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The relevance of water recirculation in large scale mineral processing plants with a remote water supply

Christian F. Ihle [a, b, *], Willy Kracht [a, b]

[a] Department of Mining Engineering, Universidad de Chile, Tupper 2069, 8370451 Santiago, Chile
[b] Advanced Mining Technology Center, Universidad de Chile, Tupper 2007, Santiago, Chile

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A B S T R A C T

Water and energy are essential requirements for mineral processing plants, and they can be either scarce or expensive. In the present paper, the implications of a remote water source location (typically the seashore) are analyzed in terms of a definition for the combined water and energy operational cost required to transport ore concentrate using long distance pipelines. A typical process route is defined to expose the relevance of key process variables, including source remoteness and water reclamation both from tailing storage facilities and thickeners. Both mass and energy balances have been made to give an estimation of specific energy consumptions for a typical large scale Chilean mineral processing operations, considering a number of realistic hydraulic design criteria to make a comparative analysis of different plant processing capacities. In the present cost function definition, an account has been made on whether the water is either makeup or has been recirculated on the plant, and a distinction has been made on whether tailings are conventional or thickened, which allowed to analyze the relevance of in-plant water recovery compared to water recovery from the tailing disposal site. This has allowed to express the cost of water in terms of the various energy cost terms, thus showing in this case that the cost of water is ultimately the cost of energy, and therefore all cost components become proportional to the unit cost of energy. A piecewise constant pipeline diameter analysis scheme has been defined to allow studying the implications of water recovery on widely different plant sizes. Results show that for fixed water recovery rates, increasing plant throughput tends to cause a decrease on the specific energy for water transport. On the other hand, for fixed pipeline diameters, increasing the throughput causes an increase on the specific energy consumption, but such increase rate becomes milder at higher throughput rates. This suggests the advantage of the existence of common makeup water supply systems for mining districts with more than one mineral processing plant. On the other hand, for constant pipeline diameters, increasing the water recovery from the tailing disposal site or from thickeners generates a decrease of water specific energy consumption, where in the case of the water recovery from the tailing disposal site such decrease rate becomes stronger with the amount of water recovered. Results also shows that, except for very high solid concentrations, water makeup costs dominate over recirculation ones, supporting that producing thickened tailings is a more efficient option than conventional ones in the sense of specific energy consumption, in spite of the additional energy costs due to slurry transport that this option implies.

1. Introduction

Water is ubiquitously used in mineral processing plants. While in the plant it is required for valuable mineral extraction, often *via* flotation, it is also used as a vehicle for both the transport of the commercial product (ore concentrate) and the residue (tailing). In either case water acts as a vehicle both for separation and transport, and is subject to losses in several stages in the process. It is therefore important to note that water, just like energy, acts as a raw material. Water and energy supply have been treated in the past as two separate problems the mining industry faces. Common approaches to the efficient use of resources have been focused on either efficient water management or life cycle analysis (LCA) to

* Corresponding author. Department of Mining Engineering, Universidad de Chile, Tupper 2069, 8370451 Santiago, Chile.
E-mail addresses: cihle@ing.uchile.cl (C.F. Ihle), wkracht@ing.uchile.cl (W. Kracht).

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both use less water and maximize recycling. Examples include Boger (2009), who focuses on the value of thickening and rheology to reduce water footprint, Norgate and Haque (2010), who suggests, using LCA, the need to improve the efficiency of major energy consuming equipment such as mills in the context of iron ore, copper and bauxite production. On the other hand, Dunne (2012) enhances the value of planning and the use of a system-level approach for water management, and Norgate and Haque (2012), using again LCA, expose the high environmental footprint of the gold extraction process. Only recently efforts have been devoted to treat the integral problem of efficient combined use of both. Indeed, only in recent years the notion of decision making after trade-off values related to water and energy has received attention in the literature. An example is the Mine Water Network Design (MWND) approach (Gunson et al., 2010) where, after the identification of the water balance and potential water sources, an optimization of the water consumption based on minimizing energy requirements *via* linear programming has been applied to water cooling of mills and compressors in a mineral processing plant. In the coal industry, the impact of the mine plan on potential decisions to optimize pumping capacity for dewatering has been depicted in terms of both the specific energy and water consumption in India (Sahoo et al., 2014) and. The opportunities to use alternative water sources increasing global efficiency is discussed by (Nguyen et al., 2014, and references therein). This has included the potential to use another mine's acquifer. The concept of water transfer between plants has been exploited in the hierarchical system model that has been proposed and applied to an Australian plant network to minimize water contamination due to overflow in plants during extreme events (Gao et al., 2016). In these cases, the explicit metric of interest is either energy or water consumption.

The interplay between water and energy use has been only relatively recently identified as a plant efficiency driver. Conceptual approaches have been proposed to expose trade-offs between water and energy consumption in mining. Donoso et al. (2013) have identified, in light of a simulation of a copper concentrator, the relevance of particle size through comminution and water recovery. They observed that larger particles are bonded to reduced energy consumption, and increased water recovery at the thickeners. However, this optimum does did not match the highest copper concentrate throughput, making this approach strongly copper and energy cost-dependent. Assuming comminution and flotation as process data, a water energy index, corresponding to the ratio of available water volume and energy demand (Nguyen et al., 2014) and a dimensionless parameter for water-energy cost in slurry transport systems, including ore concentrate and tailings (Ihle et al., 2014) have been defined. These approaches have in common the requirement of a site-specific assessment (including cost and location) for a best combined use of both resources. Apart from desalination, water supply cost and associated greenhouse gas emissions is strongly tide to delivery distances (Hiam-Galvez et al., 2012; Ihle, 2014). To account for a metric of the ore concentrate transport cost, an energy/water cost formulation can be used to measure how energy- and water-intensive is the transport process (Ihle, 2013; Ihle et al., 2013a). This approach requires a water and energy cost separately. The result after an optimization procedure is a set of optimal concentration and flow conditions given both the unit costs of water and energy. Adiansyah et al. (2016) have worked an example taking into account the combined relevance of water and energy in a coal plant using a hierarchical system model (HSM) software to seek for the best tailing transport and disposal option for thickening solids concentrations between 30% and 60% by weight. They conclude that considering water recovery rates and energy consumption there is an optimum close to 50% (in their study the most water saving option was not the best due to the high energy consumption). Although their computation did not add a metric for the cost of water and they took a qualitative valuation of the use of water based on recovery, they reinforce the point made in previous works (e.g. Ihle, 2014) regarding the relevance of considering both water and energy in plant assessment, with emphasis on the impact of water slurry transport.

The Chilean copper industry is massive on both water and energy consumption and faces several challenges involving water that range from economical to environmental aspects. While it shares the common water scarcity and/or competition with other stakeholders (Cochilco, 2008), it also needs to minimize the use of energy in the especially high energetic Chilean cost scenario (on the order of 100USD/MWh as reported by Utreras et al., 2016), that implies CAPEX can be as high as 19USD/m^3 and OPEX 3.5USD/m^3 (Soruco and Philippe, 2012). On the other hand, in Chile, the water scarcity close to many mineral processing plants in arid regions, even when compared to other mining countries with present and potential of future development, such as Peru and USA, has created the need to supply water from the sea (Northey et al., 2017), which somewhat imposes a new challenge: when resourced from the sea water should be kept to a minimum, not because it is scarce, but because desalination and transport is considerably expensive (Northey et al., 2013). In particular, there is a high energy demand associated to both pumping water from sea level to mountain locations, where most of the concentrators are, and the need to provide the supplementary energy to account for friction losses. Table 1 shows an example of long distance water transport lines connecting desalination facilities with mineral processing plants. A complement is detailed in Philippe (2012), Castillo et al. (2015) and Revista Agua (2016), whereas historical remarks on water supply pipelines for the mining industry can be found in Ghassemi and White (2007). On the other hand, a summary of potential desalinated water supply projects in the context of the Chilean mining is given by Zúñiga (2009). In the particular case of slurry pipelines, some additional operational challenges arise from the handling of segregating solid matter inside the pipeline. In several cases, paths for desalinated (or salt water) pipelines are the same than those of ore concentrate pipelines, which means that part of the water obtained from sea level travels twice the pipeline length. Both in concentrate and makeup water pipelines, power consumptions due to pumps are commonly in exceed of 1 MW. This number can be readily justified considering as a rough (and conservative) estimation of the hydraulic power for transport that it is on the order of the product of the water density, the acceleration of gravity, the altitude difference and the volume flow (i.e. neglecting friction losses and pumping efficiencies). Excluding the Balama pipeline case in that table, more than 70% of the operations in Table 1 consumes more than 1 MW in water transport.

In slurry transport systems using pipelines, from a simple mass balance, discussed below, there might be more than one combination of slurry flow and solids concentration which yields the same throughput. This suggests that it is possible to find, within the set of feasible input conditions, an optimal one given a specific criterion. If such criterion is specific energy consumption (*i.e.* the amount of energy spent per unit mass of solids), the least specific energy consumption occurs near the minimum velocity before a sediment bed appears at the pipeline section (Ihle and Tamburrino, 2012). The minimum velocity is defined as the maximum value between the deposition value and that corresponding to the laminar-turbulent transition, where most often the former controls the minimum velocity condition at small to moderate solid fraction values (Ihle and Tamburrino, 2012). In other words, in most operational long distance ore concentrate pipelines, the least specific energy consumption is controlled by the deposition velocity value.

Table 1
Some water long distance water pipelines serving mining operations. Altitudes are referential.

Mining operation (mineral)	Country	Altitude (m.a.s.l)	Length (km)	Flow (L/s)	References
Algorta (iodine)	Chile	1300	65	150	González et al. (2012)
Balama (graphite)	Mozambique	550	15	–	Davis (2017); Mining Magazine (2017)
Bao Tou (iron)	China	1065	130	694	Ausenco (2010)
Candelaria (copper)	Chile	650	80	300	Quinn (2011)
Cerro Negro Norte (iron)	Chile	1000	82	600	RETEMA (2014); Baralla (2015)
CVWS[a] (copper, gold)	Chile	1045	117	600	Revista Electricidad (2009); González et al. (2012)
Esperanza (copper)	Chile	2200	145	2500	Martínez (2014)
Impunzi (coal)	South Africa	600	1	521	Armitage and Baxter (2012)
Lomas Bayas (copper)	Chile	1700	120	500	Minería Chilena (2011)
Manto Verde (copper)	Chile	840	40	120	Riffo (2008); González et al. (2012)
Mantos de la Luna (copper)	Chile	1200	8.5	83	González et al. (2012)
Michilla	Chile	835	17	80	González et al. (2012)
Minera Escondida	Chile	3100	170	2500	Casares (2006), Montes (2016)
Olympic Dam (polymetallic)	Australia	100	320	2315	BHP (2017)
Salt River (coal)	United States	200	173	219	Klein (2006)
Santo Domingo (copper)	Chile	1150	112	355	Alfaro (2014)
Sierra Gorda (copper)	Chile	1615	143	1500	Minería Chilena (2014); García (2015)
Sino Iron Ore (iron)	Australia	50	25	1620	González et al. (2012); Mining Technology (2017)
Southdown Magnetite (iron)	Australia	600	24.5	381	Clayson (2011)

[a] Copiapó Valley Water Supply pipeline.

When the objective function is, instead of the specific energy value, the combined water and energy cost, the conclusion is similar: only at extremely high concentrations the least (water + energy) cost is driven by laminar-turbulent transition condition and, in general, it is driven by the deposit limit which, in this case, corresponds to the highest feasible transport concentrations (Ihle, 2016). This general requirement of moving to higher concentrations from a simple cost perspective coincides with the known problem of the mining industry to minimize the use of water for cleaner production (in the sense of Hilson, 2003) in a framework of best sustainable operational practices (Adiansyah et al., 2015). The disposal of these, so-called thickened or paste tailings, whose name will actually depend on the flow properties of the slurries, has well-known environmental advantages (Boger, 2009; Simms, 2017), and is not necessarily more expensive than conventional technologies in greenfield projects. This conclusion has been obtained in the review by Fourie (2012b) and, more recently, by Aitken et al. (2016), who estimated costs in a 50000ton/day ore plant. They found, for their particular case, that high rate thickening with a secondary flocculation stage implies a lower water recovery cost than paste and filter technology. Modern semi-quantitative approaches to account for the efficiency in water consumption at plant level include to consider standard water footprinting procedures based on lifecycle analysis (Northey et al., 2016). However, information regarding the energy use in processes including makeup from distant locations is often not available (Northey et al., 2013).

