

**INVESTIGATION ON TRANSPORTATION CHARACTERISTICS OF
PULVERIZED COAL SLURRY**

*A thesis submitted in the fulfilment of the requirements for
the award of the degree of*

DOCTOR OF PHILOSOPHY

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Thesis certification

I, Harmanpreet Singh, hereby state that the research presented in this thesis entitled “Investigation on Transportation Characteristics of Pulverized Coal Slurry”, submitted in the fulfilment of the requirements for the award of the degree of **Doctor of Philosophy** in the Mechanical Engineering Department, Thapar Institute of Engineering and Technology, Patiala, Punjab, India, is an authentic record of my own work carried out under the supervision of Prof. S.K. Mohapatra, Senior Professor, Mechanical Engineering Department, Thapar Institute of Engineering and Technology, Patiala and Dr. Satish Kumar, Associate Professor, Mechanical Engineering Department, National Institute of Technology Jamshedpur from August 2016 to December 2020. The contents of this thesis have not been submitted in any other University / Institute for award of any degree.

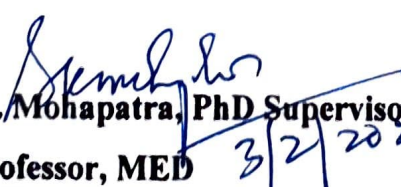


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*To the “Great Master”
and dear parents*

Abstract

Coal contributes a major fraction of the installed capacity. It is estimated that around 44 – 45% of the coal reserves are young, which contain high moisture and high ash coals. Presently, the coal water suspension has successfully replaced diesel in a variety of applications viz. low speed marine engines, diesel fired furnaces and stoves. In some places, the washed/pulverized coal is directly transported from the mine to the powerplant using the slurry pipelines. These systems are reliable, clean/dust free, and economical. Besides, these systems have very few chances of failure and are mostly used in the regions, where the mines are not accessible by rail/road transportation. In such applications, the coal fraction in the CWS can range from 10-60% (upper limit decided by the rank of coal; higher the rank, higher can be the upper limit) depending upon the factors like availability of coal, pumping power, the rank of coal, throughput required, etc.

The high rank coal reserves are limited and getting depleted at an escalating rate to meet the demands of growing population. This concern has compelled the scientific community to use the young high ash coals. The coal water slurry fuel is one promising way to use low quality coal as an economical liquid fuel. Due to poor physico-chemical and surface properties of such coals, these systems appear to be unsustainable. This research presents a sustainable solution to use the existing low-quality coals efficiently without any modification in the already installed systems. The Indian coal possessing high moisture and oxygen functional groups was successfully treated using hydrothermal and microwaves irradiation, resulting in the improvement of morphological, physical, chemical and rheological characteristics. These changes are due to the elimination of the Hydroxyl (-OH-) and Carboxyl (-COOH-) groups, which are observed to contribute the most to the hydrophilicity of the coal particles. The morphological investigations reveal the breaking up of the bigger particles into fines upon microwave irradiation. The treated coals exhibit physico-chemical and surface properties similar to high rank coals but still possess high ash that does not pose any issue with the normal operation of the system.

The hydrotransport of the industrial powders and bulk solids such as minerals, mineral tailings, coal and ash is considered to be an efficient mode of transportation. The pipelines ranging from a few meters to few kilometers in length are used for such transportation purposes. The pipe fittings such as converging section, diverging section and elbow are an integral part of these pipelines. The complex solid-liquid flow of coal water slurry through conical

converging sections and diverging sections is simulated using the computational fluid dynamics approach. Nine different geometries of the sections are analyzed in the present study. The length of the section is the key variable in the geometric variations and ranges from 0.05m to 0.6m. The influx velocity at the entrance of all the sections is varied in the range of 0.5 m/s to 5 m/s. The mass concentration of the solids dispersed inside the liquid phase is varied from 10 to 60%. The results generated by the computational fluid dynamics tool are in good agreement with the experimental data. The design of the section is evaluated based on results obtained for three characterization parameters viz. pressure recovery coefficient, head-loss across the diverging section and volumetric efficiency. The 0.3m long diverging section is found to be the optimum design for best pressure recovery, maximum volumetric efficiency and lowest head-loss.

If not well addressed, the issue of particle settling in such pipelines can lead to blockage and even bursting of the pipeline due to the continued deposition of the solids. The present study proposes the introduction of a twisted pipe section of a suitable length and geometry, that produces enough turbulence in the flow, sufficient for the re-dispersion of the already settled particles and check their further deposition. To achieve this objective, 5 different geometries (each having 4 different lengths) of twisted pipes are designed and used to model the dynamics of the particles' flow through them and in their downstream region. A low influx velocity (where maximum settling is expected) of 0.5 m/s is selected for all the cases and the influx solids' mass concentration ranges from 40-60%. The results generated by the commercial CFD software are in good agreement with the experimental data. The parameters viz. mixing index, pressure loss, and specific energy consumption are evaluated to choose the best design of the twisted pipe section. The 0.2m long 3 lobes twisted pipe section is found to deliver the suspension of highest homogeneity at the cost of a slight increase in pressure loss and specific energy consumption. The present solution ensures the mitigation of particles' settling and the other related issues

Keywords: Hydrotransportation, Coal water slurry, Particle size distribution, Rheology, Computational Fluid Dynamics, Pipe fittings, Twisted pipes

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Acronyms

Greek Symbols

α	Complete angle of converging section
β	Ratio of smaller to larger diameter of the section
$\dot{\gamma}$	Shear rate in internal flow
Γ	Blending function
ε	Turbulence dissipation rate
ζ_S	Bulk solids viscosity
Θ_S	Granular temperature
κ	Von Karman constant
λ	Jet velocity ratio
μ	Dynamic viscosity of the suspension
μ_S	Shear viscosity of coal
$\mu_{Collision}$	Collisional viscosity
$\mu_{Friction}$	Frictional viscosity
$\mu_{Kinetic}$	Kinetic viscosity
μ_P	Shear viscosity of water
μ_T	Turbulence dynamic viscosity
ν_T	Kinematic eddy viscosity
ξ	Particle shape factor
ρ	Density of the slurry/suspension
ρ_i	Density of either phase
ρ_L	Density of liquid
τ_P	Liquid phase stress tensor
τ_S	Solid phase stress tensor
τ_w	Wall shear stress
τ_{wL}	Wall shear stress in laminar flow
τ_{wT}	Wall shear stress in turbulent flow
ϕ	Angle of attack
ω	Specific dissipation rate
$CF, \beta^*, \alpha, \beta, \sigma_k, \sigma_\omega$ & $\sigma_{\omega 2}$	Closure coefficients of the turbulence model
δ	Boundary layer thickness
δ^*	Displacement layer thickness
η_V	Volumetric efficiency

Other symbols

AR	Area ratio
AR_e	Idealized equivalent area ratio

C_M	Solid concentration by mass
C_R	Roughness constant
C_{SP} or C_{PS}	Inter-phase exchange coefficient
C_{SS}	Maximum settled concentration
C_D	Drag coefficient
C_L	Lift coefficient
C_P	Actual pressure recovery coefficient
C_{Pi}	Ideal pressure recovery coefficient
C_{Pie}	Idealized equivalent pressure recovery coefficient
C_{va}	Average volumetric concentration at the plane
C_{vm}	Virtual Mass coefficient
d_s	Solid particle diameter
d_{50}	Median particle diameter
D	Diameter of the horizontal pipe
D_1	Diameter at section's inlet
D_2	Diameter at section's outlet
e_c	Coefficient of restitution
E	Empirical constant used for near wall flow formulation
f	Drag function
f_d	Darcy's friction factor
f_f	Friction factor of the flow
F_S	Total force acting on solid particles
F_P	Total force acting on the fluid
$\vec{g}$ or g	Acceleration due to gravity
h	Headloss across the flow domain
H_L	Headloss across a pipe fitting
Y_r	Probability of inter-particle contact
k	Turbulence kinetic energy
k_a	Turbulence kinetic energy at wall adjacent cell centroid
K	Consistency index
K_B	Bendloss coefficient
K_F	Friction pressure loss coefficient
K_L	Local resistance coefficient
K_R	Physical roughness of the pipe surface in meters.
K_T	Total pressure loss coefficient
L_S	Separation length
m_C	Known mass of coal sample is taken in the pycnometer
m_{PCW}	Total mass of pycnometer, coal and water
m_{PW}	Total mass of pycnometer filled completely with water
M_L	Mass of liquid in the slurry

n	Flow index
$N - S$ equations	Navier Stokes equations
$\frac{\Delta p}{\Delta l}$	Pressure loss per unit length of the pipe
P_{inlet}	The pressure at the inlet of the upstream pipe
P_{outlet}	The pressure at the outlet of the downstream pipe
P_s	Bulk solids pressure
P_t	Thermodynamic pressure
P_1	Static pressure at section's inlet
P_2	Static pressure at section's outlet
Re_c	Reynolds number between coal and water (based on the relative flow between coal and water)
Re_{cr}	Critical Reynolds number
Re	Reynolds number of the suspension/mixture relative to the pipe wall
Re_{sph}	Reynold's number for an equivalent spherical particle
Re_y	wall-distance based Reynold's number
s	Specific gravity of the dispersed phase
SEC_w	Specific energy consumption in terms weight of the bulk solids
SEC_m	Specific energy consumption in terms mass of the bulk solids
t_r	Particle relaxation time
u	Average mixture velocity
u_i	Average velocity of either phase
u_{pw}	Mean particle velocity at the wall-adjacent cell centroid
u^+	Ratio of velocity perpendicular to the wall to the frictional velocity. Where frictional velocity is derived from wall shear.
u_{lam}^+	Dimensionless velocity in the laminar region
u_{turb}^+	Dimensionless velocity in the turbulent region
U_c	Critical velocity (settling)
v_t	Terminal velocity correlation
V_i	Volumetric concentration of either phase
V_L	Volume of liquid in the slurry
V_{LS}	Volume of the separated liquid after settling
V_S	Volumetric concentration of coal (assuming no particles settling)
V_{SC}	Maximum settled concentration (volumetric)
V_{sd}	Actually, delivered volumetric concentration
$\bar{V}_1$	Average slurry velocity at section's inlet
$\bar{V}_2$	Average slurry velocity at section's outlet
y^+ or y^*	Dimensionless perpendicular distance from the wall
y_a	Distance from the centroid of the wall adjacent cell to the wall

$\widetilde{y}^*$

Scalable wall function

Abbreviations

<i>BC</i>	Boundary conditions
<i>CFD</i>	Computational Fluid Dynamics
<i>CS – I</i>	Coal sample I (size: -75µm)
<i>CS – II</i>	Coal sample II (size: -106µm)
<i>CWS</i>	Coal water slurry
<i>HAC</i>	High ash coal
<i>HT</i>	Hydrothermal
<i>HTT</i>	Hydrothermal treatment
<i>EDS</i>	Energy dispersive spectroscopy
<i>ESP</i>	Electrostatic precipitator
<i>MW</i>	Microwaves
<i>MWT</i>	Microwave treatment
<i>NO_x</i>	Oxides of Nitrogen
<i>OPEC</i>	Organization of the Petroleum Exporting Countries
<i>PSD</i>	Particle size distribution
<i>RNG</i>	Re-Normalisation Group
<i>RPM</i>	Revolutions per minute
<i>SEM</i>	Scanning electron microscope
<i>SO_x</i>	Oxides of Sulfur
<i>SST</i>	Shear stress transport

Publications from the present research

International SCI Journals

1. **Singh, Harmanpreet**, Satish Kumar, and Saroj Kumar Mohapatra. "Improved Slurryability and Rheological Characteristics of High Ash Indian Coal by Hydrothermal Treatment." *International Journal of Coal Preparation and Utilization* (2018): 1-13. doi: 10.1080/19392699.2018.1461624 (I.F.: 2.03, Q2)
2. **Singh, Harmanpreet**, Satish Kumar, and Saroj Kumar Mohapatra. "Microwave-assisted rapid upgrading of Indian high ash coal and its blending with untreated coal to improve the slurryability and rheological characteristics of blended slurry." *International Journal of Coal Preparation and Utilization* (2018): 1-17. doi: 10.1080/19392699.2018.1515738 (I.F.: 2.03, Q2)
3. **Singh, Harmanpreet**, Satish Kumar, and Saroj Kumar Mohapatra. "Improved surface morphology by reforming of oxygen functional groups of Indian coal upon irradiating with microwave radiations." *International Journal of Coal Preparation and Utilization* (2020): 1-19. doi: 10.1080/19392699.2020.1760854 (I.F.: 2.03, Q2)
4. **Singh, Harmanpreet**, Satish Kumar, and Saroj Kumar Mohapatra. "Modeling of solid-liquid flow inside conical diverging sections using computational fluid dynamics approach." *International Journal of Mechanical Sciences* 186 (2020): 105909. doi:10.1016/j.ijmecsci.2020.105909 (I.F.: 4.631, Q1)
5. **Singh, Harmanpreet**, Satish Kumar, and Saroj Kumar Mohapatra. "Design and Modelling of a Self-Dispersing Twisted Pipe to Mitigate Settling in Coal Water Suspension", *Advanced Powder Technology* (2020) doi:10.1016/j.appt.2020.12.012 (I.F.: 4.217, Q1)

International Scopus Indexed Journals (via conference proceedings)

1. **Singh, Harmanpreet**, Satish Kumar, and Saroj Kumar Mohapatra. "Improvements in the Rheological Characteristics of the Complex Coal Water Suspension Using Gasification Derived Waste Water." *Journal of Physics: Conference Series*. Vol. 1276. No. 1. IOP Publishing, 2019. (iCRAFT-2018, BITS Pilani, Dubai Campus)

Conference proceedings

1. **Singh, Harmanpreet**, Satish Kumar, and Saroj Kumar Mohapatra. "Determination of head loss in a slurry pipeline for the flow of coal water slurry using computational fluid dynamics approach" In *9th International Conference on Conveying and Handling of Particulate Solids (CHoPS-2018, University of Greenwich, UK)*
2. **Singh, Harmanpreet**, Satish Kumar, and Saroj Kumar Mohapatra. "Design and Modelling of a Self-Dispersing Twisted Pipe to Mitigate Settling in Coal Water Suspension", In *3rd International conference on Powder, Granule and Bulk Solids: Innovations and Applications (PGBSIA-2020, Thapar Institute of Engg. and Tech., Punjab)*

Chapter 1

Introduction

The present research is an outcome of a strong motivation to provide a sustainable solution to the large developing nations, that are presently struggling with fast depleting reserves of high-quality coals and eye-watering prices of petroleum and its derivatives. This study also aims at providing an energy efficient route to feed the pulverized coal, thereby reducing the technical minimum operational power consumption of the existing power generation plants.

