was used only in an initial concentration of 30%. Before the investigation, several flocculating additives have been tested and the best was selected. It was the standard additive N300 (0.5 g of N300 was solved in 250 ml water). The parameters of vibration and the rod structure were the same as in the previous experiment series:

| Marking | Amplitude [mm] |
| --- | --- |
| ○ | No vibration |
| ◊ | ~ 0.2 mm |
| Δ | ~ 0.5 mm |
| □ | ~ 0.8 mm |

In this third experimental series, 4 column settling tests have been carried out (Fig. 51). As it was expected, settling in the free settling zone was much faster and a medium amplitude vibration made it even faster (by about 10%). Unfortunately, in the compression zone the flocculant hindered the compression, so the final concentration was not as high as it was without the flocculant.

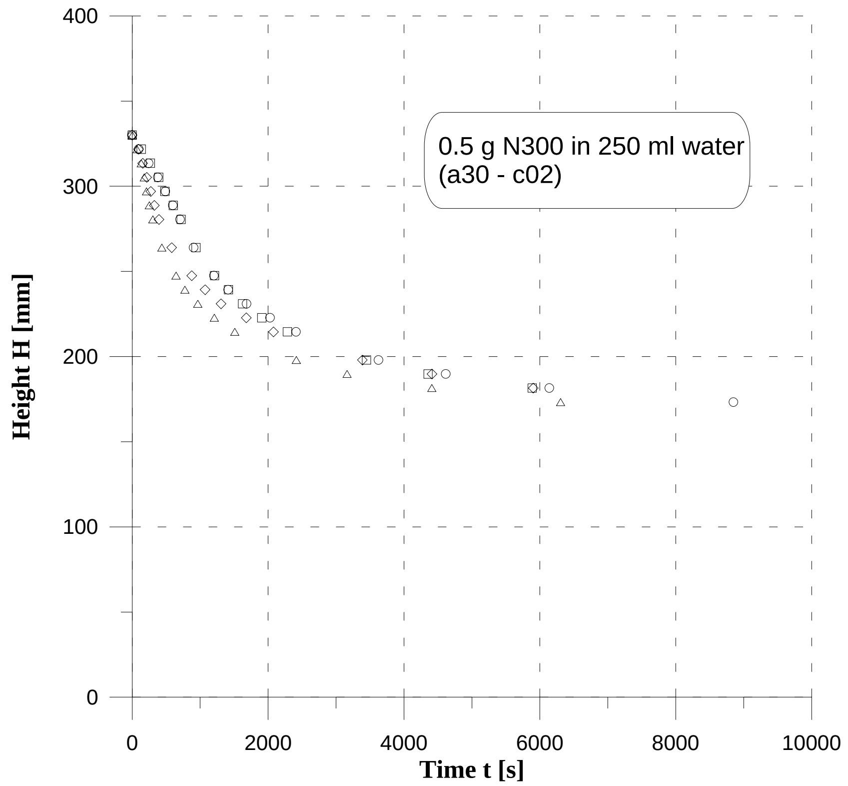



Figure 51: Settling curves of glass sand (a30) with the addition of the flocculant N300, initial solids concentration 30%.

## 4.6 Theoretical Considerations

The background of the systematic investigation of suspensions settling was the engineering point of view, to obtain the optimal values for those parameters that govern the design of real, full-scale equipments. However, it is also important to consider the various phenomena that are behind these very interesting results.

Vibration feeds energy into the settling system. This energy should not be too high because that would result in mixing instead of separation. In the free settling zone there is space for the particles to move. But the energy from vibration can disturb the laminar boundary layer around the settling particles and therefore, the drag force decreases and the settling velocity increases. The results of the first experimental series showed that a relatively high initial concentration (30% by volume) is necessary, and that a medium-amplitude vibration increases the settling velocity by 30%. The vibrating rods might simply arrange more space for the particles to move. In the compression zone this effect is probably even more significant. The vibration provides the energy for the particles to get better orientated, so depending on the shape of the particles they can pack stricter.

The very fine clay mineral particles might form a viscous media and by this way hinder the motion of the coarser particles as well. On the other hand this highly viscous media might decrease the efficiency of the energy transfer from the vibrating rods into the suspension.

The mechanism of flocculation is quite well known. The N300 additive highly catalyses the floc formation and by this way settling in the free settling zone is much faster. Unfortunately, the relative position of the formed flocs is fixed by the reagent and in the compression zone the particles cannot get really packed, leading to less water being displaced from the slurry. Our experiments showed that a medium amplitude at 50 Hz vibration made settling faster by further 10% in the free settling zone when N300 is added. This result is quite interesting and promising. In this case vibration might enhance the flocculating process by supporting the distributing of the reagent in the suspension.

## 4.7 Conclusions

In all those industries where huge quantities of residual materials are forming, the owners and operators are forced to provide for safe and environmental-friendly disposal of these materials. In the mining industry, fine-grained residues are usually disposed of in tailings management facilities. The amount of the water contained by the tailings is a key factor in many respects. Less water means lower costs for water management, higher safety for the tailings dam, and higher protection for the environment since less hazardous materials are solved and may be transported to the surroundings. As a consequence, slurry thickening, i.e. reducing the water content, is a key to better tailings management. Chapter 4 of the present report describes a computer aided method to design continuous thickeners on the basis of the traditional settling column test and third order spline curve fitting. For the sophisticated measurement of the settling characteristics a pilot-scale 40-l settling column has been developed in which the slurry height as well as the concentration distribution can be measured as a function of time.

Research was also focused on decreasing the necessary flocculating reagents. The principle of a new thickener was presented, where settling is supported by vibrating rods. A new pilot-scale settling column with a linear motor and different rod structures has been developed for the investigation of the effect of vibration into settling. This machine and test protocol is suitable to determine the optimal frequency, amplitude and rod structure for a given material for the design of the new vibrating rods thickener.

Using this pilot-scale settling column, three experimental series have been carried out, with the following summarised results:

1. Glass sand as reference material:
   - Vibration affects mainly the finer sand fraction.
   - In the free settling zone a medium size amplitude is more effective.
   - In the compression zone a high amplitude is more effective.
   - For higher feed concentrations vibration is more effective.
2. Glass sand/bentonite mixtures to investigate the effect of clay minerals:
   - Bentonite reduces the settling velocity.
   - Vibration reduces the settling velocity.

3. Glass sand and flocculant N300:
   - Medium amplitude vibration further increased settling in the free settling zone.
   - Flocculant hindered the compression of the settled solids.

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