The present paper focuses on the impact of a remote water supply on an *ad-hoc* definition of water costs of a typical mineral processing route. Attention is payed on the role of such remote makeup water replacing losses from the tailing storage facility (TSF) (a thorough discussion, centered on water saving from other loss origins, has been made by Gunson et al., 2012). The potential for water recirculation from the concentrate filter plant is analyzed as well. It is shown herein that, under the imposed conditions, the unit cost of water for transporting ore concentrate is proportional to the corresponding unit cost of energy. A simplified version of the mass balance of a mineral processing plant is detailed herein to analyze and parameterize the most important elements that decide such ore concentrate energy and water transport cost. In particular, the role of thickening is exposed from an operational cost perspective.

2. Problem description

2.1. Flow diagram

Consider the flow diagram of Fig. 1, consisting of a typical concentrator plant dispatching ore concentrate, and generating residual material dumped on a tailing storage facility (referred to herein as TSF). The arrows represent the corresponding stream of species (water, ore concentrate or tailing) connecting the various operations depicted by squares. The leftmost braces represent two distinct locations, namely the plant site area (in Chile, typically, in the Andes Mountains), and the water supply source (commonly at sea level, distant dozens of km from the latter). The problem is to analyze the impact of in-plant water recovery, at the TSF, in an *ad-hoc* definition of ore concentrate transport cost, projected in water and energy components, assuming that water transfer occurs in close conduits that preclude leaks or evaporative losses. In-plant water losses are also neglected in front of those at the TSF. This is discussed below.

In Fig. 1, water losses from thickeners and flotation cells due to evaporation have been omitted for simplicity. These are, in general, considerably small when compared to those at tailings disposal sites, excluding filtered tailings (not studied herein), because of the wide difference between wet and humid areas exposed to the environment in both cases. This schematic also shows the presence of a plant area and an off-site one. Several Chilean large scale mining operations rely on remote seawater sources, which imply the requirement of several of dozen kilometer pipelines both for the ore concentrate and the makeup water. It is exposed herein the relevance of this fact on a metric of transport, water recirculation and makeup cost. Such cost definition is described in Section 2.2.

2.2. Ore concentrate transport cost function

In the present work the ore concentrate is assumed as the sole product of the mineral processing plant. The role of water in operational cost is to participate in comminution (wet grinding), concentration, transport and tailing disposal operations. The potential for water savings within the plant are out of the scope of the present paper, and the focus is placed on the role of water on the costs associated to transport and tailings disposal.

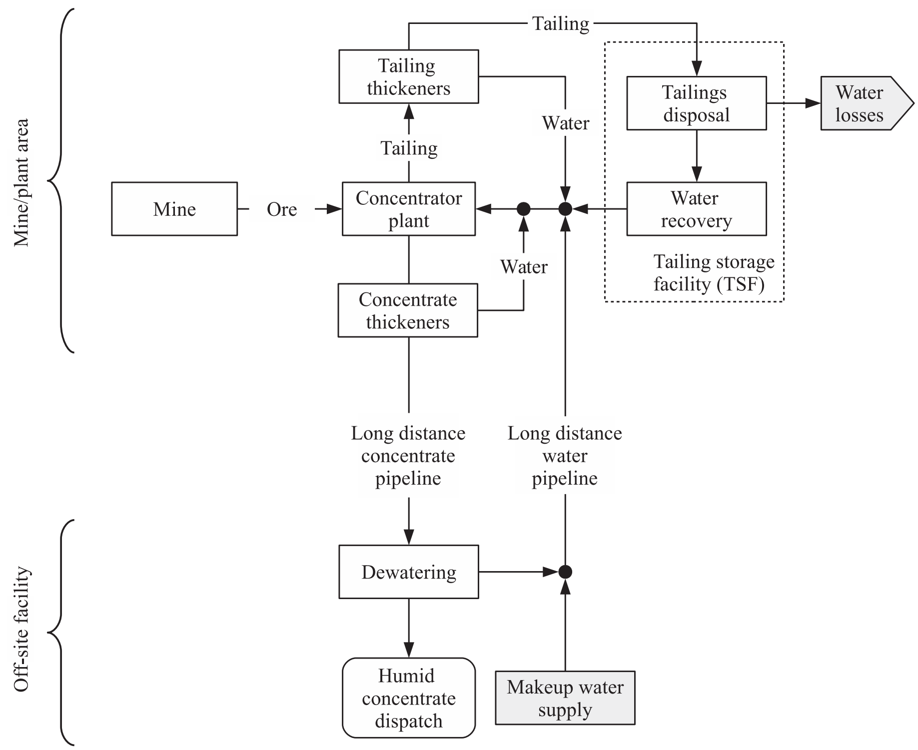

Fig. 1. Plant schematic. Water losses from the ore concentrator plant have been omitted. Both the water makeup supply and the water losses from the TSF have been highlighted in gray.

From the product perspective, a transport cost function for the ore concentrate can be written as (Ihle, 2013):

$$\Omega = Q_{WC}\theta_W + P\theta_E, \tag{1}$$

where the units of Ω are currency/time, Q_{WC} is the water flow component in the concentrate (in volume of water/time), P is the pumping power (energy/time) and the unit cost variables are θ_W and θ_E for the water cost (currency/water volume, *e.g.* USD/m^3) and the energy cost (currency/energy, *e.g.* USD/MWh). The energy unit cost, θ_E, is a constant of the problem. In contrast, the water unit cost requires a more diverse origin. In particular, the water cost comes from:

1. Makeup (θ_{WM}), including:
   (a) Desalination, when present
   (b) Transport from the filter plant or the desalination plant area to the mineral processing plant
   (c) The potential recovery of filtered water using the pumping capacity of the desalination plant and the pipeline
2. Water reclamation from or near the plant area
   (a) Thickeners (both concentrate and tailings)
   (b) TSF (total, including water pond and contour channel reclamation).

Among the most important environmental problems related to water management in mineral processing operations is precisely the amount of makeup required, which gives a measure of the lack of closure of the mass balance in the system. In the following section, the water balance for the proposed system is described and the various terms are interpreted in light of a cost item to eventually configure the water cost, θ_W.

3. Results and discussion

3.1. Mass balance

Centering a water mass balance in the mineral processing plant, the following relation holds:

$$Q_{WM} - Q_{cons} - Q_{loss} = 0, \tag{2}$$

where Q_{cons} and Q_{loss} are the water flows corresponding to consumption and water losses, respectively. The term consumption points to the use of water that ships with the ore concentrate (Wills and Napier-Munn, 2011) and does not return to the cycle, and thus needs to be supplied externally, as part of the makeup. On the other hand, Q_{loss} corresponds to seepage and evaporation losses, which in principle can be minimized (Boger, 2009) but hardly completely eliminated (some figures on evaporation rates may be found in Bleiwas, 2012). In ore concentrate plants, evaporation losses occur mainly at the TSF and the surface of the thickeners. However, in most Chilean mineral processing plants it is easy to verify using satellite imaging that evaporation areas in thickeners are, in most cases, between 1% and 3% of the effective evaporation areas at TSF's, potentially reaching hundreds of hectares in plants processing more than 100000tons/day, as can be directly inferred from Gunson et al. (2012), and also observed from satellite images.

Considering steady-state operation, the water balance in the concentrator plant is given by the relation

$$Q_{WM} + Q_{TDR} - Q_{CTU} - Q_{TTU} - Q_{other} = 0, \tag{3}$$

where the subscripts denote water makeup (*WM*), water reclamation from the TSF (*TDR*), concentrate thickener underflow (*CTU*), tailing thickener underflow (*TTU*) and other losses, such as evaporation from thickeners (*other*). The water flow recovered from the

TSF, Q_{TDR}, may be expressed as a fraction of the underflow of the tailing thickener, as:

$$Q_{TDR} = \lambda_{TDR} Q_{TTU}, \tag{4}$$

with $0 < \lambda_{TDR} < 1$. The term Q_{other}, related to water losses within the plant may be neglected given its little relevance compared to water evaporation and/or by seepage at the TSF. Using (4), it is therefore assumed:

$$Q_{WM} \approx Q_{CTU} + Q_{TTU}(1 - \lambda_{TDR}), \tag{5}$$

which allows obtaining an order of magnitude of the impact of the water losses at the TSF in the water balance of the plant. It is noted that, when desalination plants are present, part of the water corresponding to the concentrate stream is also recovered. This has not been considered at this point because such recovery occurs at a distant location from the mineral processing plant (water recycling is defined here as the in-plant or near-plant recovery of water). To make it possible, considerable energy cost, related to a long distance transport, must be taken into account. For this reason, such stream has been considered as water makeup, although subtracting desalination costs. This is discussed in section 3.2.

On the other hand, the TSF is a variable mass system, as it accumulates solids and residual water content in the tailing. A schematic is shown in Fig. 2. This residual water content in the TSF can be on the order of 15% of the solid in Chilean copper sulphide deposits (Osorio, 2009; Conejera and Pasten, 2016, and references therein). Depending on the presence of clay content, this value can be considerably higher (e.g., the measurements reported by Wels and Robertson, 2003) for a Chilean copper mine in the desert.

The corresponding mass balance at the TSF for both the water (m_l) and the solid content (m_s) is given by the following expressions:

$$\frac{dm_l}{dt} = \rho_W Q_{TTU}(1 - \lambda_{EI} - \lambda_{TDR}) \tag{6}$$

$$\frac{dm_s}{dt} = G_{s,TSF}, \tag{7}$$

where Q_{TTU} is water part of the thickener underflow stream, ρ_W is the density of water, λ_{EI} is the fraction of the water entering the TSF that is lost due to evaporation and infiltration and $G_{s,TSF}$ is the solid part of the tailing disposed in the TSF. The residual content of water in the TSF (neither lost to the environment nor recovered to the mineral processing plant) can be interpreted as a progressive water accumulation. Given the liquid-to-solid mass retained fraction, λ_{ret} and the expression for the water accumulation $dm_l/dt = \lambda_{ret} G_{s,TSF}$.

In Chilean copper concentrate plants, typical ore grades are 1% (feed, denoted fg), 0.1% (tailings, denoted tg) and 28% (Gunson et al., 2010) (concentrate, denoted cg) (a progression of Chilean copper ore feed grades is detailed in Ocaranza, 2017). It is thus noted that, given a plant feed throughput, G_F (*i.e.* the solid material entering the plant, and a weight recovery Y, which can be calculated from the grades as $\frac{fg-tg}{cg-tg}$), the concentrate throughput, equal YG_F, is on the order or less than 2% of the total amount of mineral processed (Northey et al., 2014), with associated tailing deposits which commonly exceed volumes of 1 billion cubic meters after few decades of operation (Rico et al., 2008; Icold, 2017). From a water balance, the terms Q_{CTU} and Q_{TTU} are straightforward to obtain:

$$Q_{CTU} = \frac{YG_F}{\rho_W}\left(\frac{1}{C_{CTU}} - 1\right) \tag{8}$$

$$Q_{TTU} = \frac{(1-Y)G_F}{\rho_W}\left(\frac{1}{C_{TTU}} - 1\right), \tag{9}$$

where C_{TTU} and C_{CTU} are the solid fractions by weight at the underflow of the tailing thickeners and in the ore concentrate pipeline, respectively. In both cases, the water flow at the thickener feed is assumed to be controlled by the same feed concentration as it is shown below for the case of the concentrate:

$$Q_F = \frac{YG_F}{\rho_W}\left(\frac{1}{C_F} - 1\right), \tag{10}$$

where C_F is typically on the order of 30% (Gunson et al., 2012).

In spite there is wide international agreement on the need to produce tailings with reduced amounts of water (Davies and Rice, 2001; Boger, 2009; Edraki et al., 2014; Northey et al., 2016), there is a considerable number of mineral processing plants operating within the so-called *conventional range*, with $C \approx 50\%$, which makes the present analysis especially relevant in that context.