1.1 Coal energy: A reliable, affordable, abundant, and well-established source of energy

In most parts of the world, the energy generation framework is significantly dependent upon fossil fuels. As per the World Energy Outlook 2019, around 64% of electricity was generated using fossil fuels. Coal alone contributes 38% to the total electricity generation [1]. This figure rises to 53.3% in India, where it contributes around 374.2 GW to the total installed capacity as of November 2020 [2].

Coal is a fossil fuel, that is said to form from the dead flora and fauna followed by the accumulation of layers of dust for millions of years [3]. The heat and pressure conditions play a critical role in the characteristics of the final product [4]. **Coalification** describes the extent to which the coal has attained maturity. The maturity or rank of the coal is mostly analysed according to the physico-chemical characteristics of the coal [5]. The parent-dead fauna and flora vary with the geographical locations, type and variety of vegetation, and species of animals that existed in the region. So, the physico-chemical characteristics of coal also vary significantly with the geographical location of its origin [6–8]. Coal is a fossil fuel, that defies any classification system because:

- It is a heterogeneous mixture of moisture, organic matter and mineral matter in widely variable proportions [9–12]
- Coals having similar chemical composition may have different physical properties while coals originating from different locations may have similar properties[13–16]
- The fraction and composition of the mineral matter vary significantly with the origin of the coal and strongly influences the properties of the coal[17,18].

The globally accepted and established classification coal is made on the basis of its rank [4]. According to this classification system, coal can be of 4 types:

1. **Anthracite:** The maximum amount of volatile matter in anthracite is mostly less than 7%, maximum fixed carbon can be as high as 92% on a dry mineral matter free basis. The calorific value of anthracite lies in the range of 59 to 60 MJ/kg of dry coal [19–21]. These coals are not used as CWS due to low reactivity
2. **Bituminous:** The bituminous coals on d.m.m.f. basis contain around 60 to 85% fixed carbon, 15 to 40% volatile matter. The calorific value of the bituminous coal lies in the range of 25 to 60 MJ/kg [22–26]. These coals are mostly used as CWS due to their high carbon content and high reactivity.
3. **Sub-Bituminous:** These coals have slightly high moisture as compared to the bituminous coals and carbon content less than the bituminous coals. The heating value of sub-bituminous coals is 19-30 MJ/kg. Coal water slurries from the sub-bituminous coal have average flow characteristics and stability to higher moisture and lesser hydrophobicity.
4. **Lignite:** The lignites have the lowest rank. The elemental carbon in lignites is usually greater than 60% and may have moisture as high as 25% [27,28]. The calorific value of lignite lies in the range of 9.5 to 19 MJ/kg [29]. Coal water slurries from the lignites have poor flow characteristics and stability to high moisture, ash and hydrophilicity.
5. **Peat:** (Not considered as coal in most of the regions): Peat is a compact naturally occurring carbonaceous matter composed of mostly recent decayed vegetation. The water content in peat is extremely high. The dehydrated peat can be used as a solid fuel [30–33]. Coal water slurries are not prepared from peat.

1.2 Coal water slurry

Coal water slurry is a complex, high energy density, multiphase fuel. The rheological characteristics of the coal water slurry are quite comparable to those of the heavy oil. So, it is an attractive replacement option for heavy oil [34]. The physico-chemical characteristics of the CWS are dependent on the origin, rank and composition of the coal. The other physical parameters such as the concentration of coal, particle size of the coal. For the previous decades, the world has witnessed a significant rise in the prices of crude oil and natural gas, but the coal has always been an inexpensive, reliable and abundant fuel [35].

1.2.1 Background:

The first glimpse of the development of the coal water slurry dates back to 1940s where it was proposed as an attractive solution to address the issues with the scarcity of fuel caused as a result of the war [36]. The abundance of oil at that time lead to the fading of the concept but once again the idea was revived in the 1974 OPEC meeting. A significant development in this field was seen in countries like the United States, England, Germany and Japan after this meeting [37].

The greatest development in the production of the CWS started in the US during 1980s. Bituminous coal was mainly used at that time. The initial attempts were observed to be unsustainable due to the low calorific value of the slurry. But in the later stages, dispersants were utilized to improve the stability and rheology of the slurry fuel[38–43]. Table 1.1 presents a summary of some of the major applications of CWS across the globe.

Table 1.1 Major industrial applications of coal water slurry in the past and present

Country	Major industrial application(s)
Canada [44]	• First test was conducted in 1982 by New Brunswick Electric Power Commission and Cape Breton development corporation for firing in boilers • During 1984-1985, Canada Cement LeFarge Ltd. totally shifted from coal-to-coal water slurry
China [45,46]	• Extensive research in this field has been reported since 1982 and is ongoing. • Has 5 production plants with an annual load of not less than 85000 tons • Hydrotransportation pipelines handling a load of not less than 7 million tons/year and transporting to distances as long as 850 km. • In 2000s, many large heavy oil operated power generations units were slightly modified to run on coal water slurry to address the issue of pollution in context to the Beijing Olympics.
Russia [41,42,-47,48]	• The first glimpse of coal water slurry was seen in Russia during 1940s. • The popular application was the hydrotransportation of coal to a distance of 298 km

	• Today most of the steam coal is used for the production of coal water slurry fuel meant to be fed in boilers.
Sweden [49]	• The major developments in the coal-water slurry fuel were seen during 1970s. • The Swedish company Carbogel AB was set up to produce coal water slurry from washed coal.
Unites States [50,51]	• Commercialization of the coal water slurry started in 1980s • A coal-fueled boiler was first fed with coal-water slurry in 1985 for long term operation of the plant.

1.2.2 The process:

Coal water slurry is a mixture of pulverized coal and water, that is combustible. The coal water slurry if used for combustion applications, should necessarily have at least 60% coal by mass [52,53]. However, for hydrotransportation applications, even lower concentrations can be considered. The design of the slurry transportation system is greatly influenced by the physico-chemical characteristics of solids being transported [54]. These characteristics are [55–57]:

- Particle size and particle size distribution of the coal
- Rank of coal (in context to volatile matter and ash in the coal)
- Specific gravity of coal
- Hydrophobicity of the coal
- Final pH of the slurry
- Maximum settled concentration of coal in the slurry
- Presence of dispersants/surfactants

1.2.3 Properties of coal water slurry:

The properties of coal water slurry vary significantly with coal used for the production.

1.2.3.1 Particle size and particle size distribution:

The size of the particles used for the production of coal water slurry plays an important role in the rheological characteristics of the final product. For the production of slurry for hydrotransportation, the maximum permissible particle size is $\sim 300 \mu\text{m}$. The particles greater than this size can pose a issue of pipeline blockage due to excessive settling of the particles. If for combustion applications, the particles' size is reduced from $300 \mu\text{m}$, the dynamic viscosity of the slurry is observed to decrease [58]. But very fine size of particles ($< 53 \mu\text{m}$) also leads to

a significant increase in the dynamic viscosity of the slurry. In such a situation, the particle size distribution plays its role, where based on the type of coal, the particle size distribution (example bi-modal PSD) is optimized for the lowest possible dynamic viscosity of the slurry [59,60].

1.2.3.2 Rank of the coal

Rank of the coal explains the degree of maturity that the coal has attained. As the coal matures under high temperature and pressure, there is a significant variation in the key constituents like organic matter and mineral matter. As the coal matures, the organic matter goes increasing while the mineral matter goes on decreasing. The organic matter is hydrophilic in nature. So, when a given coal sample is used to produce the coal water slurry, the mineral matter holds the water to itself and causes lack of water in the slurry. Now, if the coal contains high mineral matter, the dynamic viscosity is expected to be high, while a low dynamic viscosity is expected in the case of coal containing low mineral matter. So, it can be said that as the rank of coal increases, the dynamic viscosity of the coal decreases. But the rank of the coal in context to combustion applications has an upper limit. The use of anthracite coal for the production of coal water slurry fuel is greatly discouraged because the volatile matter, a necessary component for the ignition of the fuel is present in very small fractions. The anthracite coal water slurries are hard and unsustainable to burn due to low reactivity of anthracite. So, bituminous coal or popularly known as steam coal is the most preferred variety of coal for coal water slurry fuel production.[61,62]

1.2.3.3 Specific gravity of coal:

The specific gravity of the coal is measure of relative density of coal with respect to water. The organic matter in the coal has a specific gravity of ~1.25-1.4 while the mineral matter has a specific gravity of ~2.55. The specific gravity of the organic matter increases with rank of the coal. The lignites have the lowest specific gravity of around 1.29-1.3, due to the presence of high moisture (sp. gravity: 1). The specific gravity of the bituminous coal is 1.34 due to replacement of excessive moisture by organic and inorganic matter. The anthracite coal have the highest specific gravity of 1.42 to the presence of relatively dense organic matter. The washery rejected coal, containing very high ash can have specific gravity of >1.6. The denser coal will try settle fast as compared to lighter one. But this effect is often dominated by other surface properties. Also, the specific gravity of the coal varies over a narrow range, so the effect of specific gravity is not high pronounced [63].

1.2.3.4 Hydrophobicity of coal:

Hydrophobicity is a net surface property of the coal, that it possesses due to the presence of different oxygen functional groups, heavy hydrocarbon, aromatic groups and mineral matter. The mineral matter of the coal is hydrophilic in nature and is responsible for absorption of water. More the mineral matter, more will be water holding capacity of the coal and lesser will be the hydrophobicity. The oxygen functional groups such as -OH and -COOH are also hydrophilic in nature. The low rank coals have these functional groups in abundance and these mostly constitute the light volatile matter present in the coal. So, the hydrophobicity of the coal is inversely proportional to the fraction of these volatile acid groups. The heavy hydrocarbons and aromatic groups constitute the heavy volatile matter of the coal and are mostly hydrophobic in nature. The abundance of these heavy volatiles increases the hydrophobicity of the coal. The hydrophobicity of the coal directly affects the rheology of the coal water slurry and the hydrophobic coal are observed to produce slurry of less dynamic viscosity. Such coals do not even require any type of surfactant for the said purpose [64,65].

1.2.3.5 Final pH of the slurry:

The final pH of the slurry is dependent on the coal used for the production of slurry. If the pH of the slurry falls in the acid range, say 3.1 the presence of too many cations may lead to increase in the agglomeration of the coal particles. This will push the water out from the agglomerate, leading to reduction in the dispersion stability of the slurry. This causes the increase in the dynamic viscosity of the slurry. The slurry performs very well in the presence of anionic surfactants, that are said to improve the stability of the slurry [66,67].

1.2.3.6 Maximum settled concentration of the slurry

After preparation of the slurry, the equilibrium stability tests are conducted to analyse the stability of the slurry. The slurries containing very large particles >500 μm are more-prone to settling as compared to the fine particulate slurries. If the particles of the slurry settle too much in the pipe line, they can reduce the cross-sectional area, through which the flow is actually taking place, leading increase in the headloss in the pipeline and owing to increased operational and maintenance cost of the plant [68].

1.2.3.7 Presence of dispersants/surfactants

The dispersants are of three types, cationic, anionic and non-ionic. The use of cationic dispersant is limited by the agglomeration of the coal particles. The presence of too many cations in the slurry leads to reduction in the dispersion stability of the slurry. On the other

hand, the anionic dispersants are used to improve the dispersion stability of the coal water slurry. The non-ionic dispersants do not have any significant effect on the dispersion stability of the slurry [69,70].

1.2.4 Production of coal water slurry

While preparing the coal water slurry, big chunks of coal are first crushed using jaw crushers or hammer mills. In this operation the size of coal is reduced from chunks of few centimeters in diameter to coarse particles of size few millimeters. This size is still very large for the production of coal water slurry. So, these coarse particles are then fed in ball mills for dry pulverization. The coal is milled to size of $\sim 150\ \mu\text{m}$. Based on the requirement, the particles size and particle size can be adjusted using the standard sieves. The slurry is then prepared by simply adding the measured quantity of water and surfactant followed by continuous stirring until a (pseudo)homogenous mixture is obtained [71]. Another way of size reduction is wet pulverization using a hydroshock or wet milling machine as presented in Figure 1.1.



Figure 1.1 A hydroshock machine with a 200litres hopper and a screw conveyor fabricated by DEWA Technologies, Russia. (Source: bit.ly/hrmn13)

These machines are very popular in Russia and produced on a large scale. This machine has an advantage of significantly lower energy consumption as compared to the dry ball mill. In the case of hydroshock machine, the direct output is the required coal water slurry, ready to be fed in combustion chambers [72].

1.2.5 Benefits of coal water slurry:

The coal oil slurry, developed a century ago was considered an economical replacement to the expensive crude. But the development of coal water slurry made the already operational systems even more economical by eliminating the cost of oil. The coal water is more energy dense as compared to the coal oil slurry. The coal water slurry has the following benefits over the other fossil fuels and other combustion methods:

General benefits:

- The existing oil/gas fired or fluidized bed combustors can easily shift to coal water slurry, without any major modification.
- Coal water slurry can be operated under wide load ranges.

In reference to oil:

- The coal water slurry has higher energy density as compared to oil despite having almost same dynamic viscosity.
- The recent decades have seen a significant rise in the crude oil prices, still the coal is the most economical fossil fuel. The specific cost of coal water slurry can be 50 to 70% less than oil.
- The temperature of the combustion chamber is 150-200°C lower when operated using coal water slurry than the oil. This helps in controlling the NO_x and SO_x emissions that are a direct function of combustion chamber's temperature.
- The coal water slurry is fireproof and explosion proof.

In reference to dry pulverized coal:

- The hydrotransportation is a high economical, easy, dust proof, and environment friendly way to transport coal as compared to rail/road transportation. The rail/road transportation can still be utilized for the transportation of the coal water slurry,
- The coal water slurry is easy to store. The coal water slurry is weatherproof. The storage does not require any special yard. The coal water slurry storage can save up to 50% of the land area.
- The ESPs and bag filters can still be used for the collection of the fly ash.
- The carbon combustion ratio of over 99% can be easily achieved in the coal water slurry fuel.

- The pulverization using ball mill or similar method is highly energy extensive. The hydroshock wet milling can perform even better and consuming only a small fraction of energy consumed in dry milling.

1.2.6 Problems identified with the present systems:

The coal water slurry has been a well-tested and a sustainable fuel. Out of different types of available coals, the anthracite is not recommended due to lack of required reactivity in the fuel for its ignition. The lignites due to excessive inherent moisture, ash and well-developed pore structure have poor slurryability. The dynamic viscosity of the resultant slurries is significantly high, that may increase the pumping costs. So, the only well-suited option is the bituminous coal.