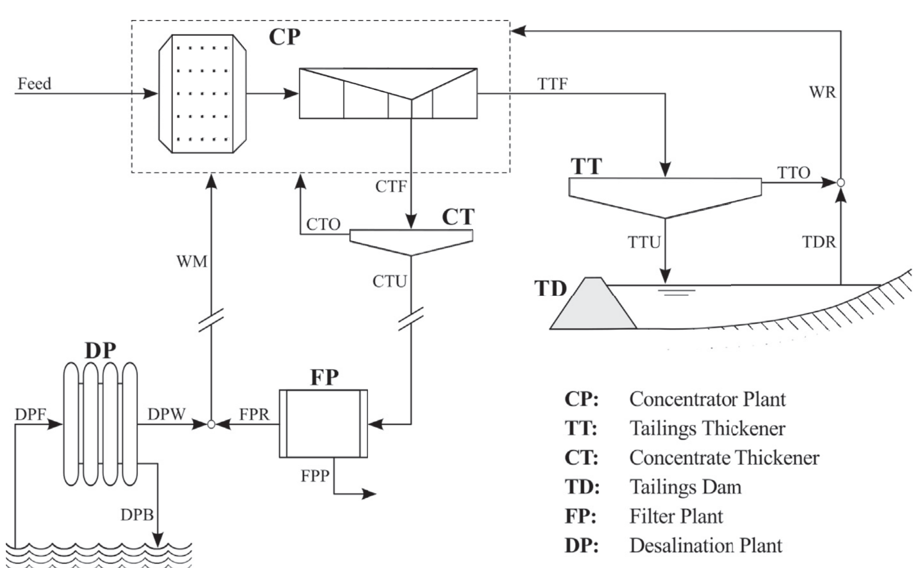

Fig. 2. Schematic of the flows at the TSF. It is noted that both solid and liquid accumulation occurs.

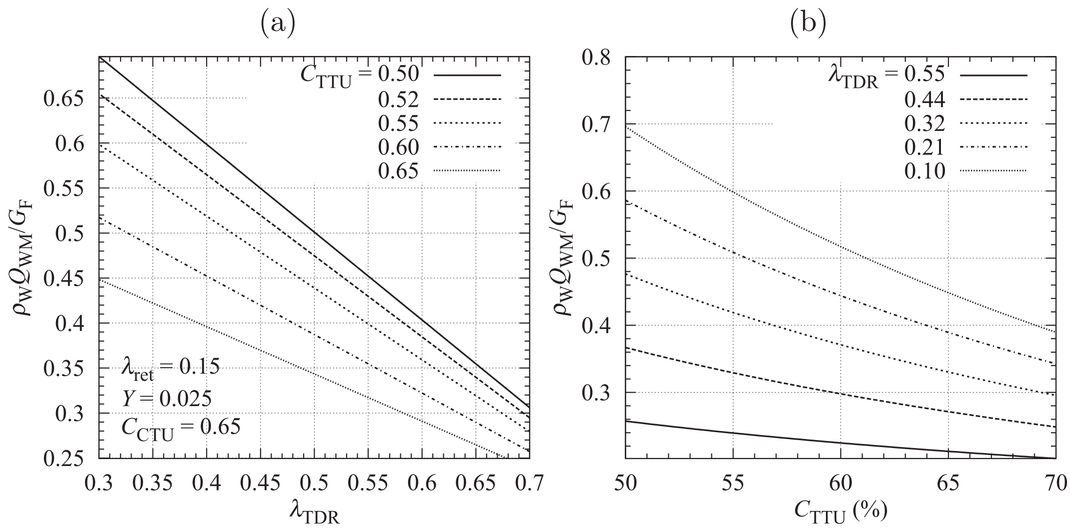

Fig. 3. Makeup water mass flow normalized by plant throughput for $\lambda_{ret} = 15\%$, $Y = 12.5\%$, $C_{CTU} = 65\%$. (a) As a function of the fraction of water recovery from the TSF and (b) in terms of tailings thickener underflow concentration.

The solid part of the tailings disposed at TSF sites verify the relation $G_{s,TSF} = (1 - Y)G_F$. This implies that the water retention rate corresponds to $dm_l/dt = \lambda_{ret}(1 - Y)G_F$ and that, by virtue of (7) and (9), the following relation holds:

$$\lambda_{TDR} = 1 - \lambda_{EI} - \frac{C_{TTU}\lambda_{ret}}{1 - C_{TTU}}. \tag{11}$$

The water make up flow can thus be computed using (5) and (8) as:

$$Q_{WM} = \frac{YG_F}{\rho_W}\left(\frac{1}{C_{CTU}} - 1\right) + \frac{(1-Y)G_F}{\rho_W}\left[\left(\frac{1}{C_{TTU}} - 1\right)\lambda_{EI} + \lambda_{ret}\right] \tag{12}$$

$$= \frac{YG_F}{\rho_W}\left(\frac{1}{C_{CTU}} - 1\right) + \frac{(1-Y)G_F}{\rho_W}\left(\frac{1}{C_{TTU}} - 1\right)(1 - \lambda_{TDR}). \tag{13}$$

Here, instead of expressing results in terms of the amount of water lost to the environment and within the TSF due to water retention, it is considered hereafter λ_{TDR} as single control parameter for the water makeup, as in (13). Knowing the retention rate, an estimation of the losses due to evaporation and infiltration in the system can thus be estimated using (11).

Fig. 3 shows the makeup normalized by the plant throughput. This dimensionless result can be expressed in the more familiar index of liters of water per tonne of processed mineral by multiplying the ratio $\rho_W Q_{WM}/G_F$ by 1000. For the relatively typical case of $C_{TTU} = 52\%$, often corresponding to conventional copper tailings[1] (Fourie, 2012a), it is seen that increasing the water recovery from the TSF from 35% to 50% of the underflow of the tailing thickener causes additional water savings on the order of 120 L/ton, or about 24.5% compared to a typical makeup water use of 490 L/ton (Ocaranza, 2017) (Fig. 3a), which in a large scale mineral processing plant (processing 100000ton/day of mineral or more) might imply water savings on the order of 10000m^3/day depending on the size. On the other hand, for a fixed water recovery ratio, the $1/C_{TTU}$ dependence of the makeup makes that for smaller concentrations the impact of thickening on water makeup is stronger. For instance, increasing the tailings thickener underflow concentration from 50% to 55% when recovering about 30% of the water from the TSF causes water savings of about 60 L/ton (Fig. 3b). In a higher concentration range, decreasing water consumption becomes more difficult and, as expected, the best opportunity for water savings appears through λ_{TDR}.

The solids concentration at the concentrate thickener underflow, C_{CTU}, is similar to C_{TTU}, the concentration at the tailing thickener underflow. In contrast, the parameter Y differs significantly from $1 - Y$. Taking this into account, comparing the first and second terms of the right hand side of Eq. (13) it is concluded that most of the makeup is associated to tailings rather than ore concentrate. Considering the dispatch of concentrate downstream the filter plant as a minimum requirement for water makeup, from a water balance in the filter plant, an absolute minimum for Q_{WM} is $Q^*_{WM} = \frac{YG_F}{\rho_W}\frac{H}{1-H}$, where H is the cake humidity of the dispatched concentrate (in humid base), typically on the order of 8% (e.g. Concha, 2014; Chap. 9). Therefore, in practice, $Q^*_{WM} \ll Q_{CTU} \ll Q_{TTU}$.

The generation of thickened and paste tailings has been indicated as the best technique for tailings disposal both in the sense of physical/chemical stability and from the point of view of water savings (e.g. Boger, 2009; Fourie, 2012b, and references therein). However, the hydraulic transport costs of paste and thickened tailings is energy-intensive compared to water or conventional tailings. In existing plants where infrastructure is suited for the generation of conventional tailings and it is not economically feasible to switch for high dewatering systems, an alternative strategy to maximize water recirculation is to increase the water recovery from the TSF as depicted in Fig. 3. In the forthcoming sections it is shown that, if possible, the best option is related to maximizing water recovery from the thickener.

3.2. Ore concentrate transport cost components

Following the cost function definition described in Section 2.2, the water cost per unit volume, θ_W (Eq. (1)), is written as the weighted sum of the makeup cost and the in- or near-plant recirculation cost:

[1] Strictly speaking, the classification is rather dependent on the rheology of the slurries than on the solids concentrations, but the values referred to herein are commonly found in large scale copper sulphide tailing facilities. The use of mass concentrations instead of the yield stress (Boger, 2009) is useful to center the analysis on the mass balance expressions, due to the water-saving oriented focus of the present paper.

$$\theta_W = \theta_{WM}\lambda_{WM} + \theta_{WR}\lambda_{WR}, \tag{14}$$

$$\lambda_{WM} + \lambda_{WR} = 1. \tag{15}$$

Here, the subscript W stands for water, WM for makeup and WR for water recirculation. It is noted that, with regard to water recovery, in the context of the present work, the concepts recirculation and reclamation are used with the same meaning. On the other hand, the parameters λ_{WM} and λ_{WR} denote the fraction of makeup water and recirculated water. They correspond to the fractions of makeup and recirculation of the total volume flow, respectively.

The cost dimension that depends on water reclamation, θ_{WR}, is expressed as follows:

$$\theta_{WR} = \left(\widehat{E}_{TDR}\lambda_{TDR} + \widehat{E}_{TT}\lambda_{TTO} + \widehat{E}_{CTO}\lambda_{CTO}\right)\rho_W Q_{WR}\theta_E \tag{16}$$

$$\lambda_{TDR} + \lambda_{TTO} + \lambda_{CTO} = 1, \tag{17}$$

with the specific energy components (with units of energy per recovered water unit weight, energy/kg$_{water}$) corresponding to the TSF ($\widehat{E}_{TDR}$), tailings thickener array overflow ($\widehat{E}_{TT}$) and concentrate thickener array overflow ($\widehat{E}_{CTO}$). The term $\widehat{E}_{TT}$ does not only include the energy input related to the recirculation from the tailing thickener overflow, but also considers the energy used for pumping thickened or paste tailings from the thickener underflow to the TSF, which is indirectly a cost of recovering water. The weighting factors multiplying the specific energy components correspond to the ratio between the water volume flow of the stream of interest and the total recirculated flow. The latter is assumed herein as the sum of the three water recovery components, associated to the overflow of the tailings and concentrate thickeners and the TSF:

$$Q_{WR} = \frac{G_F}{\rho_W}\left\{\frac{1}{C_F} - \frac{Y}{C_{CTU}} - \frac{1-Y}{C_{TTU}}\left[1 - \lambda_{TDR}(1 - C_{TTU})\right]\right\}, \tag{18}$$

On the other hand, the makeup unit cost, θ_{WM}, is expressed as:

$$\theta_{WM} = \left[\left(\widehat{E}_{MD} + \widehat{E}_{WM}\right)\lambda_{MD} + \widehat{E}_{WM}\lambda_{MC}\right]\rho_W Q_{WM}\theta_E \tag{19}$$

$$\lambda_{MC} + \lambda_{MD} = 1, \tag{20}$$

where $\widehat{E}_{MD}$ and $\widehat{E}_{WM}$ are the specific energy requirements for desalination and makeup transport, respectively. They are measured in energy/kg^{Water}. Such specific energy requirements are applied to different streams. The weighting factor λ_{MC} the stand for the volume flow fraction of the makeup that comes from the filtered concentrate, and consequently does not need desalination. On the other hand, the weighting factor λ_{MD} corresponds to the part of the makeup that requires desalination.

In case there is an available pipeline to pump the recovered water from the filter plant back to the mineral processing plant, from a water balance, then $\lambda_{MD} > 0$. In particular, the corresponding water flow, assumed as the remainder of the water that is left in the concentrate, is given by $\frac{YG_F}{\rho_W}\left(\frac{1}{C_{CTU}} - \frac{1}{1-H}\right)$. Hence,

$$\lambda_{MD} = 1 - \frac{YG_F}{\rho_W Q_{WM}}\left(\frac{1}{C_{CTU}} - \frac{1}{1-H}\right), \tag{21}$$

with λ_{MC} corresponding to (minus) the second term on the right hand side of the latter expression, thus verifying (20). Using (21) and Q_{WM} (Eq. (5)), the part of the makeup that requires desalination is $Q_{MD} = \lambda_{MD}Q_{WM}$:

$$Q_{MD} = \frac{G_F}{\rho_W}\left[(1-Y)(1-\lambda_{TDR})\left(\frac{1}{C_{TTU}} - 1\right) + \frac{YH}{1-H}\right], \tag{22}$$

where it is seen that, as $Y \ll 1$ and $\frac{H}{1-H}$ is small compared to the unity, only a small part of the makeup is recovered from the plant through the concentrate pipeline stream.

Fig. 4 shows the relative importance of the various components of the total water recirculation flow as a function of the tailing thickener underflow concentration. Apart from the obvious fact that increasing the underflow concentration reduces the requirement of water recovery from the TSF, the figure shows that increasing the overall water recirculation within the plant for a fixed thickening rate, requires increasing water reclaim recoveries from the TSF, a process that becomes increasingly expensive and challenging at high recovery rates.

3.3. Specific energy components

The expressions (16) and (19) show that the cost of water depends critically on the energy required to handle it in the corresponding pipeline systems.

The values of $\widehat{E}_{WM}$, $\widehat{E}_{TDR}$ and $\widehat{E}_{CTO}$ depend on a number of design conditions, including the value of the makeup flow, the pumping efficiency, the pipeline diameter, and the transport distance. Pipeline aging is an important element because it conditions the internal wall roughness, which controls pressure losses at high Reynolds numbers. To obtain figures in the present analysis, an approach based on imposing a specific value of mean velocity in the pipeline given a particular nominal volume flow condition, pipeline roughness and pumping efficiency has been considered, using as a reference commercial steel pipelines (API 5L Sch. 80 or the thickest wall available depending on the size) in the final diameter choice (e.g. PM, 2017). It is noted that the required amount of material can range from about 65 kg of steel per meter pipe to more than 100 kg/m in large diameter pressure pipes.

An implication of the use of commercial steel pipelines is that the diameter choice possibilities are discrete. Additionally, following a closest commercial diameter approach as a design criterion causes to adopt a constant diameter within an operational range. An advantage of this approach, as opposed to continuously computing the pipeline diameter for specific process conditions, is that it allows to expose the effect of process variations for a constant local diameter and, at the same time, switch to significantly differing conditions *via* a different pipeline diameter. A continuous variation of the diameter would make this impossible, as this parameter would couple with variation in process conditions. The transitions between pipeline diameters are readily identified as discontinuities in the corresponding specific energy curves. The contrast between continuous and discrete computations is shown in Fig. 5.

A key requirement to compute the specific energy consumption is to identify a method for the calculation of the pipeline pressure losses. Consider that the water delivery point is located at a height $\Delta z > 0$ above the head pumping station. Assuming that the water discharge occurs at atmospheric pressure, an energy balance between both points yields the relation:

$$\Delta h = \Delta z + \frac{fL}{D}\frac{\bar{v}^2}{2g}, \tag{23}$$

where L is the pipeline length, D is the internal diameter, $\bar{v}$ is the mean flow velocity ($\bar{v} = 4Q/\pi D^2$, with Q the applicable volume flow), and g is the acceleration due to gravity. Here, f is the Darcy friction factor, which in the case of water can be computed using the well known Colebrook-White equation or an equivalent

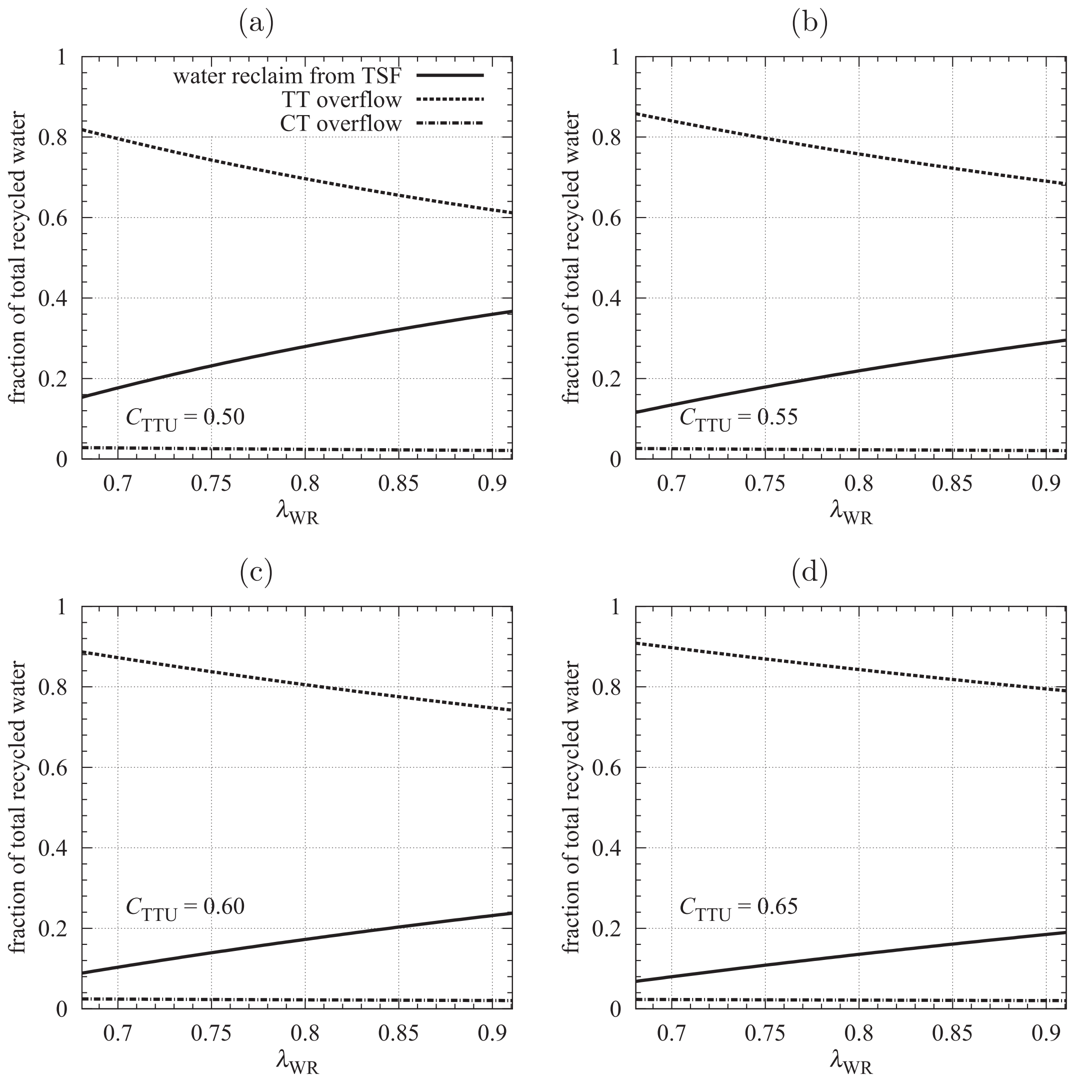

Fig. 4. Relative fractions of the total water recirculation, corresponding to water reclaimed from the TSF, water recycled from the concentrate and tailing thickeners (acronyms CT and TT, respectively), for (a) $C_{TTU} = 50\%$, (b) $C_{TTU} = 55\%$, (c) $C_{TTU} = 60\%$ and (d) $C_{TTU} = 65\%$. Here, λ_{WR} is the fraction of the total flow which is recirculated in the plant.

expression (Granger, 1987). The pumping power for the corresponding stream is computed as $P = \rho_W g Q \Delta h/\varepsilon$, with ε the global efficiency of the pumping system, and the associated specific energy requirement is expressed as $P/\rho_W Q$. Thus, for the water makeup and reclamation streams $\widehat{E}_{WM}$, $\widehat{E}_{TDR}$ and $\widehat{E}_{CTO}$, the following expression holds:

$$\widehat{E} = \frac{1}{\varepsilon}\left(g\Delta z + \frac{8}{\pi^2}\frac{fL}{D^5}Q^2\right), \tag{24}$$

where Δz, f, L, D and Q should be used at the water makeup, reclamation from the TSF and concentrate thickener overflow pipelines, considering for notational purposes the appropriate subscripts.

From (24), it is noted that the specific energy consumption increases with the square of the water flow and decreases with the fifth power of the pipeline internal diameter, and thus both can be applied at design or operational level to seek for optimal infrastructure and operational points. However, a limitation of increasing the internal diameter to decreasing the energy consumption is given by the cost of the pipeline, which is proportional to the required amount of steel per unit length. This is a growing function of both the square of the internal diameter and the pipeline thickness (Ihle et al., 2013b). An exact result would therefore be the outcome of an economical analysis, which is out of the scope of the present paper. However, it is commonplace that the resulting mean velocities are close to or slightly above 1 m/s (e.g. Saskatchewan Environment, 2004). In the present calculations, mean velocity values on this order have been used. To this purpose, a commercial pipe diameter has been set accordingly. On the other hand, a set of distances and height differences between reclamation and delivery points reflecting the case of ore concentrate plants in Chile will be assumed to give a sense of reality to present calculations. A detail of the corresponding values is given in Appendix A.

3.3.1. Desalination energy requirements

The parameter $\widehat{E}_{MD}$ depends on the efficiency of the desalination process. It is accepted that a relatively efficient reverse osmosis plant spends about 3 kWh/m^3 of water (Chong et al., 2015), but values can range from 2 kWh/m^3 to 7 kWh/m^3 (Lattemann and Höpner, 2008, and references therein). In a recent report, a value of 3 kWh/m^3 for desalination in Chile has been assumed (Castillo et al., 2015). For simplicity, the latter value is considered throughout the present calculations.

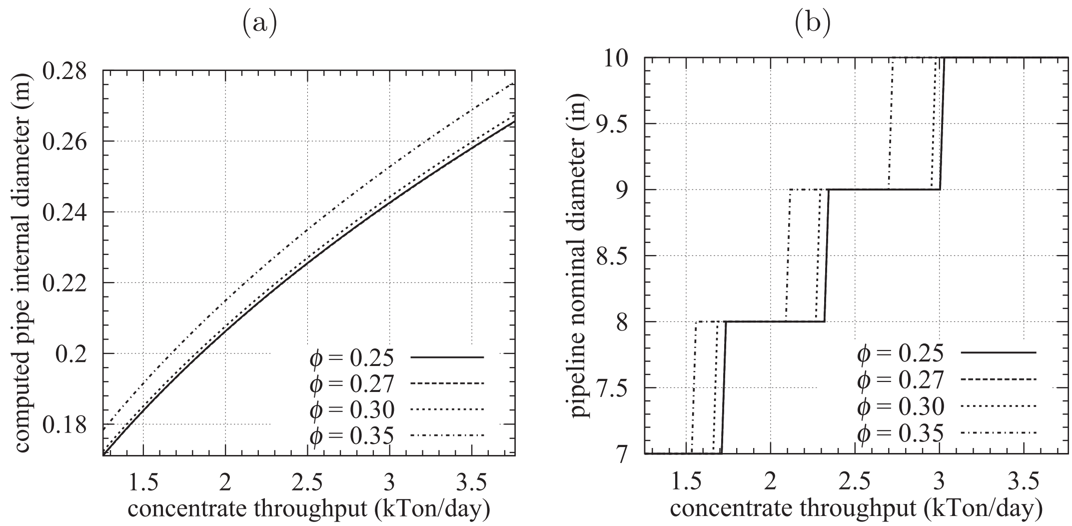

Fig. 5. Pipeline diameter determination for ore concentrate using (a) a continuous approach and (b) the choice of the closest commercial available value, for different slurry volume fractions. In the present nomenclature, the *x*-axis corresponds to the concentrate throughput, equal to YG_F. Present values of the volume solids fraction of the concentrate are similar to those in real concentrate pipelines (Betinol and Jaime, 2004; Ihle and Tamburrino, 2012). Ore concentrate reference throughput values are on the order of those reported for large scale Chilean mining operations (Consejo Minero, 2002).