It is estimated that around 44 – 45% of the coal reserves are young, which contain high moisture and high ash coals, also termed as lignites and high ash coals (HAC) respectively. It is a great challenge to produce energy from HAC, as it is difficult to clean and grind it and due to its spontaneous igniting nature, it is quite difficult to handle.[73,74] The fast depletion of the high-quality coal reserves has forced the energy producers to shift to lignites and HACs [75]. The power generation systems using lignites or HACs usually face the issues such as low overall plant efficiency, increase in power generation costs, increase in plant maintenance, and increase in ash transportation costs [76,77]. The coal water slurries prepared from lignites and HACs have poor stability and slurryability. The low-quality coals are available in abundance throughout the world and in the near future, the low-quality coal reserves will be a significant component of the energy generation systems [78]. So, the question arises what will be the future of the coal water slurry?

The pipelines used for the hydrotransportation of coal have different pipe fittings viz. straight pipes, bends, elbows, converging sections and diverging sections. The experimental and computational studies on the flow of coal water slurry through straight pipes and bends have progressed significantly [79–83]. The computational fluid dynamics is a versatile field of study to virtualize any realistic problem and has a tendency of producing results close to the realistic ones. Previously authors have used the CFD approach for heat transfer applications [84] and multiphase flow predications in cyclones and fluidized bed reactors. [85,86]. The CFD approach has been widely used to model the flows on rough beds and ducts [87–89]. The cavitation phenomenon, quite common in pipeline transportation can be predicted using CFD codes to prevent a major loss of the system [90–92].

At present, universal correlations are not available, that can be utilized to predict the flow behavior of the coal water slurry at different concentrations of coal through the other pipe fittings viz. elbows, converging sections and diverging sections.

1.3 Proposed objectives and scope of the study:

1.3.1 Objectives:

The primary objective of the study is analyse the rheological and headloss characteristics of the coal water slurry prepared from an Indian high ash coal and to develop a sustainable methodology to efficiently utilize these high ash coals. The specific objectives to meet the primary objective are following:

1. Investigation of the rheological behavior of the coal slurry at different concentrations with and without additives.
2. Investigation of slurryability and rheological characteristics of high ash pulverized coal treated with hydrothermal and microwave treatment processes.
3. Investigation of headloss characteristics in horizontal pipeline for the flow of coal slurry at different solid concentration with and without additives.
4. Numerical modelling of solid liquid flow through different pipe fittings such as converging section, diverging section and elbow.

1.3.2 Scope of the study:

The present study is conducted on an Indian high ash coal collected from the Guru Nanak Dev Thermal Power Plant, Bathinda, Punjab. The source of coal to this thermal power plant is the Makum coal field, Assam. The following studies are conducted on the coal sample:

1.3.2.1 Physical characteristics:

- Particle size distribution
- Specific gravity
- Settling characterization

1.3.2.2 Chemical characteristics:

- Proximate analysis
- Ultimate analysis
- Energy Dispersive X-ray spectroscopy

1.3.2.3 Morphological characteristics:

- Scanning electron microscopy

1.3.2.4 Rheological characteristics

1.3.2.5 Headloss characteristics

1.3.2.6 Computational Fluid Dynamics modelling:

- Elbow
- Converging section
- Diverging section
- Twisted pipes (to address issues related to particles settling)

1.4 Methodology:

Coal sample is collected from Guru Nanak Dev thermal power plant, Bathinda, Punjab, India. The source of this coal is Makum coal fields of Assam. Post sampling, the coal is subjected to two different experimental routes. The detailed engineering flow charts followed are presented in Figures 1.4 and 1.5.

The rheological and headloss characteristics of the HAC are used to validate the CFD model. This CFD model is then used to predict the flow the slurry through different pipe fittings. The detailed engineering flow chart of the process followed is presented in Figure 1.6 The proposed objectives are achieved by:

- The sample collection from Guru Nanak Dev Thermal power plant and its preparation for various benchscale tests.
- The physical, chemical and morphological characterization of the prepared sample.
- Hydrothermal and microwave treatment of the coal followed by their physical, chemical and morphological characterization
- Rheological characterization of untreated, treated and blended samples by varying shear rate and concentration of coal in the slurry
- Headloss characterization of coal water slurry in a straight pipe at different influx velocities and coal concentration
- Computational fluid dynamics modelling of solid-liquid flow through different pipe fittings at a wide range of influx velocity and concentration of coal.

Experimental route-I: (Characterization of untreated and treated coals)

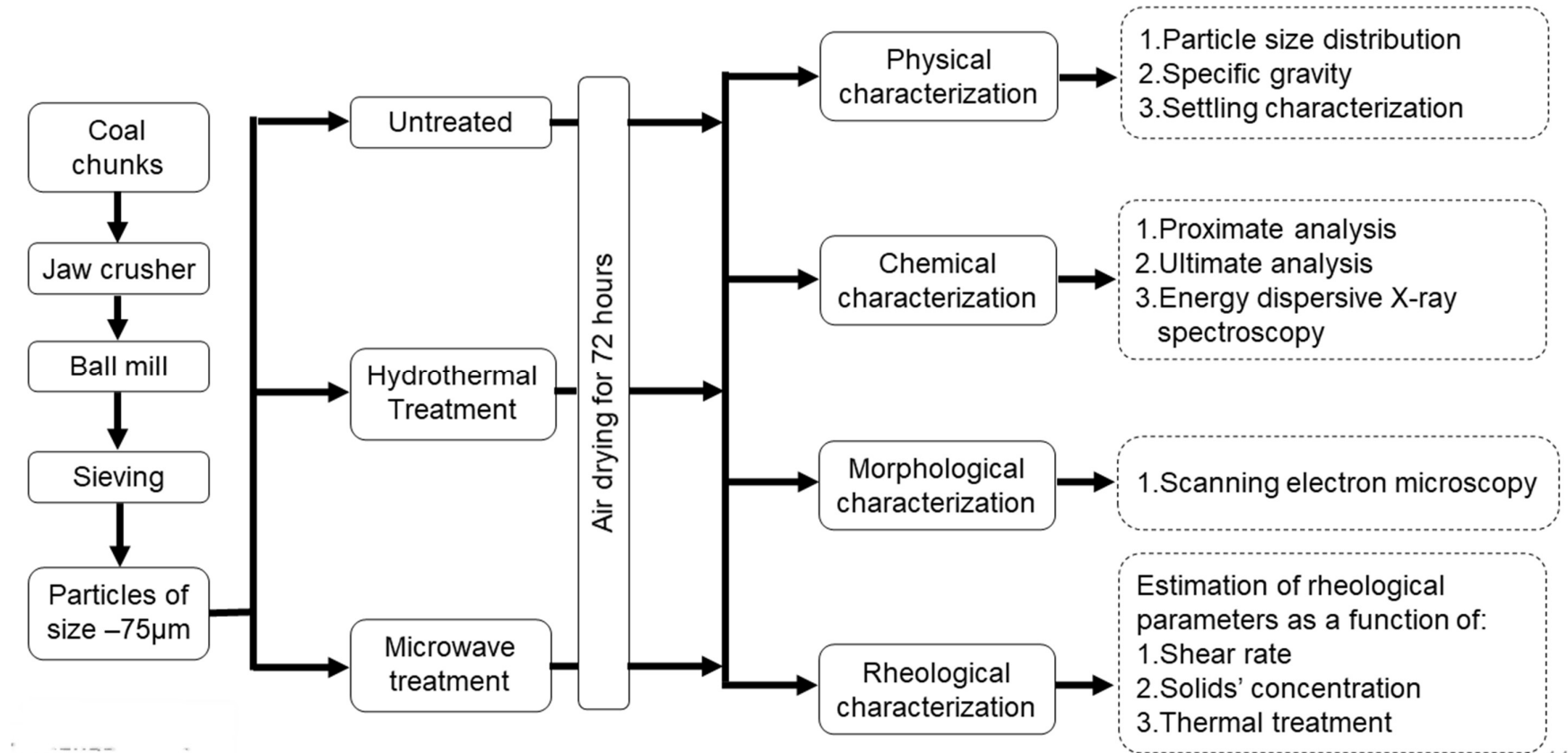


Figure 1.2 Engineering flow chart for experimental route-I

Experimental route-II: (Characterization of blends of untreated and treated coals)

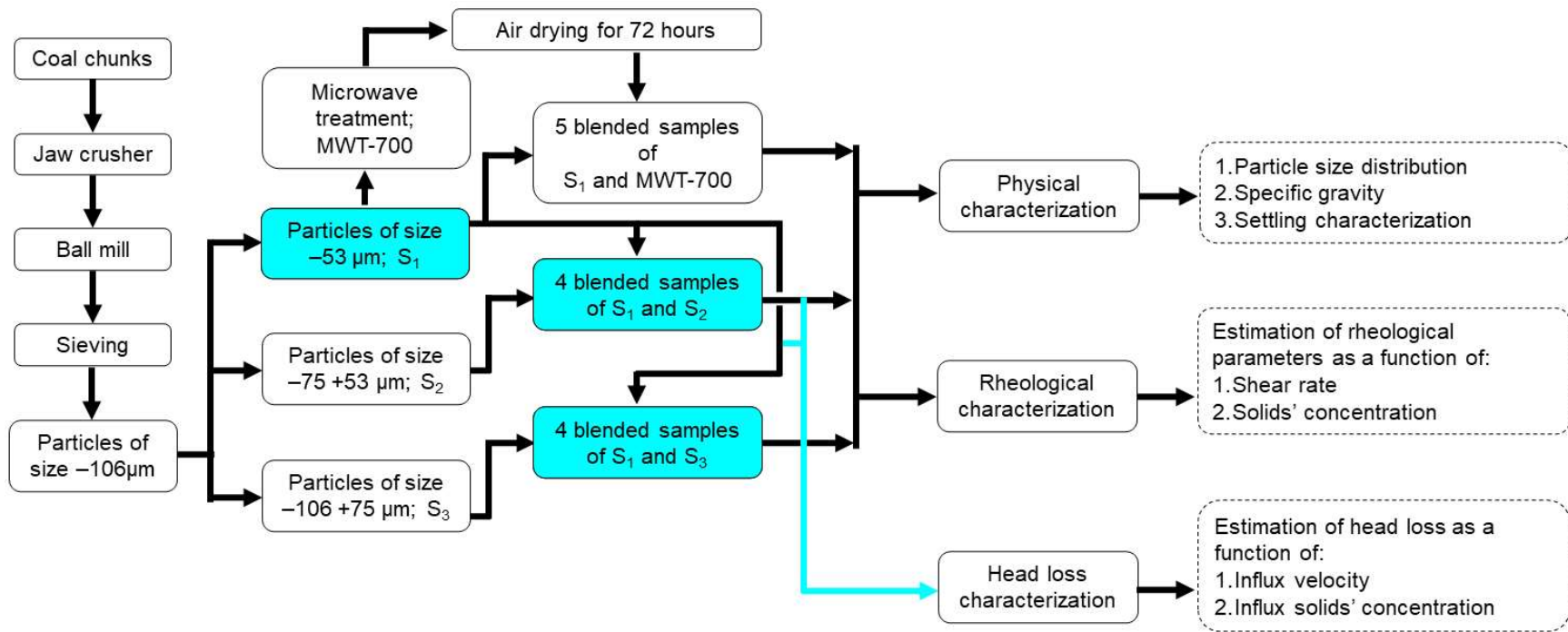


Figure 1.3 Engineering flow chart for experimental route-II

Methodology followed for computational modelling of solid-liquid flow through pipe fittings

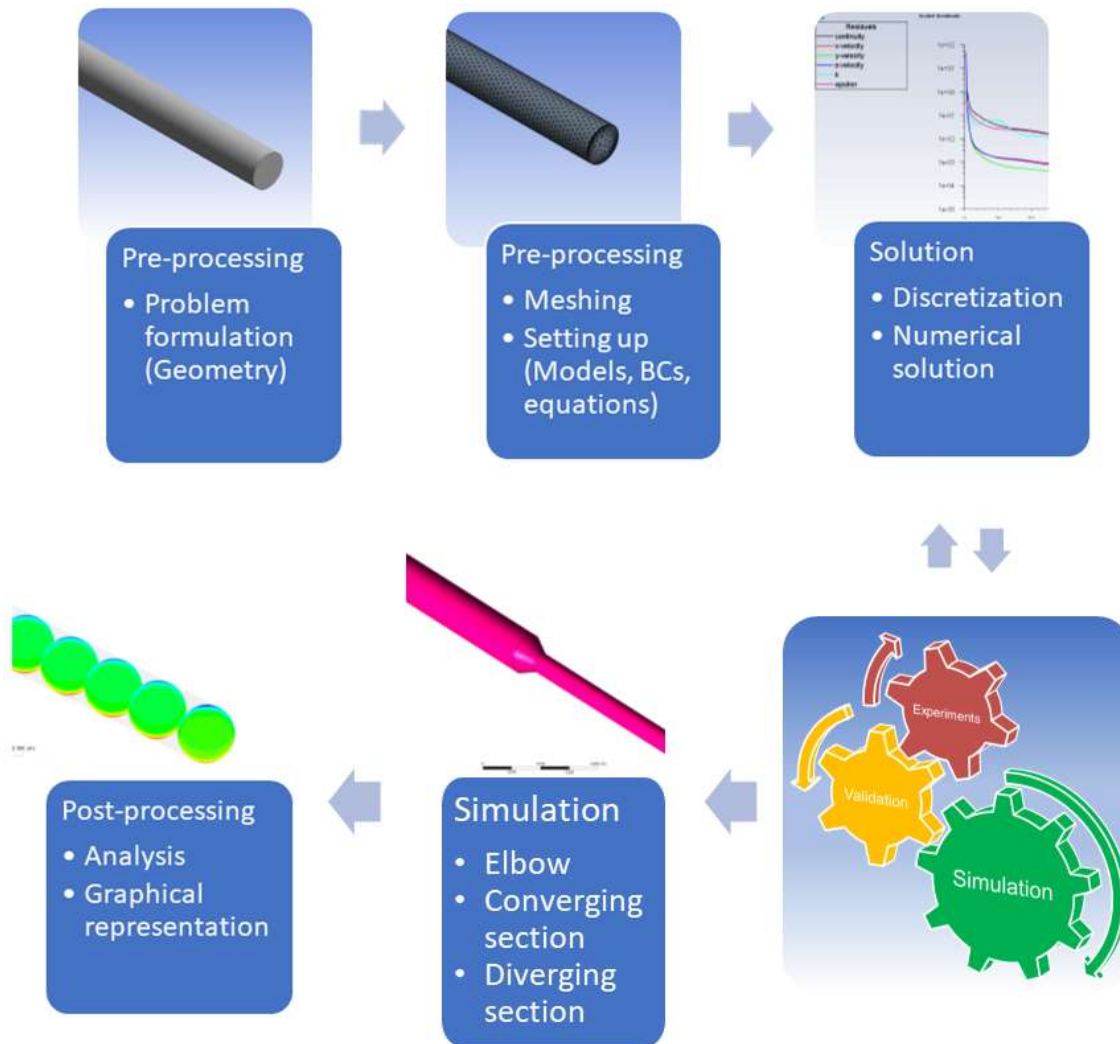


Figure 1.4 Engineering flow chart for CFD modelling of solid liquid flow through pipe fittings.