3.3.2. Specific energy for water recirculation from ore concentrate thickeners, $\widehat{E}_{CTO}$

The underflow concentration in copper concentrate thickeners is mostly between 65% and 70% by weight. From a plant-wide mass balance perspective, it affects a very small fraction of the total mass passed through the plant (it is recalled here that the ratio of concentrate to feed mineral by mass is *Y*). The effect of concentration with plant feed throughput is computed using Eq. (24), and considering the concentrate thickener overflow $Q_{CTO} = Q_F - Q_{CTU}$. From (8) and (10), it follows that:

$$Q_{CTO} = \frac{YG_F}{\rho_W}\left(\frac{1}{C_F} - \frac{1}{C_{CTU}}\right). \tag{25}$$

The result is shown in Table 2, and suggests that comparatively smaller throughputs are related to higher specific energy values. Increasing the plant throughput requires to increase the pipeline diameter (at relatively constant design mean flow velocity). Replacing Q by $\pi D^2\overline{v}/4$ in (24) implies that the specific energy consumption depends on $1/D$, a trend that becomes weaker when increasing the volume flow requirement. A relevant outcome of this result is that small and medium-sized operations might benefit from merging their makeup streams into a single pipeline, a thing that has been recently explored an implemented in the Chilean context, including CVWS (Table 1) and the agreement for shared water supply between Teck Resources Ltd and Collahuasi (Reuters, 2016). The potential to reduce specific energy consumption at this point in larger-scale pipelines naturally offers opportunities for water supply for local communities and agriculture. An example is the case of Cerro Verde copper operation (CV), near Arequipa, Peru, which reached to an agreement with the local community where CV would, on one hand, participate on the construction and operation of a wastewater plant and receive, in exchange, a guaranteed volume of the treated wastewater for the plant (Fraser, 2017). In this context, water distribution networks can be optimized together with the location desalination plants (Herrera et al., 2015).

Table 2
Estimation of specific energy consumption of overflow water recovery from concentrate thickeners. Here, $(Y, C_F, C_{CTU}) = (2.5\%, 30\%, 65\%)$ has been considered. The rest of the computational assumptions are given in Table B.6 (Appendix B).

G_F (kton/day)	$\widehat{E}_{CTO}$(J/kg$_{water}$)
50	600.3
150	466.3
250	319.6
350	322.8

3.3.3. Specific energy for water recirculation from tailing thickeners, $\widehat{E}_{TT}$

The energy estimation for water reclamation from the overflow of thickeners has the same calculation method explained above. However, tailing thickeners may deliver slurries that can be either easily transported to the TSF by gravity using open channels or, at higher concentrations, may require additional mechanical energy for their transport, using either centrifugal or positive displacement pumps. At concentrations exceeding $C_{TTU} \approx 60\%$, they commonly have a strong non-Newtonian behavior, with more energy requirements and differing infrastructure from conventional tailings (Concha, 2014). Therefore, the estimation of the specific energy for water reclamation for thickeners will critically depend on the underflow solids concentration or, more specifically, the rheology of the tailings that are to be produced. Thus, the specific energy consumption related to water reclamation from tailing thickeners has an additional component related to tailing transport. This value is highly variable, as it strongly depends on the specific value of the thickener underflow concentration and, given a fixed value of concentration, on the fine contents, including the potential for clays, and the physicochemical characteristics of the slurry. In this regard, there are numerous examples of influential variables on flow properties, including zeta potential (Zhou et al., 2001), origin for a similar process tailing (the case of red mud is exposed in Sofrá and Boger, 2002), and presence of clays (Ndlovu et al., 2014; Zhang et al., 2015). Also, solid-liquid separation process itself is also affected by these elements (Addai-Mensah et al., 2007). Given such a broad range of parameters that can add variability to the results, a reported reference value for $\widehat{E}_{TTO}$, will be assumed. Here, as the corresponding energy consumption is rather interpreted as a water reclamation cost, it is convenient to convert it to energy per water unit weight. If $\tilde{E}$ is the specific energy per solid unit mass, then $\widehat{E} = \tilde{E}/\rho_W Q_{TTU}$ is the specific energy per water unit mass. Assuming that total specific energy consumption corresponds to the sum of the contributions of the thickener underflow and the overflow pumping, then:

$$\widehat{E}_{TT} = \frac{C_{TTU}\tilde{E}_{TTU}}{1 - C_{TTU}} + \widehat{E}_{TTO}, \tag{26}$$

where the first term on the right hand side corresponds to the energy requirement due to the tailing disposal using (either centrifugal or positive displacement) pumps, and the second corresponds to the energy consumption due to water transport from the overflow of the thickeners to the recirculation delivery point in plant. In the particular case of gravitational transport, $\tilde{E}_{TTU} = 0$ and $\widehat{E}_{TT} = \widehat{E}_{TTO}$, as expected. A reference value for $\tilde{E}_{TTU} = 0.7 \text{J/kg}_{\text{solids}}$ has been estimated from data related to large scale copper district in North Chile (Rayo et al., 2009), for the TSF pipeline length considered herein (Appendix B), within the so-called economic range for using this technology (Nguyen and Boger, 1998, and references therein). In conventional tailing TSF's, it is common to obtain the dam fill material from the coarse fraction of the tailing (Blight, 2010). This has not been considered in the present analysis, as the costs involved are strongly dependent on the topography, and thus the site closure conditions. An implication of this hypothesis is that, at equal water-to-energy costs for concentrate transport, it is better to use thickened tailings rather than conventional ones.

The specific energy consumption of tailing thickeners depends strongly on the concentration range at the underflow. Whereas high rate thickeners report much more water at the overflow than conventional units —similarly as in Eq. (25), the water recovery from the thickeners is proportional to $\frac{1}{C_F} - \frac{1}{C_{TTU}}$ —, the existence of paste flows impose the need for tailing pumping to the TSF. Fig. 6 shows the impact of the underflow concentration on the specific energy required per kg water, in terms of the thickener underflow concentration and various feed plant throughputs. The steep change on the specific energy at $C_{TTU} = 0.54$ corresponds to the imposition of slurry pumping above that value. Although this definition is arbitrary, it gives some insight on the energetic impact of producing thickened or paste tailings. In the present example, the specific energy related to recycling water using conventional or thickened/paste tailings from the thickener nearly triplicates. On the other hand, it is noted that for a constant pipeline diameter, the specific energy tends to increase with concentration. This is explained by the fact that increasing the underflow concentration cause an increase on the water overflow at the thickener. From (27) (Section 3.3.4), this roughly implies an increment of the specific energy with Q^3_{TTO}. While in the lowest throughput value (50kton/day) the hydraulic conditions for pumping required several different diameters (suggesting giving a slight tendency to decrease), in the higher throughput cases only two pipe diameters appear. However, as mentioned, this decreasing tendency of the specific energy can be somewhat misleading because occurs at different pipeline diameters, a situation that is no possible in operating plants.

3.3.4. Specific energy related to water makeup

The specific energy associated to water makeup depends critically on both the water makeup flow (Eq. (13)), and the calculation hypotheses indicated on Section 3.3. The effect of the thickener underflow concentration is exemplified on Fig. 7. For constant diameters of the makeup line there is a tendency to reduce the specific energy consumption with the C_{TTU}. This should not be confused with the direct consequence of reducing the makeup by increasing the recycled water from the plant (from the overflow of the tailing thickeners). The result shows, indeed, that the water transport becomes least expensive per unit weight of makeup water given a single pipeline diameter. While this is true varying the solids concentration *ceteris paribus*, it is also noticeable for various different pipeline lengths. Fig. 7 shows that for fixed pipeline diameters, reductions on the specific energy consumption are significantly lower for 200 km pipelines than for 50 km pipelines, whereas the corresponding constant-diameter slope of curves are roughly constant for given pipeline lengths.

Fig. 8 shows the effect of throughput on the specific energy consumption linked to makeup. For fixed pipeline diameters, the energy consumption tends to increase with throughput, as previously concluded in similar calculations (Nguyen and Boger, 1998). On the other hand, higher throughputs imply higher pipeline diameters because the required slurry flow is higher. Computations show that progressively increasing the pipeline diameter causes a

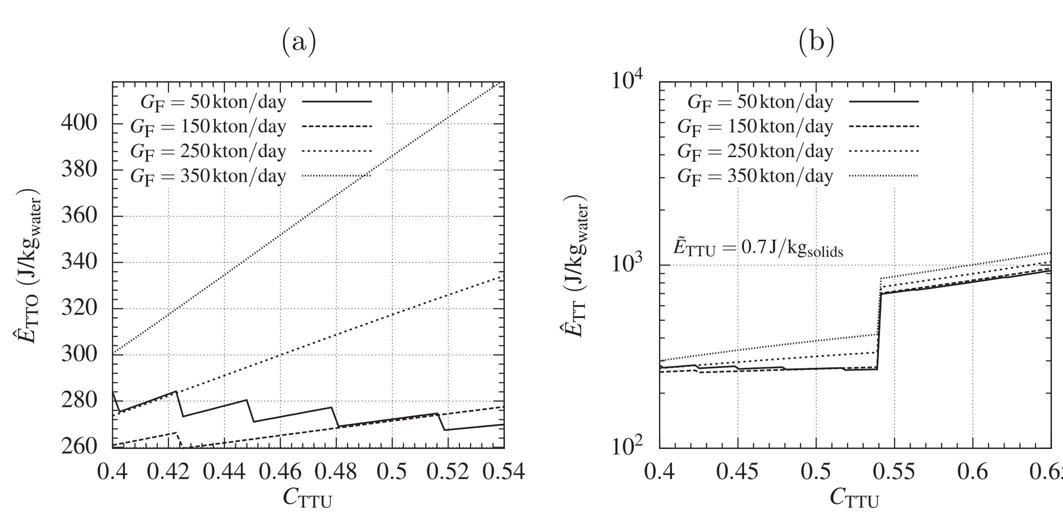

Fig. 6. Specific energy consumption for water recovery from the tailing thickeners (Eq. (26)). The sawtooth-like form of the curves are explained by the multiple diameters assumed for the sake of the analysis. (a) Specific energy corresponding to water recovery from thickener overflow. (b) Specific energy including pumping energy of underflow for thickened and paste. The discontinuity at $C_{TTU} = 0.54$ corresponds to the transition between gravitational underflow transport and pumping. The specific energy for solids transport considered is given by $\tilde{E}$.

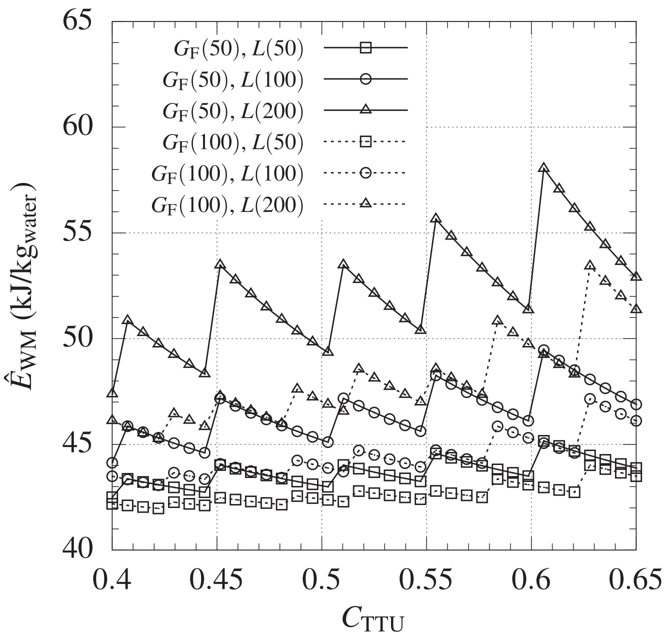

Fig. 7. Specific energy component related to water makeup ($\widehat{E}_{WM}$), in terms of the thickener underflow concentration, C_{TTU}, with $Y = 0.025$. In the legend, values in brackets correspond to thousand tonnes per day for G_F and pipeline length in kilometers in the case of the variable L.