1.5 Organization of thesis:

The present study aims at developing experimental methods for the improvement of physico-chemical and rheological characteristics of high ash coals and their slurries. This is expected to make the hydrotransportation process more sustainable and efficient. The whole study and the results are organized in different chapters and sections of the thesis.

A brief description of the organization of the thesis is as follows:

-
- **Chapter 1-** Presents the comprehensive information on coal and its types. It also describes the past and present of the coal water slurry. The identified problems with the coal water slurry are introduced followed by the specific objectives of the present research. This chapter highlights the methodology followed to achieve the proposed objectives.
 - **Chapter 2-** Presents the detailed thematic literature review on the rheological characteristics of coal water slurry, thermal treatment of low quality of coals, headloss characteristics of slurry flows in pipes and pipe fittings and numerical simulation of solid-liquid flow in the pipes and pipe fittings.
 - **Chapter 3-** Present the methodology and results of physico-chemical and morphological characterization of the untreated, treated and blended coals and coal water slurry
 - **Chapter 4-** Presents the methodology, test setup and results of rheological characterization of untreated, treated and blended coals' water slurry
 - **Chapter 5-** Presents the pilot test loop, methodology and results of experimental analysis of headloss during the hydrotransportation of fine and blended coals at different velocities and concentrations of coal
 - **Chapter 6-** This chapter presents the methodology and results of numerical modelling of the solid-liquid flow through the pipe fittings using the commercial CFD code ANSYSTM Fluent.
 - **Chapter 7-** Summarizes the observations and important conclusions drawn from the experiment and numerical studies. This chapter also highlights the possible scope of extension of the present work.

Chapter 2

Literature review

2.1 Introduction:

The homogenous, heterogenous or pseudo-homogenous mixture of solids in a carrier liquid is called as slurry. The physico-chemical and rheological properties of the slurry as a strong function of carrier fluid's rheology, temperature, concentration of dispersed phase, particles' size distribution and particles size. [93–95] These slurries are mostly prepared from water but several other fluids viz. alcohol, biodiesel, vegetable and mineral oils can be used for the preparation of the slurries.

The fast depletion of the oil reserves is an important issue these days. To a great extent, these issues have been resolved in countries like Russia, by its replacement with a more versatile fuel; coal slurry. There are a variety of coal slurry preparation techniques. These are classified based on the type of carrier fluid used for the preparation of the coal water slurry. Some of the popular coal slurries are: Coal methanol water slurry, coal methanol slurry, coal oil water slurry, coal oil slurry, coal sludge slurry and coal water slurry. The most commonly used is the coal water slurry. It is composed of cheap materials viz. coal and normal water. The coal in the slurry, sometimes has a tendency to settle when stored for a long period. To increase the stability of the coal water slurry, certain dispersants are used [96–100].

2.2 Literature review on the rheological characteristics of coal water slurry:

The rheological characteristics of the slurry play an important role in the design of the slurry transportation system. Based on the rheological characteristics, the headloss in the slurry pipeline can be estimated and modelled. The rheological parameters such as yield stress, consistency index and flow index can be used to model the startup and operational characteristics of the slurry pipeline.

Aktaş et al. (2000) [101] studied the influence of surface active agent on the rheological characteristics of coal water suspension. The authors used British low-rank coal for their experimental investigation. The demineralization of the high ash coal was carried out and the final ash content was reduced to as low as 4% by mass. The experimental investigation indicated that higher solids' loading is possible in the sample, that have low

mineral matter. The slurry prepared from the high ash (24.5%) coal, appeared to unpumpable even at high dosage of dispersant. Demineralization is the key solution to make very high ash coal water slurry pumpable.

Li et al. (2000) [102] proposed a new method to produce the coal water slurry in the presence of ultrasound waves. The ultrasound irradiation leads to the generation of fines. The low dosage of additives is favorable for better solid loading. The ultrasonic irradiation leads to increase in the concentration of cations. At high ultrasound exposer, the cations dissolve in the coal water slurry and form bridging among the coal water slurry leading to the formation stable slurry structure. This leads to increase in the viscosity of the slurry.

Mishra et al. (2002) [103] Studied the rheological characteristics of coal water slurry prepared from coal collected from Orissa, India. The maximum size of particles considered in their study was around 90 μ m. The rheological characteristics were investigated on HAAKE rotational viscometer. The concentration of coal in the slurry was 50, 55 and 57% respectively. The results indicate that the shear stress increase with increase in shear rate. Similarly, with increase in concentration of coal, shear stress also increases. The viscosity is observed to decrease with shear rate, indicating a pseudoplastic behavior of coal water slurry. The viscosity is significantly increased when the solids' concentration is increased to 57%. The reason for this increase in apparent viscosity is the hydrophilic nature of coal.

Tiwari et al. (2004) [104] developed new additives to enhance the rheological characteristics and stability of the slurry. The Indian Ledo and Sirka coals collected from Assam were used in their investigation. Two new developed additives were anionic in nature and were naphthalene based and naphthalene-toluene based. They demonstrated that slurries of highly concentrated coals can be produced by just 0.8 to 0.9% dosage of the dispersant. The study also highlighted the benefits of wet milling over the dry milling. The wet milling can be used to produce particles of 21-22 μ m very easily. The low ash coal, containing low fraction of hydroxyl and carboxyl functional groups can be favorable for the production of the slurry.

Zhou et al. (2006) [105] analysed the effect of molecular weight of the dispersant on the rheology and the stability of the slurry. The study was Chinese Panjiang coal. The results described that fractions that had higher mass averaged molecular mass have better effectiveness in the improvement of the rheology of the slurry. The dispersant, Sodium lignosulfonate having molecular weight of 10000-50000 had the best performance.

Li et al (2008) [106] studied the dispersion effectiveness of different lignin series and naphthalene series dispersants. The Turbiscan Lab stability analyser is used to analyse the dispersion stability of the coal water slurry. It was observed that the lignin series dispersants have better performance as compared to the naphthalene-based dispersants. The clarifying zone height of the slurry prepared using lignin series dispersant is 68% of the other prepared using naphthalene series dispersant.

Li et al. (2009) [107] studied the rheology of coal sludge slurry. The Chinese coal sample considered in their study contains significant fraction of ash. The sewage sludge in the as received form contained high proportion of moisture. The coal sludge slurry is prepared from the sludge, that is not pre-dried and used as it is. So, the maximum possible solid load of the coal is significantly low. The maximum coal loading was increased to 61 and 60% using two different dispersants. The addition of the sewage sludge enhanced the dispersion of the slurry and eliminated the frequency of settling of the coarse particles. The coal sludge slurry behaves like a pseudoplastic fluid.

Li et al. (2010) [108] studied the effect of addition of sewage sludge on the rheological characteristics of coal water slurry. The results described that the addition of the sewage sludge leads to the increase in the yield stress of the coal sludge slurry. The addition of dispersant proves to be favorable in reducing the yields stress to 22.9 Pa. The coal sludge slurry behave like a pseudo-plastic fluid as the apparent viscosity is observed to decrease with increasing shear rate. The coal sludge slurry in the presence of NaOH and the dispersant has a quick recovery rate to the deformation cause as a result of shearing.

Zhou et al. (2011) [109] conducted static stability tests on a Chinese low rank coal. The proximate, elemental and functional group composition indicated the presence of a significant amount of oxygen functional group. These groups are carboxyl and hydroxyl groups. The presence of such hydrophilic groups helps coal to retain a significant amount of water. The low rank coal was coated with different surfactants to cut out the interaction of the oxygen functional groups from the water. The Span-40 surfactant proved to be the best performer. The stability of the slurry was significantly improved and the zeta potential of the coal particles was reduced from -77.5 mV to -91.4mV with use of Span-40

Buranasrisak et al. (2012) [110] conducted experiments to study the packing characteristics as function of particle size of coal. The experiments were also extended to the investigation of rheological characteristics different mono-modal, bi-modal and multimodal

samples. The results prove that the multimodal samples have better packing characteristics, leading to higher stability of the slurry. The mono-modal as well as bi-modal suspension behave like non-Newtonian and pseudoplastic fluid.

Yi et al. (2014) [111] performed rheological investigation on the Powder River Basin coal based coal water slurry. The slurry preparation was accompanied with the addition of ionic and nonionic dispersants. The ionic dispersant, sodium polystyrene sulphonate showed poor performance in enhancing the stability of the suspension. However, the non-ionic surfactant TritonX-100 perform good in improvement of the stability and rheology of the slurry. The non-ionic dispersant failed to perform well at higher concentrations of coal.

Tu et al. (2015) [112] developed a method to evaluate the packing of particles in coal water slurry. The results demonstrated that the packing density of unimodal slurries is lower than the bi-modal or multimodal suspensions. The consistency of the final produced slurry was not affected by the average sizes of the mixed samples. The present method can be used estimated the density of the tapped dry coal with some modification in the methodology.

Zhang et al. (2016) [113] studied the effect of the pre-absorption on the rheology of the coal water slurry. The results indicate that the pre-absorption filled the pores with water. The water added for the preparation of the coal water slurry remained in the particles' surrounding instead of penetrating into it. The activated carbon was observed to absorb the highest amount of moisture as compared to the lignite and coke. The addition kerosene in 1% concentration even proved to be beneficial. The pre-absorption was helpful in the improvement of the stability of the coal water slurry. The maximum solid loading was achieved in the case of pre absorption with moisture in the air.

Li et al. (2018) [114] developed three ionic dispersant using the tannic acid and itaconic acid. The dispersants were named as TIA-1, TIA-2 and TIA-3 respectively. The TIA-1 dispersant due to its 3-D structure is observed to have the best performance. The very small dosage of the dispersant was enough for the excellent dispersion and improved rheology of the slurry. The maximum loading of as high as 59% can be easily achieved using 0.1% dosage of the TIA-1 dispersant. The dispersant has both nonpolar and polar sites, beneficial for the efficient coating of particles leading to the improvement in the tension between water and coal.

Konduri et al. (2019) [115] studied the effect of oxidized sulfomethylated lignin and lignosulfonate on the rheological and dispersion characteristics of the coal water slurry. The

coal surface chemistry and particle size strongly decided the interaction of coal particles with either of the dispersant. The lignosulfonate has better performance due to greater degree of adsorption on the particles' surface. The mechanism for the improvement of the suspension stability is attributed by the increased steric and electrostatic repulsions among the particles by the action of the dispersant.

Wang et al. (2019) [116] studied the rheology of coal water slurry with a small dosage of alkaline additives. The study was focused on the sensitivity of the coal water slurry towards the concentration of the alkaline additive. Experiments showed that around 1% dosage of alkaline dispersant was sufficient for improvement of the stability of the particles. The higher dosage of the dispersant resulted in the deterioration of the rheological characteristics of the slurry. The low dosage of the dispersant increased the absolute zeta potential, so the coal particles in the presence of additives had more free water.

Pinchuk and Kuzmin (2020) [117] studied the thermo-physical properties of coal water slurry produced from different ranked coals. The focus of the study was on thermo-chemical activation of the coal with the help of TiO₂ nanoparticles. The addition of TiO₂ resulted in the increase in the thermal conductivity of the coal water slurry. Further, the time for the release of moisture was observed to fall by around 45% and the volatile evaporated at temperatures 30-50°C lower than the other case. The optimum concentration of TiO₂ nanoparticles was around 2.11%.

Zhu et al. (2020) [118] developed block polycarboxylate based dispersants. The study was focused on the effect of block structure on the performance of the dispersant. The use of dispersant affects stability, zeta potential and the rheology of the suspensions. The key variable in the dispersants was the molar ratio of the chains. The optimum molar ratio of the dispersant is observed to be 1 for the best results. A larger dispersion stability is obtained due to large adsorption on the coal surface.

The above literature clearly shows that small dosage of dispersants can be favorable for the improvement in the stability and rheology of the coal water slurry. The literature review on rheological characteristics of coal water slurry is presented in Table 2.1

Table 2.1 Summary of previous research on the rheological characteristics of coal water slurry with and without additives

Ref. (Year)	Investigation	Coal type	Dispersant (if any)	Key result(s)
[101] (2000)	Rheological characteristics of low rank coal	Low-rank; British	Non-ionic; Triton X-405	• High ash makes the slurry unpumpable • Demineralization is the key for solution to make coal water pumpable • The non-ionic dispersant fails to work in high ash coal.
[102] (2000)	Preparation of coal water slurry using ultrasonic irradiation	Low-ash; Chinese	Anionic; NSF	• Ultrasonication leads to generation of fines • Ultrasonication significantly increases the concentration of cations in the slurry
[103] (2002)	Rheological characteristics of concentrated coal water slurry	High ash; Indian coal	None	• Due to high ash, the coal shows a hydrophilic nature • The apparent viscosity increases with solid concentration and decreases with shear rate.
[104] (2004)	Highly concentrated coal-water slurry preparation using dispersants	Low ash; Indian coal	Anionic; Naphthalene based and Naphthalene-toluene based	• Small dosage of dispersants can help in the production of highly concentrated coal-water slurry • Wet grinding can easily produce fine particles of 21-22 μm

Ref. (Year)	Investigation	Coal type	Dispersant (if any)	Key result(s)
[105] (2006)	Effect of molecular weight of dispersant on the effectiveness of the dispersant	Low ash; Chinese coal	Anionic; Lignosulphonate	The dispersant having molecular weight of 10000 to 50000 is observed to have the best performance.
[106] (2008)	Comparison of lignin based and naphthalene-based dispersants	Low ash; Chinese coal	Anionic; FZN and GCL3S	The lignin based dispersant shave better performance and stability.
[107] (2009)	Rheology of coal sludge slurry	High ash; Chinese coal	Anionic; Sodium naphthalene sulfonate formaldehyde	- The viscosity of coal sludge slurry is significantly higher than the coal water slurry - The maximum loading of coal sludge is increased to 61% using a dispersant
[108] (2010)	Effect of sewage sludge on the rheology of coal water slurry	Low ash; Chinese coal	Sodium naphthalene sulfonate formaldehyde	- Yield stress is observed to increase with addition of sewage sludge. - The rheological characteristics are improved using the synthetic dispersant.
[109] (2011)	Effect of coated coal surface on the stability of coal water slurry	High Ash; Chinese coal	Span-40, Span-60, 0 diesel and CTAB	- Span-40 outperforms all other dispersants - The optimum dosage is just 0.3% by mass
[110] (2012)	Effect of particle size distribution on the rheological characteristics of coal water slurry	Low ash; Indonesian coal	Naphthalene Sulfonate formaldehyde	The particle size distribution significantly affects the rheological characteristics of coal water slurry.