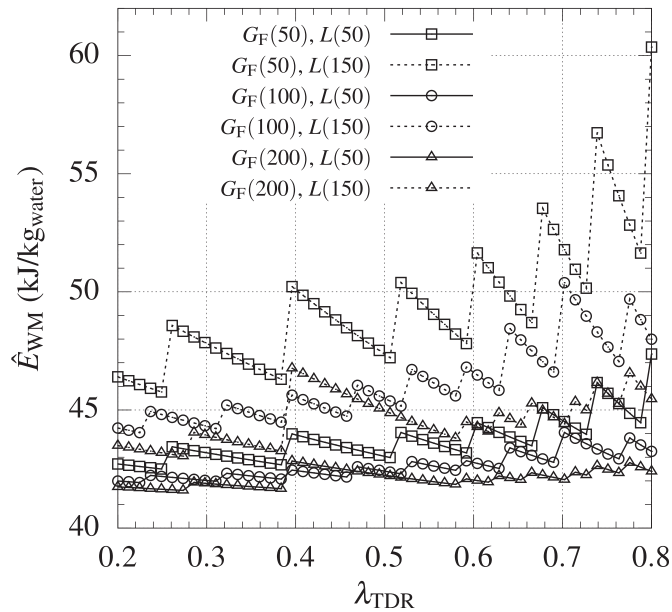

Fig. 9. Specific energy component related to water makeup ($\widehat{E}_{WM}$), in terms of the fraction of water recovery from the TSF, λ_{TDR}, with $Y = 0.025$. In the legend, values in brackets correspond to thousand tonnes per day for G_F and pipeline length in kilometers in the case of the variable L.

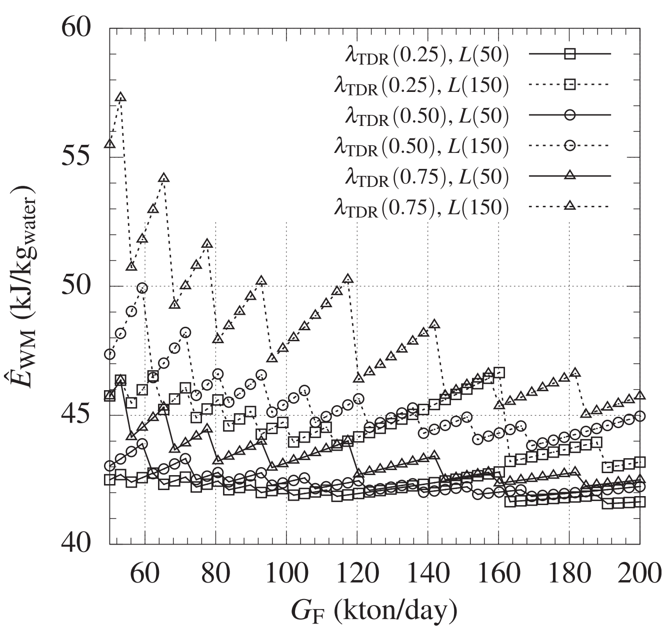

Fig. 8. Specific energy component related to water makeup ($\widehat{E}_{WM}$), in terms of the plant feed throughput, G_F, with $Y = 0.025$. In the legend, values in brackets correspond recovery fraction from the TSF (λ_{TDR}) and pipeline length in kilometers in the case of the variable L.

progressive decrease on the energy consumption for makeup. Fig. 8 also shows that the slope of each constant diameter segment decreases increasing the throughput. This reveals that at higher tonnages, the specific energy consumption becomes less sensitive to local changes on the throughput.

Fig. 9 shows the effect of the fraction of water recovered from the TSF, λ_{TDR}, on the specific energy consumption for makeup. The resulting set of curves shows a tendency to increase the value of $\widehat{E}_{WM}$ with λ_{TDR}. However, given a constant makeup pipeline diameter, increasing λ_{TDR} would cause steeper decreases on $\widehat{E}_{WM}$. That means that in existing systems, improving water recovery will decrease the makeup cost per m^3 of water. The corresponding (negative) slopes become steeper increasing the pipeline length, as inferred from (24). The slope of each section, on the other hand, tends to decrease with pipeline length.

3.3.5. Specific energy requirement for dispatched ore concentrate, $\tilde{E}_{OCP}$

Concentrate is dispatched using a long-distance pipeline in a natural topography. At the end of the transport system a dissipation station is installed to provide for the necessary tilting of the energy line to avoid that the energy line crosses the topography and thus pressures going near the vapor value. The specific energy requirement is computed considering a high point located at position $x_{HP} = \lambda_{HP} L$, with $0 \le \lambda_{HP} \le 1$ (Fig. 10a). The existence of such highpoint is most common in Chilean concentrate pipelines, given that both in the north and central regions in the country there are two major mountain formations, namely the Andes Mountains and the Coastal Mountains.

From the energy standpoint, the most energy-efficient option is to set the transport so that the energy line is as close as possible to the high point (Ihle, 2016). Assuming the limiting condition that the energy line is exactly touching the high point, then, an energy balance reveals that the specific energy consumption (measured per unit solids) $\tilde{E}_{OCP} = P/YG_F$, is given by:

$$\tilde{E}_{OCP} = \frac{\rho_{sl} Q_{sl}}{Y G_F \varepsilon_{sl}} \left[g(z_{HP} - z_{PS})_{sl} + \frac{8}{\pi^2} \frac{\lambda_{HP} f_{sl} L_{sl} Q_{sl}^2}{D_{sl}^5} \right], \tag{27}$$

where $\rho_{sl} = \rho_{s,i}\phi + \rho_W(1-\phi)$ is the slurry density, and Q_{sl} is the slurry flow. Here, $\rho_{s,i}$ is the solid phase density, where i can be either T (tailings) or C (concentrate) and the subscript W denotes water. On the other hand, z_{HP} and z_{PS} are the elevation of the high point and the pump station, respectively. The variables f_{sl}, L_{sl}, D_{sl} and ε_{sl} follow the definitions of the variables in (24), albeit this time

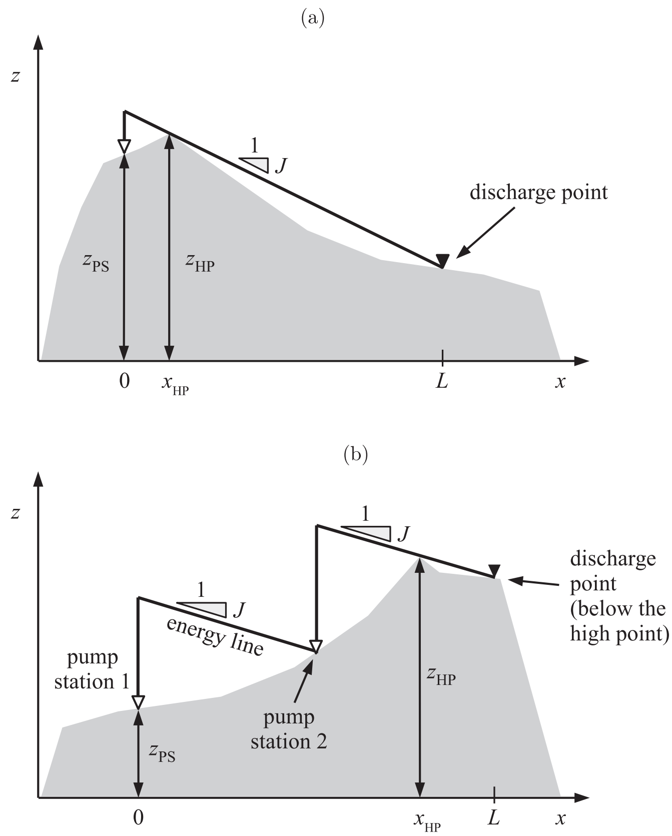

Fig. 10. Schematic of the topography and energy components considered in the specific energy calculation for single diameter pipeline with the energy line with virtually zero relative pressure at the high point, located at $x_{HP} = \lambda_{HP}L$. (a) Ore concentrate pipeline, (b) Water pipeline. In (b), the facility has been shown with two pump stations. This has no influence on the specific energy analysis result during steady state when all the pipeline segments have constant diameter.

related to the slurry transport system, as denoted by the subscript *sl*. It is noted that (27) can be further reduced and expressed in terms of the plant throughput noting that, by solid mass conservation,

$$YG_F = \rho_s \phi Q_{sl}, \tag{28}$$

where ρ_s is the solids density. Thus,

$$\tilde{E}_{OCP} = \frac{1}{\phi \varepsilon_{sl}} \left[g(z_{HP} - z_{PS})_{sl} + \frac{8}{\pi^2} \frac{\lambda_{HP} f_{sl} L_{sl} Q_{sl}^2}{D_{sl}^5} \right]. \tag{29}$$

It is noted that the presence of a high point in the pipeline often requires the definition of some degree of point energy dissipation in the line, which is commonly achieved using a dissipation station. Alternatively, an energy generation station can be used. In either case, it is required that the energy line must be above the topography and as close as possible to the highest point (Ihle, 2016).

If the stream of interest is water and the delivery point is above the pump station, (24) can be expressed, using the high point location instead of the delivery point, as:

$$\widehat{E}_W = \frac{1}{\varepsilon_W} \left[g(z_{HP} - z_{PS})_W + \frac{8}{\pi^2} \frac{\lambda_{HP} f_W L_W Q_W^2}{D_W^5} \right], \tag{30}$$

Fig. 10b shows a schematic with the axes definition used for this case, with two pumps stations (the number of pumps connected in

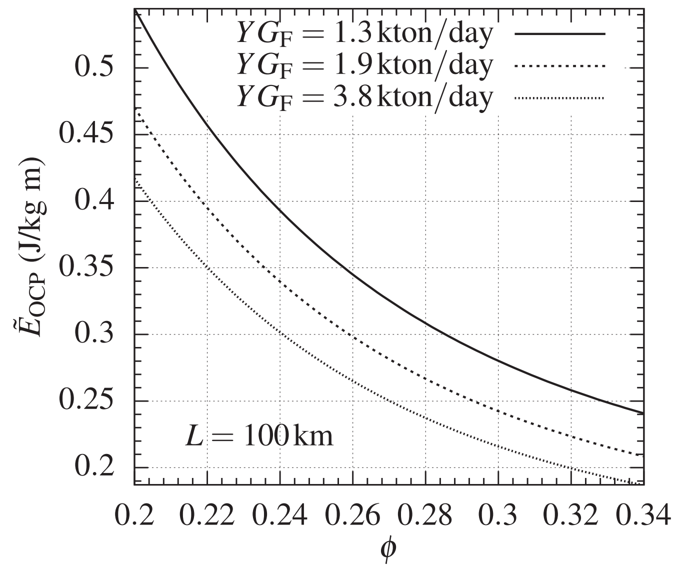

Fig. 11. Specific energy consumption in the ore concentrate pipeline, $\tilde{E}_{OCP}$, as a function of the solids volume fraction.

series is irrelevant in steady state and if pipeline has a constant internal diameter). The details of the hydraulic design assumptions for the computation of (27) in the present case, are given in Appendix A.

Fig. 11 shows the resulting specific energy obtained from computation algorithm utilized herein. It is noted that, as a consequence of the relatively stronger impact of the volume flow in comparison with the slurry concentration on the specific energy (Eq. (27)), the system requires lower transport energies per unit solids (everything else kept constant), for lower flow rates, (*i.e.* for higher concentrations) the specific energies are lower. On the other hand, Fig. 11 shows that increasing the throughput causes an increase on $\tilde{E}_{OCP}$, thus causing that, at equal solid transport concentrations and a single pipeline used for the transport, large scale plants have higher energy requirements that small scale ones. This unavoidable effect is a consequence of both the nonlinear dependence of the consumed hydraulic power with the slurry flow (Eq. (27)), and the linear dependence of the concentrate solid flow with the slurry flow, Q_{sl}, as seen in (28). In this Figure, the value YG_F is kept constant, but fluctuations on feed ore grade cause fluctuations on the concentrate throughput, and consequently on the tailing flow. For instance, given a reference value for grades of fg = 0.8%, tg = 0.1% and cg = 28%, a 10% fluctuation of the ore feed grade would propagate to fluctuations of 11% in *Y*.