Ref. (Year)	Investigation	Coal type	Dispersant (if any)	Key result(s)
[111] (2014)	Rheological characteristics of Powder River Basin coal	American River Basin coal	Polystyrene sulphonate and Triton X-100	• The bi-modal samples have better slurryability and rheology as compared to the mono-modal samples. • The non-ionic dispersant was a better performer. • The dispersant fails at high concentration of coal.
[112] (2015)	Estimation of packing density of coal water slurry	Medium and High ash; Chinese coals	None	• The packing density of multimodal suspensions is higher than the unimodal suspension
[113] (2016)	Effect of moisture pre-absorption on the stability of coal water slurry.	High ash; Chinese coal, activated carbon and coke	Methyl naphthalene sulfonate formaldehyde	• The pre-absorption was helpful in the improvement of the stability of the coal water slurry • The activated carbon was observed to absorb the highest amount of moisture as compared to the lignite and coke
[114] (2018)	Development of 3-D structured dispersants for coal water slurry	Low ash; Chinese coals	TIA-1, TIA-2 and TIA-3	• TIA-1 has the best performance due to 3-D structure and large reticulation • Very small dosage of dispersant is sufficient.

Ref. (Year)	Investigation	Coal type	Dispersant (if any)	Key result(s)
[115] (2019)	Effect of lignosulfonate on the surface characteristics of coal particle	Low ash; Canadian coal	Anionic; Lignosulfonate	• The lignosulfonate has better performance due to greater degree of adsorption on the particles' surface. • The increased steric and electrostatic repulsions lead to improved stability.
[116] (2019)	Rheology of coal water slurry with a small dosage of alkaline additives	Low ash; Chinese coals	K ₃ PO ₄ , Na ₃ PO ₄ , Na ₂ CO ₃ , NH ₃ .H ₂ O, NaOH	• Low dosage improved the stability and the rheology • High dosage deteriorated the characteristics of the slurry
[117] (2020)	Influence of TiO ₂ nanoparticles on the combustion characteristics of coal water slurry	High ash	TiO ₂ Nanoparticles	• The nanoparticles increased the activation potential of the fuel. • The optimum concentration of the nano particles was 2.11%
[118] (2020)	Performance characteristics of block type poly carboxylate	Medium ash; Chinese coal	Anionic; Block polycarboxylate	• The optimum molar ratio of the dispersant is observed to be 1 for the best results. • A larger dispersion stability is obtained due to large adsorption on the coal surface.

2.3 Literature review on the thermal treatment of coal:

The coal water slurries are nowadays proving their application in diesel/crude oil-fired furnaces and heavy diesel engines [119]. But, the lignites show poor slurryability and rheological characteristics, due to the presence of high inherent moisture and well-developed pore structure [75]. In this regard, solutions such as air drying [120–122], thermal heating [123–125], mechanical thermal expression [126,127], hydrothermal treatment [128–134] and the fastest of all, microwave heating/treatment [135,136] have been proposed by different researchers. The air drying of the lignites is a simple technique and involves no or very less skill. The results of air drying are moderate and can improve the physical properties of the lignites up to a certain limit but have a merit of involving the least resources and capital costs. The thermal upgrading processes such as thermal heating, mechanical thermal expression and hydrothermal treatment can produce significantly upgraded lignites. The upgraded lignites vary slightly in terms of physical and chemical properties depending upon the upgrading process.

Ateşok et al. (2000) [137] conducted flotation study on a low rank coal. Using the conventional method, the recovery of the coal was very small. So, the thermal treatment of the coal was proposed, where the coal is first subjected to a thermal heating at 300-600°C followed by grinding. The thermal heating changed the surface morphology of the coal and was helpful in the removal of hydrophilic functional groups. The thermal heating made the coal hydrophobic as result the flotation efficiency of the coal was improved.

Serranno et al. (2004) [138] conducted characterization studies on some low rank coals, where they found that these coals contained significant amount ash. Further investigation showed that the micro porosity and the surface area is small in all the coal samples. A series of experiments were further conducted to understand the effect of thermal treatment and demineralization on the surface properties of coal. The results revealed that the demineralization and the thermal treatment led to increased porosity of the particles.

Nakagawa et al. (2004) [139] treated an Australian brown coal using the hydrothermal treatment process. They provided an alternative way to eliminate the issue of generation of waste water from the hydrothermal treatment process. The hydrothermally treated coal has better slurryability and stability. The treatment also leads to the increase in the calorific value of the coal. The waste water from the process was subjected to gasification, where the hydrocarbon content of the coal was successfully converted in combustible gases.

Lester et al. (2006) [140] studied the potential of microwaves with a perspective of preparation of coke from a low rank U.K. coal. The coal particles of -3mm were subjected to microwaves of 8kW of power for 8, 25 and 70 minutes of exposure time. The optical texture of the resultant coke was quite similar to that of the commercially available coke. The authors indicated the requirement of further studies in the matter.

Mi et al. (2007) [141] conducted ultrasonic and microwave treatment of coal in the presence of tetrachloroethylene. The combination of both the processes was beneficial for the higher removal rate of sulfur. Due to the removal of sulfur, there was a slight reduction in the calorific value of the treated coal. The removal of the sulfur was dependent upon the form in which the sulfur is existing.

Karthikeyan et al. (2008) [142] in their study provided alternate ways to minimize the moisture re-adsorption in the thermally treated/dried coal. The study proposes to use a coating of bitumen on the coal surface to choke the pores. This eliminates the chance of re-adsorption of moisture. This study also highlights the benefit of using hot dry coal for drying the wet coal.

Çınar et al. (2009) [143] investigated the flotation behavior of high ash coals. The experiments included the thermal treatment of coal at a low temperature of 105°C. The different characterization studies on the untreated and the treated coals indicated the improvement in the surface characteristics of the coal after thermal treatment. The hydrophobicity of the coal was improved as a result of thermal treatment of the coal. The net sulphur content in the coal was reduced from 2.5% to 1.52%

Ochoa et al. (2010) [144] conducted studies on Argentinean low rank coals. The physico-chemical characteristics of coal were studied as the part of their study. The coal was washed with acids to remove the ash. The demineralized coal was then subject to thermal treatment at 1000°C in nitrogen rich atmosphere for an hour. The demineralization followed by thermal treatment leads to the improvement in the physicochemical characteristics of the coal. The treatment process destroys the mesopores, while increases the number of micropores.

Jorjani et al. (2011) [145] performed experiments on Iranian coals of medium ash content, with an aim of producing ultra clean coal. The combination of microwave treatment and acid leaching was used to upgrade the coal. The ash content in the treated coals was reduced from 22 to 76%. The fine particles were more responsive to ash removal. The final ash content in the treated coal was as low as 0.39%.

Inoue et al. (2012) [146] studied the effect of hydrothermal treatment on the physico-chemical and rheological characteristics of coal-solvent slurry. Hydrothermal treatment was carried out at 200 to 350°C temperature. The hydrothermal treatment led to a significant removal of carboxylic and hydroxyl groups. The components of mineral matter viz. sodium, chlorine, calcium and magnesium were also removed. The resultant HTT coal-solvent slurry had viscosity 0.1 times of the untreated coal-solvent slurry.

Ge et al. (2013) [147] studied the influence of microwaves on the physico-chemical characteristics of coal. The results indicate a significant drop in the inherent moisture content. The microwave irradiation generated new pores on the surface of coal. Upgraded coal had better ignition characteristics and high calorific value than the untreated coal. The upgraded coal exhibited weak combustion characteristics, eliminating the risk of spontaneous ignition.

Fu et al. (2014) [148] performed a comparative study on the effect of thermal and hydrothermal treatment processes on the physico-chemical and rheological characteristics of coal and coal water slurry. The thermal treatment was performed at temperatures ranging from 100 to 700°C and 150 to 300°C for hydrothermal treatment process. The results obtained using the hydrothermal treatment are observed to be better than the thermal treatment. The hydrothermal treatment successfully removes greater proportion of oxygen functional groups as compared to the thermal treatment.

Mesroghli et al. (2015) [149] studied the effect of microwave irradiation on the structural changes in coal and sulfur content in the coal. The microwave irradiation resulted in the degradation of the matrix structure of the coal. This also resulted in phase changes in the mineral matter of the sample. The microwaves resulted into the removal of the sulfur content of the coal.

Wei et al. (2016) [150] conducted experiments on Chinese lignite to study its slurryability. The lignite was also subjected to hydrogasification to convert it into char rich in carbon. The residual char also had poor slurryability due to the presence of micropores and capillary structure on the surface. The two samples were blended and the resultant sample had better slurryability.

Shi et al. (2017) [151] conducted experiments on a Chinese low rank coal to upgrade it using hydrothermal treatment process. The sample was treated in a lab scale autoclave at a temperature of 200° to 350°C for 30 minutes. The structural and physicochemical changes in

the treated coal were investigated using FTIR and thermogravimetry experiments. The results indicated a significant elimination of oxygen and volatile matter in the treated coal along with CO₂ and CO gases. The FTIR investigation revealed the loss of oxygen containing functional groups that include carboxyl, carbonyl and oxygenated aliphatic carbons.

Wei et al. (2017) [152] performed the experiments on hydrothermal treatment of Chinese lignite and char obtained from hydro-gasification. Both the samples are blended and subjected to hydrothermal treatment in the lab scale autoclave at a temperature of 150° to 320 °C with the residence time of 30 minutes. The effect of hydrothermal treatment on the physicochemical characteristics of were also investigated. The results indicated an appreciable improvement in the slurryability of the blended sample up to temperature of 300°C. However insignificant change in physicochemical characteristics of the blended sample is observed when the temperature is increased beyond 300°C.

Hong et al. (2018) [153] performed the experiments on the microwave treatment of high ash coal. The study was based on analysis of effect of microwaves on the pore structure of coal. The coal was treated to temperatures ranging from 25-300°C and microwave input power of 1-6kW. The results indicated a rise in the pore volume upon irradiation with microwaves. The study presented a proposal to be applied for the degassing of the coal seams.

Ullah et al. (2020) [154] treated a low rank coal using hydrothermal treatment process. The aim of their study was to migrate the selenium content in the coal. The results indicated an improvement in the physico-chemical characteristics of the coal. The selenium content in coal was observed to fall after hydrothermal treatment at 300°C. The hydrothermal treatment also leads to the densification of the structure of the coal.

Table 2.2 Summary of literature review on various treatment processes for upgradation of coal and coal water slurry

Ref. (Year)	Investigation	Coal type	Type of treatment	Key result(s)
[137] 2000	Flotation characteristics of low rank coal	High ash coal	Conventional heating (300-600)	• The thermal heating improved the grindability of the coal • The thermal treatment improved the flotation efficiency of coal
[138] 2004	Characterization studies on low rank coals	High ash coal	Conventional heating	• Demineralization and the thermal treatment led to increased porosity of the particles
[139] 2004	Hydrothermal treatment of coal and gasification of waste water	High coal	Hydrothermal treatment	• Demineralization and removal of acidic functional groups.
[140] 2006	Coke production using microwaves	Low rank coal	Microwave treatment	• The resultant coke had a similar optical texture to that of the commercially available coke. • The coke shows anisotropic properties.
[141] 2007	Ultrasonic and microwave treatment of coal in the presence of tetrachloroethylene.	Low rank coal	Ultrasonic and Microwave treatment	• The combination of both the processes was beneficial for the higher removal rate of sulfur. • The removal of the sulfur was dependent upon the form in which the sulfur is existing.
[142] 2008	Alternate ways to minimize the moisture re-adsorption in the thermally treated/dried coal	Low rank coal	Conventional heating followed by coating with bitumen	• The coating of bitumen on the coal surface to choke the pores. • Hot dry coal was used for drying the wet coal.

Ref. (Year)	Investigation	Coal type	Type of treatment	Key result(s)
[143] 2009	Effect of thermal treatment on flotation behaviour of high ash coals	High ash coal	Low temperature heating	• The hydrophobicity of the coal was improved as a result of thermal treatment of the coal. • The net sulphur content in the coal was reduced from 2.5% to 1.52%
[144] 2010	Effect of demineralization followed by thermal treatment on the physico-chemical characteristics of coal.	Low rank coal	Demineralization and thermal treatment	• The combined treatment process helps in the removal of ash and mesopores • The number of micropores are observed to increase.
[145] 2011	Microwave treatment of coal followed by acid leaching	Medium ash coal	Microwave treatment and demineralization	• 22 to 76% reduction in the ash content. • Final ash content was as low as 0.39%
[146] 2012	Effect of hydrothermal treatment on the physico-chemical and rheological characteristics of coal-solvent slurry	Lignite	Hydrothermal treatment	• Hydrothermal treatment leads to the removal of acidic functional groups and some fractions of mineral matter.
[147] 2013	Microwave irradiation of low rank coal	Low rank coal	Microwave treatment	• Microwave irradiation generates new pores. • The calorific value of the upgraded coal increases upon microwave irradiation.
[148] 2014	Effect of hydrothermal treatment on slurryability of the coal	Low rank coal	Thermal and Hydrothermal treatment	• The hydrothermal treatment was more effective than the thermal treatment.
[149] 2015	Influence of microwaves on the structure of the coal	Low rank coal	Microwave treatment	• The microwaves result in the degradation of the coal matrix

Ref. (Year)	Investigation	Coal type	Type of treatment	Key result(s)
[150] 2016	Hydrogasification of lignite	Lignite	Hydrogasification	• The treatment process results into desulfurization. • The lignite and its hydrogasified char have poor slurryability and stability. • The blended sample leads to improvement in the slurryability and stability of the coal.
[151] 2017	Hydrothermal treatment of low rank coal	Low rank coal	Hydrothermal treatment	• Leads to elimination of oxygen functional groups • The elemental oxygen and volatile matter is released
[152] 2017	Hydrothermal treatment of lignite	Lignite and char	Hydrothermal treatment	• Appreciable improvement in the physicochemical characteristics and slurryability of the coal sample
[153] 2018	Effect of microwave time and power on the pore structure of coal	Medium ash coal	Microwave treatment	• The microwaves increase the pore volume.
[154] 2020	Effect of hydrothermal treatment on the Selenium content in the coal	Lignite	Hydrothermal treatment	• Hydrothermal treatment leads to the elimination of pores of the coal • The selenium content in the coal is observed to fall.

2.4 Literature review on the experimental and numerical analysis of solid-liquid flow:

The flow characteristics of solid-liquid slurry through the horizontal pipelines and pipe bends has been extensively reported in the previous literature. Some of the previous studies include the simulation of flow a coal water suspension through a horizontal pipe by [155], where the pressure drop characteristics of the pipeline have been reported. The researchers used various turbulence models; the SST $k-\omega$ model is found to be the most accurate of the all [156]. Similar studies were also reported by [157–159].

Lu and Zhang (2002) [160] Studied the frictional characteristics of coal water paste through a horizontal pipe. The experiments were based on the flow characteristics through pipes of four different diameters viz. 62, 40, 32, and 25 mm. The concentration of water in the slurry was varied in the range of 22.1 to 33.5% by mass. The experimental results indicated that the headloss increases with influx velocity. The energy loss coefficient in the case of laminar flow showed similar characteristics as shown by a Newtonian fluid.

Ling et al. (2003) [161] followed the CFD approach for the simulation of flow of solids laden liquid through a horizontal pipe. The authors used RNG $k-\epsilon$ turbulence model along with the algebraic slip mixture model. Using these methods, the mean pressure gradient of the flow across the pipeline was investigated and compared but the previously published experimental data. The predicted results indicated a good resonance of the model with the previously published experimental data. Microscopic properties of the flow such as the solids' distribution across the flow domain, density of the slurry, mean velocity distribution of the slurry and the skin friction of the flow were also evaluated using the validated model.

Guohua et al. (2007) [162] performed experimental and CFD studies on the flow characteristics of coal water slurry. The rheology data obtained from the bench scale test was used as a base for the CFD model used in fluent software. In their study the effect of pipe diameter on the slip velocity was investigated. The results indicated a weak correlation between the slip velocity and the pipe diameter. The CFD model was corrected with the slip correction and the new results were quite close to the experimental results.

Liu et al. (2009) [163] conducted experiments to investigate the headloss across three common pipe fittings. These fittings were a gradual contraction, a 90° bend and a sudden contraction. The key variation in the gradually contracting geometry was the angle of contraction. The results indicated that on increasing the angle of contraction, the headloss initially starts decreasing, attains a minima and then again starts increasing. Similarly, for the

90° bend, an optimum diameter ratio was obtained. In the case of sudden contraction, with the increase in the pipe diameter ratio, the pressure loss across the fitting was observed to increase.

Chen et al. (2009) [164] used the CFD approach to predict the flow characteristics of coal water slurry through a horizontal pipe. In their study, the Eulerian multiphase model and RNG- $k-\epsilon$ turbulence modelling scheme was used. In their study the coal particles of bimodal distribution were used. The model was validated with the existing literature. In their study the influence of grain composition, influx concentration, and influx velocity was investigated. The results indicated that the difference of specific gravity of small and large particles has a significant effect on the concentration distribution and velocity distribution across the pipeline.

Chen et al. (2010) [165] investigated the flow behavior of coal water slurry in the regions close to the wall. The pipe of diameter 50, 40, 32, and 25 mm were selected for the experimental investigation. The results indicated a reduction in the wall slip velocity with increase in the particle size, temperature and solid concentration. The wall slip mainly depended on the critical shear and yield stress at low shear rates. However, it was observed to be dependent on shear viscosity at high shear rates.

Antaya et al. (2013) [166] carried out numerical simulation of flow characteristics of flow characteristics of concentrated slurry in pipeline of diameter 0.15 meters and 0.0515 meter using CFD code ANSYS CFX. The particles of both mono-model and bi-model particle size distribution were used. The model was used for the investigation of frictional pressure gradient, solids distribution, and particle velocity. The choice turbulence models greatly affected the prediction of the results.

Hashemi et al. (2014) [167] used the experimental approach to see the effect of solid concentration and influx velocity on the fluctuation of particles velocity and concentration fluctuations. In order to see the in-situ concentration at any perpendicular plane inside the pipeline the EIT was used. The results indicated that the fluctuations are more in the regions close to the wall that are observed to increase with the mixture velocity. They further found that the reason for the fluctuation in the solids distribution across the floor domain was mainly due to turbulence interactions of particles and the fluid.

Messa et al. (2014) [168] developed a new robust CFD model for the prediction of solid liquid flow through the straight pipes and bends. The mechanisms such as interface friction turbulence dispersion that govern the solid liquid flow were modeled in a different way. The

model was first compared to the experimental data published elsewhere and then was used to predict the flow of the slurry having volumetric concentration ranging from 9 to 40% an influx velocity in the range of 1 to 5.5 m/s. The pipe diameters in the model were 50 and 160mm.

Wu et al. (2015) [169] conducted experiments for the flow of coal gangue-fly ash through a pipe loop. The experimental results were used for validation purposes of a numerical model. The numerical model was based on COMSOL Multiphysics. This model was validated with the experimental results of the pipe loop. The results produced by the validated model were in a good agreement to the experimental results. This model was further used to investigate they effect of pumping pressure and influx velocity along with the pipe diameter on the pressure gradient across the pipeline.

Messa et al. (2015) [170] used the CFD approach for the modeling of fully suspended slurry flow inside the horizontal pipeline. The study was based on the modification of already existing CFD model for the enhancement of flow prediction by the model. The main focus of the modification was on the improvement in the flow prediction near the wall. A decisive modeling approach was used near the wall where if the particle size is smaller than the boundary layer thickness then Launder and Spalding model was used else Ferre and Shook approach was used. The results predicted by the modified model were better than the already existing model.

Li et al. (2018) [171] conducted a 3-D steady state simulation of solid-liquid flow. The particles of sizes 0.125mm and 0.44mm are selected for the present analysis. The influx concentration is kept in the range of 20-50% and flow velocity is varied in the range of 2-5 m/s. The results indicate a significant difference in the flow characteristics of fine and coarse particles. The results are observed to deviate slightly at higher solids concentrations.

Sultan et al. (2019) [172] used a CFD model to simulate the flow of three phase system through an annular geometry. The commercial CFD code ANSYS Fluent was used. The flow domain was meshed by adding inflation layers, 10 in number and with a growth factor of 1.2. The results predicted by the model were in a good agreement to the experimental results. The predicted results were valid for a wide range of particles and parameters.

Matoušek et al. (2019) [173] conducted experiments on a variable inclination pilot test loop for the investigation of effect of inclination on the distribution of solids across the flow domain. The sand of median diameter of 0.87mm size allowed to flow through a pipe of internal

diameter of 100mm. The Wilson-Tse approach, that is popularly used for the prediction of effect of pipe inclination on the settling velocity, fails to predict the same for partially stratified flows. To generalize, the curves for settling slurries of d/D ratio of <0.04 were added, that otherwise were over predicted by the Wilson-Tse approach.

Zheng et al. (2019) [174] developed a general DNS based CFD model for the prediction of flow of shear thinning fluids through pipes. The accuracy of the DNS for the flow of non-Newtonian and Newtonian fluids is estimated by comparing the results with the experimental results. For this purpose, Sementex, a DNS code was used. The maximum difference in the results was less than 16% in all the cases. Their research described the possible application of OpenFOAM for the prediction solids' laden flows.

Zheng et al. (2020) [175] used the DNS-DEM model to investigate the flow of coarse particles through a pipeline. The flow predicted by the model was in good agreement to the experimental data. The deviation in the prediction of the concentration profile was less than 10%, while the maximum deviation from the experimental results existed at the pipe centre. Their study recommended the usage of DNS-DEM modelling technique for the such predictions of the flow.

The literature review on the experimental and numerical analysis of solid liquid flow through horizontal pipes and pipe fittings is most reported on the hydrophilic materials such as sand, coal ash and mineral tailings. The CFD approach is a versatile technique to virtualize almost any case of fluid flow. Previously, the CFD approach has been effectively used to predict the flow through different pipes and pipe fittings and using different solid materials.

Table 2.3 Summary of literature review on experimental and numerical analysis of solid-liquid flow

Ref. (Year)	Investigation	Material	Analysis type	Key result(s)
[160] 2002	Flow characteristics of coal water paste	Coal	Experimental investigation	• Headloss increases with influx velocity • The energy loss coefficient was similar to that of Newtonian fluid.
[161] 2003	Flow characteristics of solids' laden flow	--	CFD simulation	• The model was in a good agreement to the experimental results
[162] 2007	Flow characteristics of coal water slurry	Coal	CFD simulation	• The slip velocity was weakly associated to pipe diameter
[163] 2009	Flow of coal water slurry through pipe fittings	Coal	Experimental investigation	• The headloss decreases with contraction angle, attains a minima and again increases with angle • The headloss increases with pipe diameter ratio
[164] 2009	Prediction of flow of coal water slurry	Coal	CFD Simulation	• The results indicated that the difference of specific gravity of small and large particles has a significant effect on the concentration distribution and velocity distribution across the pipeline.
[165] 2010	Flow prediction in the near wall regions	Coal	Experimental investigation	• The wall slip mainly depended on the critical shear and yield stress at low shear rates. • The results indicated a reduction in the wall slip velocity with increase in the particle size, temperature and solid concentration.

Ref. (Year)	Investigation	Material	Analysis type	Key result(s)
[166] 2013	Flow characteristics of highly concentrated slurry	--	CFD Simulation	• The model was used for the investigation of frictional pressure gradient, solids distribution, and particle velocity. • The choice turbulence models greatly affected the prediction of the results.
[167] 2014	Flow characteristics of solid liquid flow	Sand	Experimental investigation	• The results indicated act the fluctuations one more in the regions close to the wall that are observed to increase with the mixture velocity
[168] 2014	Prediction of solid liquid flow through the straight pipes and bends	Sand	CFD Simulation	• Model was used to predict the flow characteristics through pipe and pipe bends.
[169] 2015	Prediction of flow characteristics through pipe loop	Coal gangue-fly ash	CFD Study	• The results produced by the validated model were in a good agreement to the experimental results. • This model was further used to investigate they effect of pumping pressure and influx velocity along with the pipe diameter on the pressure gradient across the pipeline
[170] 2015	Modelling of fully suspended slurry flow	Sand	CFD study	• The results predicted by the modified model were better than the already existing model.

Ref. (Year)	Investigation	Material	Analysis type	Key result(s)
[171] 2018	3-D steady state simulation of solid-liquid flow	Sand	CFD study	- The results indicate a significant difference in the flow characteristics of fine and coarse particles. - The results are observed to deviate slightly at higher solids concentrations
[172] 2019	Flow of three phase system through an annular geometry	Air, water and sand	CFD	- The results predicted by the model were in a good agreement to the experimental results. - The predicted results were valid for a wide range of particles and parameters.
[173] 2019	Effect of inclination of pipe loop on the flow of partially stratified	Sand	Experimental investigation	To generalize, the curves for settling slurries of d/D ratio of <0.04 were added, that otherwise were over predicted by the Wilson-Tse approach.
[174] 2019	Prediction of flow of shear thinning fluids through pipes	--	CFD based on DNS	- The maximum difference in the results was less than 16% in all the cases. - Their research described the possible application of OpenFOAM for the prediction solids' laden flows.
[175] 2020	DNS-DEM model to investigate the flow of coarse particles through a pipeline	--	CFD based on DNS-DEM	- The deviation in the prediction of the concentration profile was less than 10% - The maximum deviation from the experimental results existed at the pipe center

2.5 Gaps in literature:

Based on the literature review, the following gaps in existing literature have been identified:

1. The physico-chemical characteristics of coal vary significantly with topography of its origin. The previous studies on the physico-chemical characteristics of Indian coals indicate that these coals are quite different from those found elsewhere. The Indian coals are mostly Sub-bituminous and lignites, that have poor slurryability. The literature review on shows that the effect of particle size distribution on the rheological characteristics of Indian high ash coal is least reported. The literature on the effect of hydrothermal and microwave treatment of coal on the physico-chemical and rheological characteristics is very limited.
2. The flow characteristics of Indian high ash coal-water slurry of variable particle size distribution is not well established.
3. The pipe fittings are an integral part of the slurry pipeline. The flow of coal water slurry through different pipe fittings viz. elbows, converging section, and diverging section are least reported.

Chapter 3

Physico-chemical and morphological characterization of coal

3.1 Introduction:

The coal water slurry has well-established applications for hydrotransportation and energy production. The coal water slurry, is either prepared using pre-pulverized dry coal or wet milling of the coal to a final size of $<150\ \mu\text{m}$. This slurry is then transported to the power stations, located at distance ranging from few kilometers to few 100 kilometers. For an efficient design of pipeline, properties of coal need investigation. The rheological characteristics of coal water slurry at low and high concentrations are dependent on different properties of coal such as particle size distribution, specific gravity, settling characteristics, proximate and ultimate analysis.

The particle size distribution, specific gravity and settling characteristics describe the stability and rheological characteristics of the slurry during transportation or for storage purposes. Chemical properties describe the hydrophobic or hydrophilic nature of the coal. The investigation of all these characteristics is done by as per the following methodology:

3.2 Sample preparation

3.2.1 *Sample collection:*

Coal sample is collected from Guru Nanak Dev thermal power plant, Bathinda, Punjab, India. The source of this coal is Makum coal fields of Assam. Post sampling, the coal is subjected to two different experimental routes. The detailed engineering flow charts followed are presented in Figures 1.4 and 1.5.

3.2.2 *Crushing and pulverization:*

The chunks of coal are initially crushed in a jaw crusher and reduced to a maximum size of 7 cm. The fines as well as the crusher particles are then fed into a lab scale ball mill. Nine steel balls, each having a mass of 0.5 kg are used to pulverize the crushed coal. The further segregation of the particles is done using a sieve shaker that uses British standard sieves.

3.2.3 Experimental route-I: (Characterization of untreated and treated coals)

3.2.3.1 Hydrothermal treatment:

The crushed coal is sieved through a 200-mesh sieve in order to remove the particles greater than 75 μm . The sieved coal samples are hydrothermally treated in a 15 l autoclave, which can be operated at a maximum temperature of 394 K. The sieved sample is weighed on a lab scale weighing balance (least count 0.001g) and 200g sample is taken in a hardened glass container, that can easily withstand temperatures $< 300^\circ\text{C}$. The weighed coal sample is suspended in 200 ml of deionized water and mixed slowly for 13–15 minutes in order to ensure that each coal particle is surrounded by water molecules. The coal water suspension is placed in the autoclave and heated at a constant rate of 10 K/minute to 394 K. The coal samples are prepared and treated as per above-mentioned method but at different residence times, viz., 30, 45, 75, 120 minutes and named as HTT30, HTT45, HTT75 and HTT120, respectively. After the hydrothermal treatment, the samples are dried in an oven to remove excess moisture and are kept in an open atmosphere for 72 hours, so that equilibrium moisture levels are achieved.