3.4. Effect on costs

The relations (16) and (19) are proportional to the unit cost of energy, θ_E. This means, in light of (1), that under the present set of hypotheses, the cost function for ore concentrate transport may be expressed as to be proportional to the unit cost of energy, thus reducing in one degree of freedom the problem. This approach effectively implies a difference with the slightly more general approach adopted in Ihle (2013) and Ihle et al. (2013a), where θ_E and θ_W are treated as independent variables. It is thus convenient to express the corresponding cost results normalised by the unit cost of energy. In particular, the normalised transport cost Ω/θ_E, has units of power (energy per unit time). On the other hand, the ratio of water to energy costs has units of energy per volume.

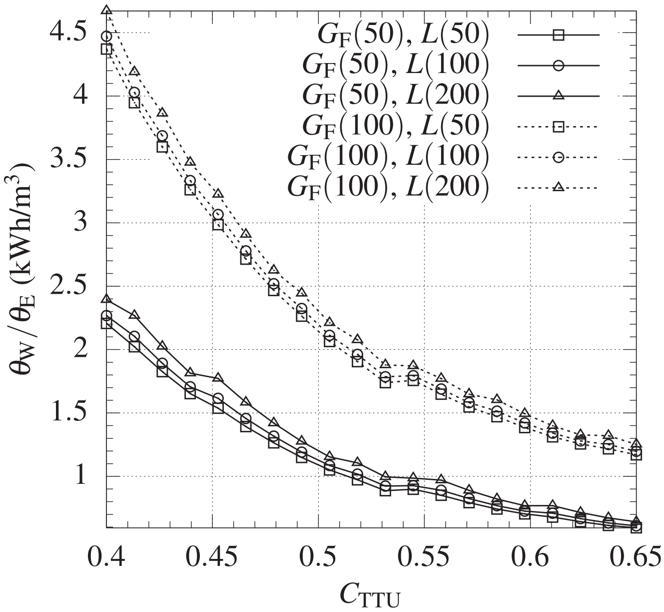

Fig. 12. Ratio between water and energy unit costs in terms of the thickener underflow concentration. In the legend, values in brackets correspond to thousand tonnes per day for G_F and pipeline length in kilometers in the case of the variable L.

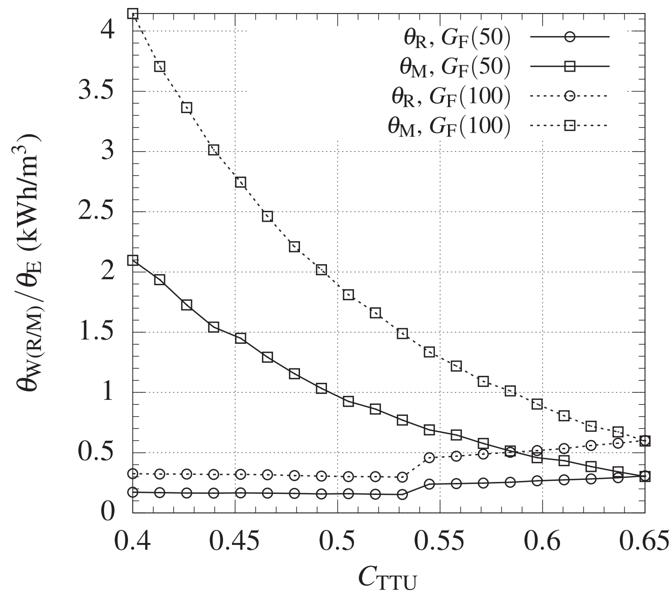

Fig. 13. Components of the ratio between water and energy unit costs in terms of the thickener underflow concentration, C_{TTU}, for fixed values of the pipeline length and the fraction of water recovered from the TSF of $L = 100$ km and $\lambda_{TDR} = 0.5$, respectively. In the legend, values in brackets correspond to thousand tonnes per day for G_F.

[Fig. 12](#) shows the ratio between water and energy unit costs in terms of the thickener underflow concentration, C_{TTU}, where a strong tendency to reduce the normalised unit cost of water is observed with an increase of water recovery from tailing thickeners is observed. Present computations show that such drop on the water cost has a greater slope for underflow concentrations below 0.54. This corresponds to the imposed transition between pumping requirement (above 0.54) and gravitational flow to the underflow. An important result is that, from the operational cost standpoint, water costs tend to drop when increasing recovery from thickeners, in spite of the additional energy requirement related to thickened or paste tailing transport. It is also shown that, while the length of the makeup line has a slight effect on results, the throughput has a major influence on the normalized water costs, with higher unit costs associated to higher throughputs.

Fig. 13 shows separately the recirculation and makeup components, in terms of the underflow thickener concentration, C_{TTU}. Increasing the recirculation *via* the value of C_{TTU}, causes an increase of the value of the recirculation per unit weight of water, whereas the opposite occurs with the makeup, with monotonically decreasing curves. Trends are similar when the components of the water unit cost are plotted against the fraction recovered from the TSF, λ_{TDR} (Fig. 14). This is explained by the fact that both water input streams are in parallel.

The water cost, θ_W, is proportional to θ_E, and consists of the contributions of water recirculation and makeup, as shown on the relations (14), (16) and (19). Fig. 15 shows the ratio of the recirculation to the makeup component of the water cost. The monotonically increasing characteristic of this curve is the direct consequence of the trade-off between makeup and recirculation. The non-linear increase of the recirculation component compared to the makeup above $C_{TTU} = 0.54$ is, again, the consequence of the tailing transport cost downstream the underflow of the thickener. Here, the rheology of high density slurry plays a substantial role on the energy consumption at high concentrations (this relevance is reproduced herein in a simple way using a Krieger-type model, as detailed in Appendix A). Fig. 15 also shows that at high thickener underflow concentrations, some combinations of throughput and

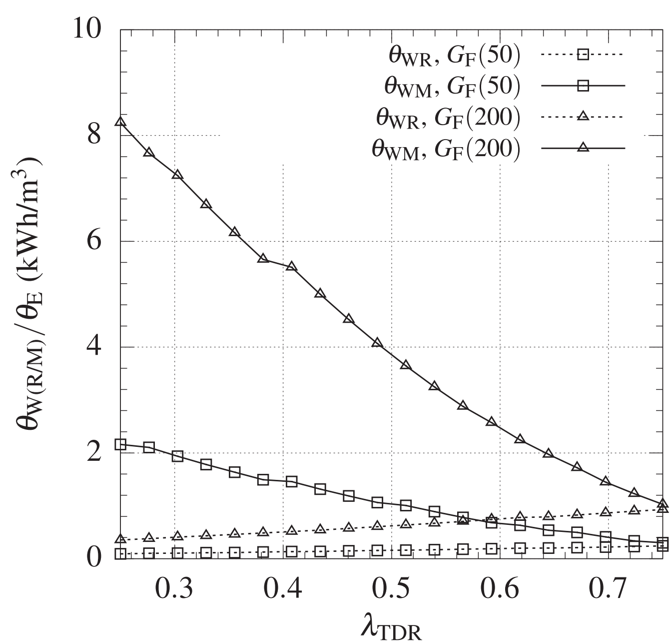

Fig. 14. Components of the ratio between water and energy unit costs in terms of the fraction of water recovered from the TSF, λ_{TDR}, for fixed values of the pipeline length and the thickener underflow concentration of $L = 100$ km and $C_{TTU} = 0.55$, respectively. In the legend, values in brackets correspond to thousand tonnes per day for G_F.

pipeline length may outstrip the unit cost of makeup. This is controlled by (24), where it is revealed that shorter pipelines reduce the cost, but higher throughputs and the nonlinearity of the rheology with the concentration tends to increase it.

An interesting implication of the curves given by Fig. 15 is that both the threshold where the tailing needs to be pumped (here set at $C_{TTU} = 0.54$) and the growth rate of the ratio θ_{WR}/θ_{WM} will depend critically on the value of $\tilde{E}$ (kept fixed herein), which depends on the microstructure of the suspension, including the

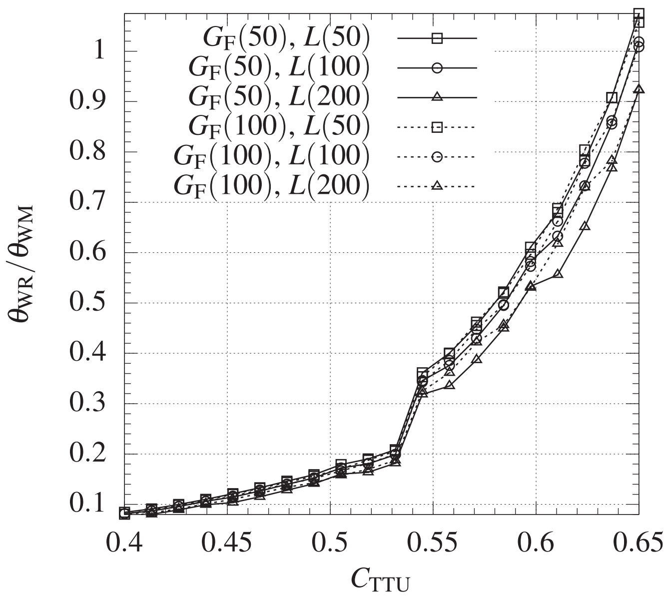

Fig. 15. Ratio between recirculation and makeup components of the water unit cost, as a function of the thickener underflow concentration. In the legend, values in brackets correspond to thousand tonnes per day for G_F and pipeline length in kilometers in the case of the variable L.

rheology of the solid-liquid mixture and the consolidation characteristics. While it is desirable to decrease the energy requirements during the transport phase (*i.e.* the mean slope on the rightmost segment of Fig. 15), and thus parameters such as the viscosity of the tailing, thickened and paste tailings are disposed at beach slopes between 1% and 4% (Simms, 2017), where disposal is at the expense of rheological/geotechnical properties such as the yield stress. In particular, if the yield stress is high enough, centrifugal pumps may perform poorly (Wilson et al., 2006) and be replaced by positive displacement pumps, whose capital costs are significantly higher. On the other hand, rheological attributes such as time-dependency of the tailing may play a substantial role on their behavior during the transport-disposal phase (Nguyen and Boger, 1985). Thus, there is a major challenge to produce thickened tailings that can be treated in a suitable manner (both from a hydrodynamic and chemical point of view) to optimize the transport-disposal process, seen as a whole. Although some attempts have been made using thickener flocculants downstream spigot discharges at the TSF (Jewell, 2012), there is not a widely accepted strategy to tackle this problem.

It is noted that the notion of cost acts as metric of importance, which can be adapted to other factors than those strictly economic. For instance, the shared water routing example worked by Gao et al. (2016) in a coal district might also be interpreted in terms of cost of pollution by unregulated discharges in front of extreme climatic events, thus looking for optimal water storage and transport values globally and for each plant in the district.

4. Conclusions and final remarks

4.1. Conclusions

In the present work, it has been shown that both the costs of water recirculation and makeup can be included in a single water cost function, associated to the product (ore concentrate). If seawater is considered as a water source, then the cost of water actually becomes the cost of energy. A metric for the water cost for concentrate transport in concentrate processing plants with a remote water makeup source location has been proposed. From an operational perspective, the effect of having a remote makeup water source location has significant implications on the supply costs of water, both from a bulk perspective and from the point of view of unit costs. In particular, increasing water recovery both from thickeners and from the TSF in existing plants helps reducing the makeup specific energy cost.

Given the relatively common configuration of large scale Chilean mineral processing operations presented herein, computations suggest that using thickened tailings offers a cost effective alternative to conventional tailings, complementing previous works on the environmental benefits of this approach. Although in light of present results very high thickening may be costly due to underflow transport requirements, it seems this might be so only in a limited set of cases of very high tailing concentrations or considerable distances between the thickener underflow and the disposal sites (it is assumed herein a fixed thickener underflow pumping distance for thickened and paste tailings of 1000 m which, if reduced, would imply that even paste-like tailings would require specific energies virtually proportional to such tubelength).

For those operational plants that produce conventional tailings (below the limit where pumps are required for hydraulic transport), results reveal that the energy required for makeup is on the order of 80% (or even more for solids concentrations close to 50% in copper sulphide tailings) the energy required for the concentrate hydraulic transport. The main controlling factor in this water cost ratio (even over pipeline length) is thickener underflow concentration.