3.2.3.2 Microwave treatment:

The pulverized coal is sieved through standard sieves to separate different sized particles. In the present research, the particles that pass through the BS 200 mesh sieve are taken and these particles have size of $< 75 \mu\text{m}$. The untreated coal (each sample weighing 100g) is treated in a microwave oven clocked at 2450 MHz and having variable power output ranging from 180–900 W. The sieved sample is weighed on a lab scale weighing balance (least count 0.001g) and 100g sample is taken in a hardened glass container, that can easily withstand temperatures $< 300^\circ\text{C}$. The sample is then placed inside the ventilated heating cavity of the MW (microwave) heater. The MW heating process has two variables viz. the heating power and the heating time, together these comprise the microwave energy released by the coil. So, in the present cases, the heating power is fixed at 720 W but the MW treatment time is varied as 30, 60, 90 and 120s. The heated sample is then allowed to cool gradually and stored under equilibrium conditions for at least a week before performing any kind of test on it. The samples treated for 30, 60, 90 and 120 seconds are named as MWT30, MWT60, MWT90 and MWT120 respectively.

3.2.4 Experimental route-II: (Characterization of blends of untreated and treated coals)

The fine particles of $< 53 \mu\text{m}$ size have a variety of application such as entrained flow gasification, thermal furnaces, low speed marine diesel engines, etc. These sophisticated devices use nozzles to spray the coal water slurry fuel to be burnt. However, the previous

studies report that the viscosity of the coal water slurry increases with decrease in the particles size. As a result, the -53 μm particulate slurry is expected to have very high viscosity. This high viscosity may lead to substantially high pressure losses in the pipeline transportation systems. So, this part of the study is aimed to improve the physico-chemical and flow characteristics of the fine (-53 μm sized) particulate slurry.

The hydrothermal and microwave treatment processes significantly improve the physico-chemical and flow characteristics of the coal water slurry. However, given the energy required to thermally upgrade the coal, the upgraded coals appear to be uneconomical to be fired in the powerplants. As a solution to this issue, experimental route-II is also proposed, where the untreated as well as the treated/upgraded coal will serve as an additive to improve the physico-chemical and flow characteristics of the existing -53 μm size coal. Out of the two thermal upgrading processes viz. hydrothermal treatment and microwave treatment, the latter one is opted for experimental route-II due to the following reasons:

- The heating process in microwaves is volumetric i.e. the microwaves can penetrate deep into the particle in a very short span of time. So, the MWT process is faster than the HTT by many folds.
- The MWT produces better results than HTT in a considerably low time.

So, the following coals serve as additives:

- i. Bigger coal particles as additive for the fine (-53 μm) particulate slurry.
- ii. Coal treated with microwaves.

3.2.4.1 Coarse coal particles as additive for the fine (-53 μm) particulate slurry:

The sample ground in the ball mill has a maximum size $> 106 \mu\text{m}$. The sample is sieved using a standard sieve of 140 mesh to remove the particles greater than $106 \mu\text{m}$ size. The sieved sample is further distributed in the size range of -53 μm , -75+53 μm and -106+75 μm using standard sieves of 270, 200 and 140 mesh respectively. The finally obtained coal samples are named as S₁, S₂ & S₃ respectively. Four bi-modal coal samples are prepared by mixing 10, 20, 30 & 40% (by mass) of -75+53 μm sized coal particles in -53 μm sized fine particles. These samples are named as S₄, S₅, S₆ & S₇ respectively. Similarly, four bi-modal coal samples are prepared by mixing 10, 20, 30 & 40% (by mass) of -106+75 μm sized coal particles in -53 μm sized fine particles. These samples are named as S₈, S₉, S₁₀ & S₁₁.

Table 3.1 represents the distribution of fine and coarse particles in the different samples; S₁-S₁₁.

Table 3.1 Distribution of fine and coarse particles in different mono-modal and bi-modal samples.

Sample nomenclature	Fine (-53 μm) %	Coarse (-75+53 μm) %	Coarse (-106+75 μm) %
S ₁	100	0	0
S ₂	0	100	0
S ₃	0	0	100
S ₄	90	10	0
S ₅	80	20	0
S ₆	70	30	0
S ₇	60	40	0
S ₈	90	0	10
S ₉	80	0	20
S ₁₀	70	0	30
S ₁₁	60	0	40

3.2.4.2 Microwave treated coal as additive:

The crushed sample of coal is sieved through BS 270 sieve to remove the particles greater than 50 μm . The sieved sample is weighed on a lab scale weighing balance (least count 0.001g) and 100g sample is taken in a hardened glass container, that can easily withstand temperatures $< 300^\circ\text{C}$. The sample is then placed inside the ventilated heating cavity of the MW (microwave) heater. The preliminary results depict that the coal properties show variation up to certain values of time and input power and these values are 120 seconds and 720 W respectively. Post these values of treatment time and input power, almost insignificant changes in the physical and chemical properties of coal are observed. So, the coal sample is treated for

120 seconds at an input power of 720 W. This coal sample is then blended with the untreated coal in different proportions. The treated coal sample is blended with the untreated coal sample according to the distribution scheme mentioned in Table 3.2.

Table 3.2 The distribution of untreated and treated samples for the preparation of coal samples

Sample nomenclature	Untreated coal (%)	Treated coal (%)
Untreated	100	0
MWT10%	90	10
MWT20%	80	20
MWT30%	70	30
MWT40%	60	40
MWT50%	50	50

3.3 Physical characterization

The physical characteristics of the untreated and treated coals are analysed using the bench scale tests. The following parameters are studied to analyse the physical characteristics of the all the coal samples:

- i. Particle size distribution (PSD)
- ii. Specific gravity
- iii. Settling characteristics

3.3.1. Particle size distribution (PSD):

The particle size distribution of the raw and the treated coal samples are studied using Malvern 3601 particle size analyzer. It works on the principle of laser diffraction and measure the volume occupied the particles smaller than a given size. The crushed and pulverized coal received from the ball mill is further divided into two parts viz. CS-I for experimental route-I and CS-II for experimental route-II. As the sample CS-I is sieved through BS 200 mesh, so the maximum particle size in CS-I is expected to be less than 75 μm . The CS-II is used for studying

the effect of blends of coarse and treated (fine) particles on various characteristics of fine particles. So, the particles of $-106\ \mu\text{m}$ are considered for CS-II.

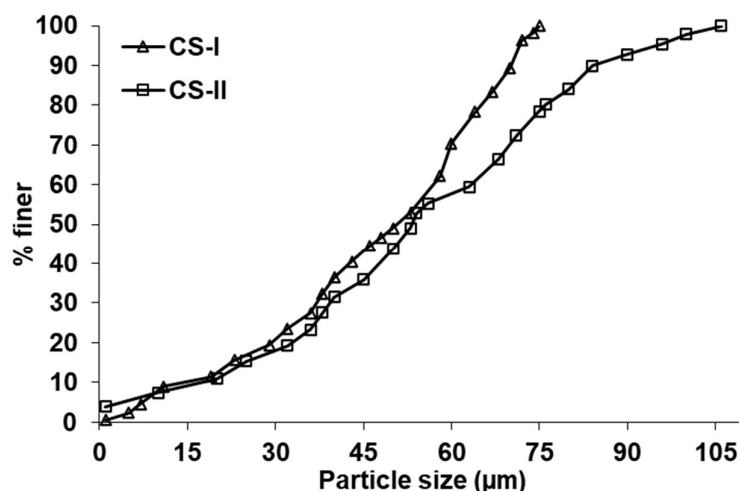


Figure 3.1 Cumulative particle size distribution of coals CS-I and CS-II

The particle size distribution of the coals CS-I and CS-II are presented in Figure 3.1. The median diameter for coal CS-I is around $51\ \mu\text{m}$ and 53.1% particles have size less than $53\ \mu\text{m}$. Around 70 and 90% particles are finer than $60\ \mu\text{m}$ and $71\ \mu\text{m}$ respectively. The maximum particle size for sample CS-II is $106\ \mu\text{m}$ as the bigger particles are sieved out from the sample. Around 35% particles of coal CS-II are finer than $45\ \mu\text{m}$. The median diameter of coal CS-II is $53.25\ \mu\text{m}$. Around 80 and 95% particles are finer than 76 and $96\ \mu\text{m}$. The PSD of lower (finer) half of coal CS-II is quite comparable with that of coal CS-I.

The particle size distribution of the hydrothermally treated (HTT) coal at various residence times is presented in Figure 3.2. The HTT coals show an increase in the fraction of fines in the sample. The HTT30 sample shows a significant increase in the fine fraction as compared to the untreated sample. The fraction of fines in the HTT samples increases with treatment time. However, the subsequent increase in the fines in the other HTT samples is quite less as compared to the HTT30 sample. The increase in the fine fraction can be due to the thermal fractures in the coal particles. These fractures may have originated when the trapped inherent moisture bursts its cavity and escapes as steam. From the PSD, it can be concluded that the most of the pores containing the inherent moisture collapse within 30 minutes of hydrothermal treatment. As a result, the remaining pores continue to collapse gradually with time.

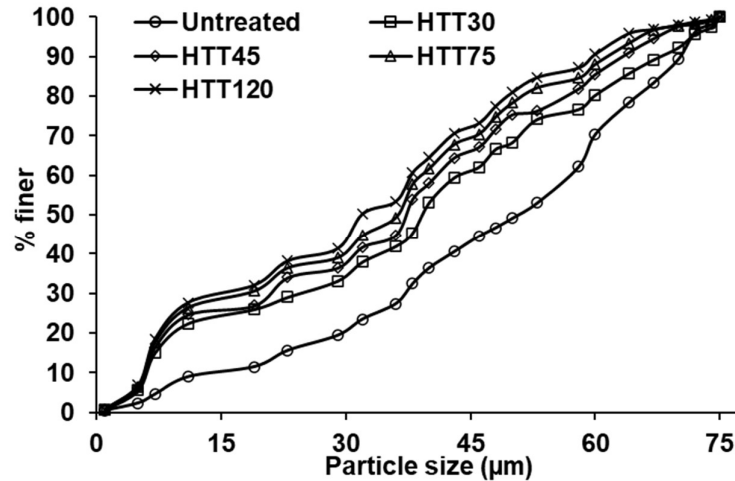


Figure 3.2 Particle size distribution of untreated C-1 and HTT coals at various treatment times.

The PSD of the untreated and the MWT (Microwave treated) samples is represented in Figure 3.3. The MWT, is observed to moderately increase in the fine fraction of the treated coal samples. This increase in the fine fractions is observed to be diminishing with treatment time. As a result, the MWT90 and MWT120 samples have almost same PSD. The moisture locked inside the pore escapes (due to its conversion into steam) after destroying it.

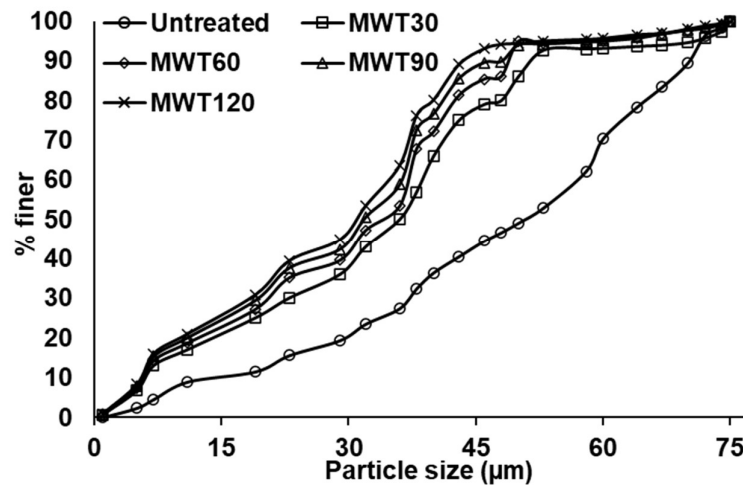


Figure 3.3 Particle size distribution of untreated and MWT coals at various treatment times.

This pressure destruction is the primary source of the generation of the fines in the cases of shorter exposures to the microwaves. In the case of longer exposures, the particles achieve significantly high temperatures close to 200°C and stay at this temperature for 6-7 minutes due to the generation of local hotspots (that stay active for 6-7 minutes). So, in such conditions, the

primary source of the fines' generation is definitely the destruction of the pores due to pressurized steam besides, the high temperature thermal fracture also contributes to the fines' generation.

The untreated sample contains greater fractions of the bigger particles of size $-75+53 \mu\text{m}$. As the bigger particle possess greater moisture (and other MW coupling materials) so, these particles have greater tendency to absorb and accumulate microwaves. As a result, these particles become the primary targets of the microwaves and get eliminated in the consecutive cases. All the MW treated particles possess very small fractions of bigger particles and these fractions decrease with the treatment time. Subsequently, the fine fractions show relative increase with treatment time. The particle size distribution of the segregated samples S_1 , S_2 & S_3 are presented in Figure 3.4. All the samples are monomodal in nature and the modal sizes in S_1 , S_2 & S_3 are 50, 71 and $84 \mu\text{m}$ respectively. The PSD of S_1 is quite uniform and a small peak is observed for the particles of size $+45-50 \mu\text{m}$.

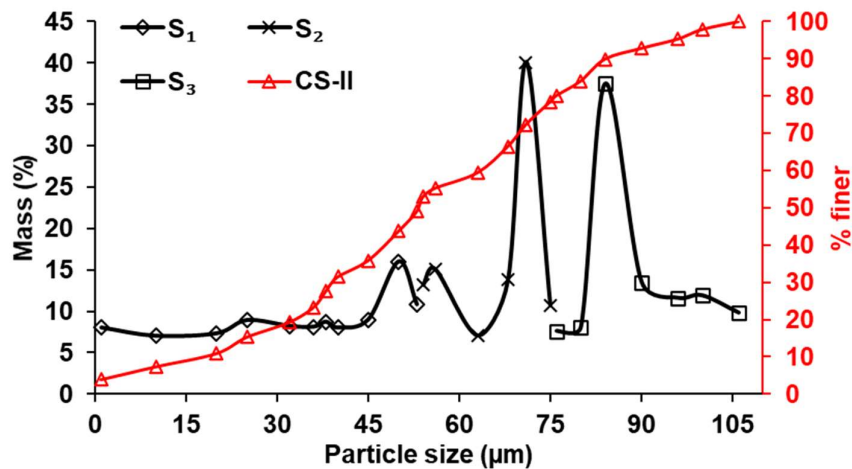


Figure 3.4 Differential PSD of segregated samples (S_1 , S_2 & S_3) of parent sample CS-II and cumulative PSD of coal CS-II.

The sample S_2 is observed to possess a significant fraction of particles of size $+68-71 \mu\text{m}$. Similarly, the coal sample S_3 shows to have a substantial fraction of particles of $+80-84 \mu\text{m}$. The median diameter of S_1 , S_2 & S_3 are 36.6, 68 and $83.65 \mu\text{m}$ respectively.