Water reclamation from the TSF, and thus the need to optimize disposal management, appears as a key point to put efforts for better resource utilization. However, as the water reclamation process itself is a secondary stage downstream the thickener underflow, the efficiency of this process is bounded by the inevitable water retention in the tailings, the rate of evaporation and the need to operate pumping equipment on the tailing disposal site. This, again, places the focus on the convenience of increasing thickener downstream concentrations to enhance water recovery in the plant, where even a single additional point of solids concentration may imply thousands (or even millions) of cubic meters of water over the lifespan of an operation. This observation holds under the present set of hypotheses including relatively small distances between the mineral processing plant and the TSF, and to have thickening infrastructure within the plant area. In particular, given a constant pipeline diameter, increasing the water recovery from the TSF or the concentration from the tailing thickeners decreases the specific energy consumption and the makeup component of the cost. From a mass balance perspective, variations on the water content in the concentrate stream are irrelevant on water savings.

4.2. Final remarks

The present work suggests the need for a broader integration between neighboring mining operations in the sense of water resourcing. Both from the capital cost and the operational cost perspectives, the use of mainstream pipelines to source more than one operation would tend to reduce the specific energy consumption for each of them. This naturally suggests the need for integrated project plan at district level, potentially requiring coordination and combined efforts of more than one mining companies.

Focusing separately on the control of the tailing transport and the disposal would help promoting the minimization of head losses in the hydraulic transport system and subsequently, maximizing the geotechnical stability at high slopes. This is a challenge that lies close to the boundary between process engineering, material science and geotechnical engineering. The development and adaption of multi-process additives, is also an aspect to look at, both from a cost perspective and from an environmental one. In particular, a reduced use of water in the system (and therefore less make up) implies a reduced potential for chemical destabilization of the TSF in the long term due to acid mine drainage of sulphide mineral tailings.

The present work has not assessed the impact of capital costs, which are strongly site-dependent. However, it is worth noting that conventional tailing infrastructure can compete with higher density one, especially when the water supply has to occur by means of a hundred kilometer long pipeline of several inches in diameter, implying dozens of kilograms per meter pipe, where every additional inch in diameter implies considerable investment differences that may play, both from a purely economic and from carbon footprint standpoint, in favor of increasing thickener underflow solid content as much as possible.

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List of symbols

cg	concentrate grade
C	solids concentration by weight (can be fractional or percent)
D	pipeline internal diameter
E	specific energy (energy per weight solid or water depending on the subscript)
fg	feed grade
g	gravity acceleration magnitude
G	plant throughput
H	humidity (humid base)
l_x	ore grade of species x
L	pipeline length
OCP	acronym for ore concentrate pipeline.
Q	volume flow
tg	tailing grade
TSF	acronym for tailings storage facility
Y	concentrate to feed weight ratio ($1-Y$ denote tailing)

Greek letters

ε	pump efficiency
ϕ	slurry volume fraction
λ	fraction
θ	cost component
ρ	density

Subscripts and superscripts

C	concentrate
CTO	concentrate thickener overflow
CTU	concentrate thickener underflow
EI	evaporation and infiltration
E	energy
F	feed
HP	high point
PS	pump station
ret	fraction of water retained in the TSF
s	solid phase
sl	slurry
T	tailing
TT	tailing thickener
TDR	water recovered from the TSF
TTO	tailing thickener overflow
TTU	tailing thickener underflow
W	water
WC	water content in concentrate stream
WM	makeup water
WR	TSF recirculation stream
$\hat{x}$	x, per kg solids
$\tilde{x}$	x, per kg water

Appendix A. Calculation hypotheses

Appendix A.1. Slurry pipeline (ore concentrate)

For simplicity, the present slurry has been assumed as a Bingham plastic (see Ihle, 2013, and references herein), which means that to the slurry viscosity, a yield stress representing the local solid to liquid transition is a required parameter.

The problem presented herein requires the choice of the pipeline diameter the mean flow velocity and the corresponding concentration, given the plant throughput. On the other hand, the diameter determination requires an additional criterion. Following the conclusions of Wu et al. (2010) and Ihle (2016), a least energy criterion has been adopted as a design guideline for slurry transport, which implies that the applicable mean flow velocity has to be as close as possible without the formation of dunes or any on the other sort of solids bed within the pipeline section. Assuming additionally that the flow regime must be turbulent, the minimum velocity is defined herein as the maximum value between the deposit velocity and the laminar-turbulent transition velocity. Once this parameter is set, given the mass balance in the slurry that relates the throughput with the flow (Eq. (28)), the pipeline diameter is computed for a given slurry concentration. The slurry flow iteration cycle has been made following the steps given in Algorithm 1.

Algorithm 1 Iterative algorithm for the computation of the pipeline diameter (D) and the mean flow velocity ($Q_{sl}/A = v_{min}$), where $A = \pi D_{sl}^2/4$ is the (circular) cross-sectional area of the pipeline. The Hedström number, used herein, is defined in (A.2)

Require: First guess for pipeline diameter, D_0, and an auxiliary variable D_{sl}
Ensure: Initialize $D_{sl} \neq D_0$ for first iteration

while $\left|1-\frac{D_{sl}}{D_0}\right| > \varepsilon$ $\{\varepsilon \ll 1\}$ **do**
  $D_{sl} \leftarrow D_0$
  Compute He (Hedström number)
  Compute Re_c (critical Reynolds number)
  Obtain laminar-turbulent transition velocity (v_T) from Re_c, D_{sl} and slurry properties
  Obtain deposition velocity (v_D) from D_{sl} and slurry properties {A safety factor may be applied to v_T and/or v_D}
  $v_{min} = \max(v_T, v_D)$
  $A \leftarrow \frac{Q_{sl}}{v_{min}}$ {Q_{sl} is known from (28)}
  $D_0 \leftarrow 2\sqrt{\frac{A}{\pi}}$
end while
$D_{sl} \leftarrow D_{com}$ {D_{com} is the closest commercial diameter}

The algorithm is dependent on the method followed for the computation of the critical Reynolds number and the deposition velocity. Here, we consider for the computation of the critical Reynolds number the model proposed by Slatter (2001) for large Hedström numbers, a common result in ore concentrate pipelines:

$$Re_c = 26\sqrt{He}, \tag{A.1}$$

where He is the Hedström number, defined as

$$He = \rho_{sl}\tau_y(D_{sl}/\eta)^2, \tag{A.2}$$

where τ_y is the slurry yield stress and η is the slurry viscosity. It is noted that in the reference above, this result is expressed with dimensions, in terms of the transitional velocity, proportional to $\sqrt{\tau_y/\rho_{sl}}$. To build the expression (A.1), the slurry viscosity has been determined using a Krieger-type model (Ovarlez et al., 2006):

$$\frac{\eta}{\mu} = \left(1 - \frac{\phi}{\phi_m}\right)^{-2}, \tag{A.3}$$

where μ is the viscosity of water and $\phi_m = 0.5$ is a loose packing concentration value suitable for the present application (see Ihle, 2013; for a broader discussion).

The yield stress has been computed following the empirical form (e.g. Heymann et al., 2002):

$$\tau_y = \frac{\alpha_y}{(\phi_m - \phi)^{\beta_y}}, \tag{A.4}$$

where α_y and β_y are empirical constants that give a reasonable order-of-magnitude estimation of τ_y for the present slurry of interest (on the orden of 1Pa, Betinol and Jaime, 2004). Their values are detailed in Table B.6, in Appendix B.

The deposition model has been assumed as in Ihle (2013), following Poloski et al. (2010), which has been calibrated for fine slurries, thus being appropriate for many ore concentrates and tailings. The corresponding values for the various parameters described herein are detailed in Table B.6.

The resulting flow and pipeline diameter is used in the specific energy model for ore concentrate transport, as expressed in Eq. (27). To finalize this calculation, the friction factor (f_{sl} in Eq. (27)) needs to be computed out the pipeline diameter, flow and slurry data. To this purpose we consider here the model proposed by Chilton and Stainsby (1998), based on the Blasius empirical model for the friction factor which, for a Bingham plastic (assumed herein) has the form $f_{sl} = 0.316\left(1 - \frac{\tau_w}{\tau_y}\right)Re^{-1/4}$, where τ_w is the wall stress. From the definition of the Darcy friction factor, $f_{sl} = \frac{8\tau_w}{\rho_{sl}(Q_{sl}/A)^2}$, this relation can be explicitly expressed in terms of the Reynolds and Hedström numbers as:

$$\frac{\tau_w}{\tau_y} = \frac{1}{2}\left(1 - \sqrt{1 - \frac{32He}{0.316Re^{7/4}}}\right). \tag{A.5}$$

On the other hand, the hydraulic calculation in Eq. (29) depends on the slurry efficiency, which becomes poorer with the solids fraction (Wilson et al., 2006). To account for this effect, a linear pump derating scheme (in a similar fashion to that proposed by Kazim et al., 1997) has been used:

$$\varepsilon_{sl} = \varepsilon_{sl0} - \frac{\varepsilon_{sl0} - \varepsilon_{sl1}}{\phi_{\varepsilon_{sl1}}}\phi. \tag{A.6}$$

The corresponding values of the parameters ε_{sl0}, ε_{sl1} and $\phi_{\varepsilon_{sl1}}$ are given in Table B.6, in Appendix B.

Appendix A.2. Water pipelines

Water pipelines, in contrast to slurry pipelines, don't have strong lower flow restrictions imposed by the physics. They, however, have economical restrictions as too small mean flow velocities are bonded to high diameters and thus expensive capital costs. In the present work, a water mean flow velocity of 1.5 m/s is assumed as a design criterion. Although there is not an economic analysis behind this decision and the assumed value is certainly arbitrary, the choice is within the common value range for this kind of infrastructure.

Given the aforementioned velocity criterion, once the diameter is estimated from the water volume flow as in lines 8 and 9 in Algorithm 1, the final diameter choice corresponds to the closest commercial diameter. The specific energy requirement is finally computed using Eq. (24).

Appendix B. Parameter values

Table B.6 shows the assumed values for the various parameters used throughout the present work.

Table B.6
Parameter values used in present computations, except those specified in the figures

Description	value	Calculation involved
Specific energy for water desalination ($\hat{E}_{MD}$)	3 kWh/m^3	Eq. (19)
Water reclamation pipeline (TSF-plant, L)	3000 m	Eq. (24)
Thickener overflow water pipeline length (L)	1000 m	Eq. (24)
Concentrate overflow water pipeline length (L)	1000 m	Eq. (24)
Tailing discharge pipeline (plant-TSF, L)	2000m	Eq. (26)
Specific energy for underflow thickened tailing transport ($\tilde{E}_{TTU}$)	0.7 J/kg^{solids}	Eq. (26)
Slurry pump efficiency at $\phi = 0$ (ε_{sl0})	0.7	Eq. (29)
Slurry pump efficiency reference volume fraction ($\phi_{\varepsilon_{sl1}}$)	0.4	Eq. (29)
Slurry pump efficiency at $\phi = \phi_{\varepsilon_{sl1}}$ (ε_{sl1})	0.4	Eq. (29)
Tailings solids density in ρ_{sl} ($\rho_{s,T}$)	2700 kg/m^3	Eq. (29)
Concentrate solids density in ρ_{sl} ($\rho_{s,C}$)	4200 kg/m^3	Eq. (29)
Position of high point in concentrate stream (λ_{HP})	0	Eq. (29)
Elevation difference between high point and pump station in concentrate stream ($z_{HP} - z_{PS}$)	2500 m	Eq. (29)
Water pump efficiency (ε_W)	0.7	Eq. (30)
High point location in water recovery stream from TSF (λ_{HP})	1	Eq. (30)
Elev. diff. between high point and pump station in water recovery from TSF stream ($z_{HP} - z_{PS}$)	100 m	Eq. (29)
High point location in water recovery stream from thickeners (λ_{HP})	1	Eq. (30)
Elev. diff. between high point and pump station in water recovery from TSF stream ($z_{HP} - z_{PS}$)	15 m	Eq. (30)
Packing concentration assumed in viscosity and yield stress models (ϕ_m)	0.5	Eqs. (A.3)/(A.4)
Yield stress empirical parameters (α_y, β_y)	(0.013Pa, 1.82)	Eq. (A.4)

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