The particle size distribution of the 8 (4+4) bi-modal coal samples are presented in Figure 3.5. To clearly present the bi-modal PSD, the differential PSD is opted. In samples S_4 and S_8 , a small peak, peak 2 is observed to originate on the addition of 10% (by mass) of S_2 & S_3 respectively. The peak 2 continues to grow with upon addition of more and more coarse

particles as represented in Figure 3.5 b, c, d, f, g and h. The median diameters of different monomodal and bi-modal samples are presented in Table 3.3.

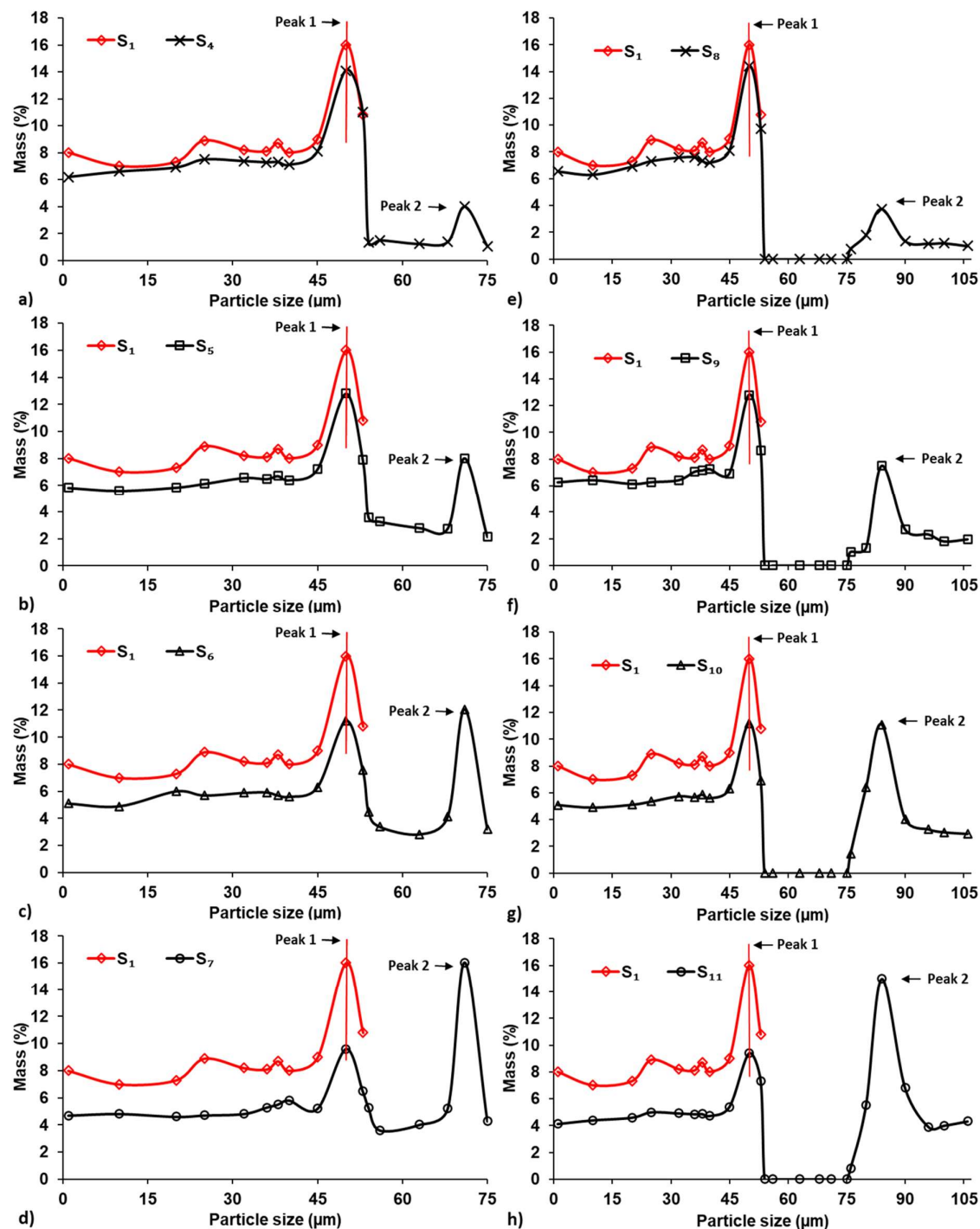


Figure 3.5 The particle size distribution of monomodal S_1 and bi-modal a) S_4 , b) S_5 , c) S_6 , d) S_7 , e) S_8 , f) S_9 , g) S_{10} and h) S_{11} samples

The median diameter is observed to increase with the addition of coarse coal particles.

The samples, S₄ & S₈, S₅ & S₉, S₆ & S₁₀ and S₇ & S₁₁, have almost similar median diameters as presented in Table 3.3.

Table 3.3 The median diameter of the monomodal and bi-modal coal samples

Sample	Median diameter (μm)
S ₁	36.6
S ₂	68.09
S ₃	83.65
S ₄	38.22
S ₅	40.33
S ₆	44.11
S ₇	47.35
S ₈	38.11
S ₉	39.16
S ₁₀	45.17
S ₁₁	48.77

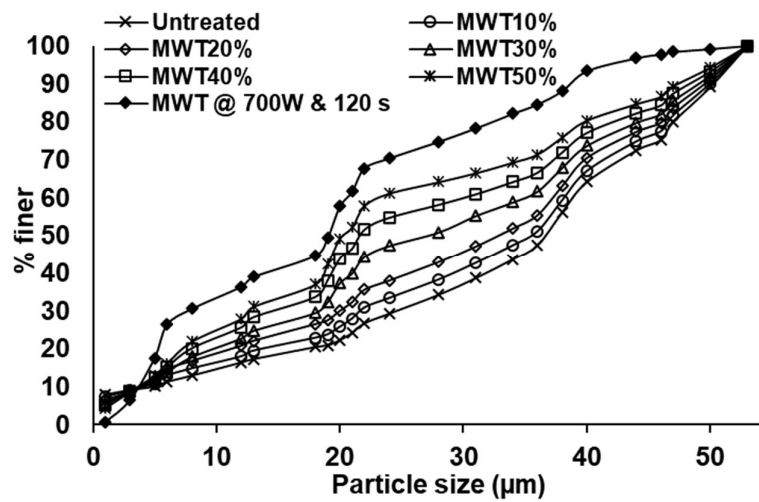


Figure 3.6 Particle size distribution of untreated coal and its blends with its microwave treated coal in different proportions

The -53 μm particles of sample CS-II are blended with the microwave treated sample at 700W and residence time of 120s in 10, 20, 30, 40 and 50% proportion by mass of treated coal. Figure 3.6 presents the particle size distribution of the untreated sample, the blended samples and the 100% microwave treated sample. The untreated coal contains the least fraction of the fine particles while the 100% MWT sample contains the highest fraction of fines. This due to the generation of fine particles as a result of thermal fractures created by the microwaves. When the treated coal is added to the untreated coal, there is subsequent rise in the fine fractions. The median diameter is also observed to fall with the addition of the MWT particles to the untreated sample.

3.3.2. Specific gravity:

The specific gravity of the slurry is measured using pycnometer. A known mass m_c of each coal sample is taken in the pycnometer. The pycnometer is then filled completely with distilled water. The mass of pycnometer, coal sample and water is recorded and represented as m_{PCW} . The pycnometer is now refilled with distilled water only. The mass of pycnometer and water is recorded and represented as m_{PW} . The specific gravity of the coal sample is calculated as:

$$Sp. Gravity = \frac{m_c}{m_c - (m_{PCW} - m_{PW})} \quad 3.1$$

The specific gravity of the untreated sample is 1.378. The HTT leads to increase in the specific gravity of the coal and the maximum specific gravity is observed for the HTT120 sample. Similarly, the MWT coals show an increase in the specific gravity. However, the increase in the specific gravity is higher in the case of the MWT coal as compared to the HTT coal. The increase in the specific gravity is due to the destruction of the pore structure of the coal due to the thermal treatment processes. Due the rapid nature of the microwaves, a greater fraction of pores may have collapsed that leads to greater increase in the specific gravity. The S_1 sample is observed to possess greatest specific gravity out of the three monomodal samples, while S_3 possesses the least specific gravity. The addition of coarse particles of -75+53 μm to the -53 μm sized particles leads to the decrease in the specific gravity of the blended samples. Similar observations have also been made in the case of the samples $S_8 - S_{11}$. The addition of the MWT100% sample to the S_1 leads to the increase in the specific gravity as the former possesses higher specific gravity.

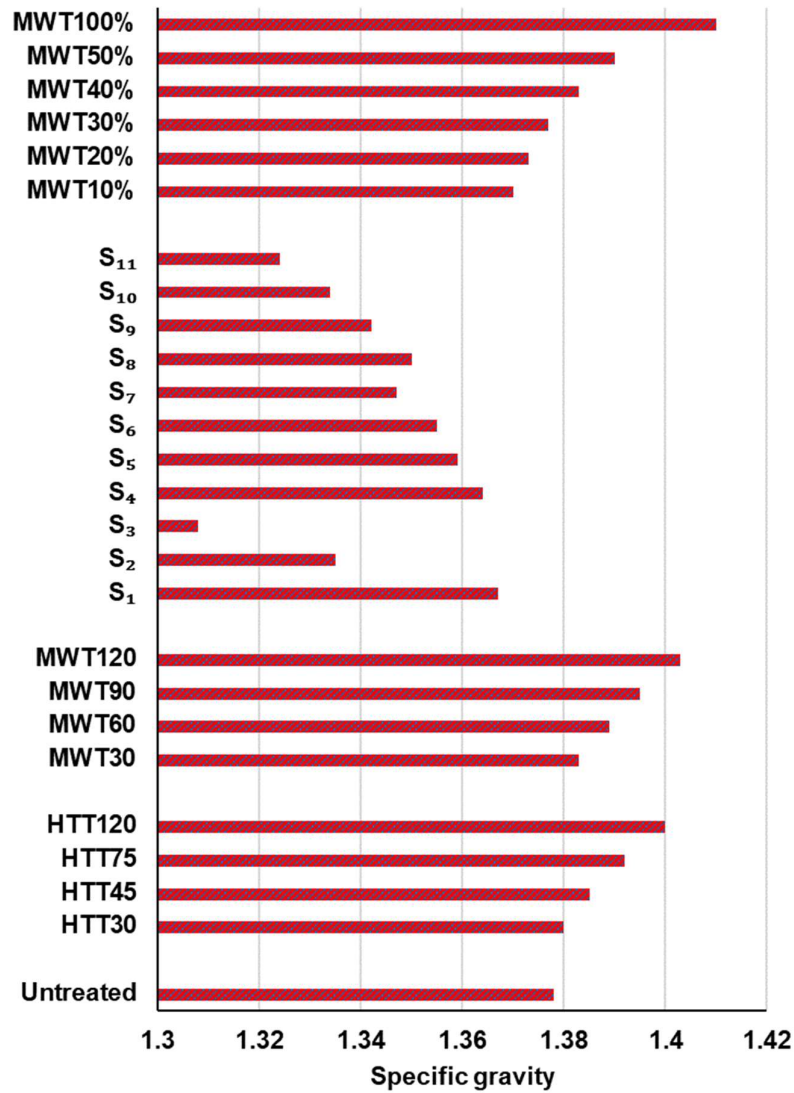


Figure 3.7 Specific gravity of the different sample untreated, treated and blended samples

3.3.3. *Settling characteristics:*

The stability of the slurry is analysed using the gravitational settling method. A sample of known initial solid concentration is taken in graduated cylinder. The initial volume of the well mixed slurry is recorded and is then allowed to settle under stable conditions. A separation layer starts forming after some time, which contains separated water above it and the settled slurry below it. The height of the separation layer is recorded at regular intervals until the maximum settling is achieved. The maximum settled concentration is calculated using equation 3.2. The maximum settled concentration test conducted on the coal sample is presented in Figure 3.8. The maximum settled concentration is found to increase with increase in the initial solid concentration from 30 to 50% by mass.

$$C_{SS} = \frac{\frac{C_M M_L}{1 - C_M}}{\frac{C_M M_L}{1 - C_M} + \rho_L (V_L - V_{LS})} \quad 3.2$$

C_{SS} : Maximum settled concentration V_L : Volume of liquid in the slurry (ml)

C_M : Solid concentration by mass V_{LS} : Volume of the separated liquid after settling (ml)

M_L : Mass of liquid in the slurry (g) ρ_L : Density of liquid (g/cc)

However, the relative increase is found to decrease, with initial solid concentration, indicating greater stability at higher concentrations. The slurries at higher concentrations are overcrowded as a result, the frequent collisions of the particles among them

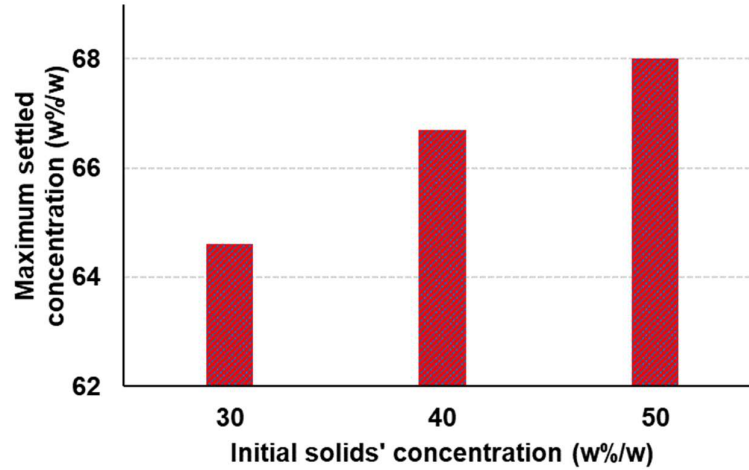


Figure 3.8 Settling characteristics of untreated coal

The settling characteristics of different coal slurries have been reported in Figure 3.9. There is a significant decrease in the maximum settled concentration values for HTT coals as compared to the raw coal. It is observed that these values further decrease with the treatment time and increase with initial solid concentration. From the chemical characteristics (as discussed in section 3.3.1) of untreated and HTT coals, it is observed that there is significant reduction in the oxygen containing hydrophilic groups, as a result, the hydrophobicity of coal is observed to be increasing upon HTT. Due to increased hydrophobicity, all the water remains in the particle's surroundings, in spite of penetrating into it. So, there is a greater fraction of water in the settled slurry of HTT coals, which leads to the decrease in the maximum settled concentration with treatment time. So, the hydrothermal treatment proves to be beneficial for increasing the stability of the CWSs.

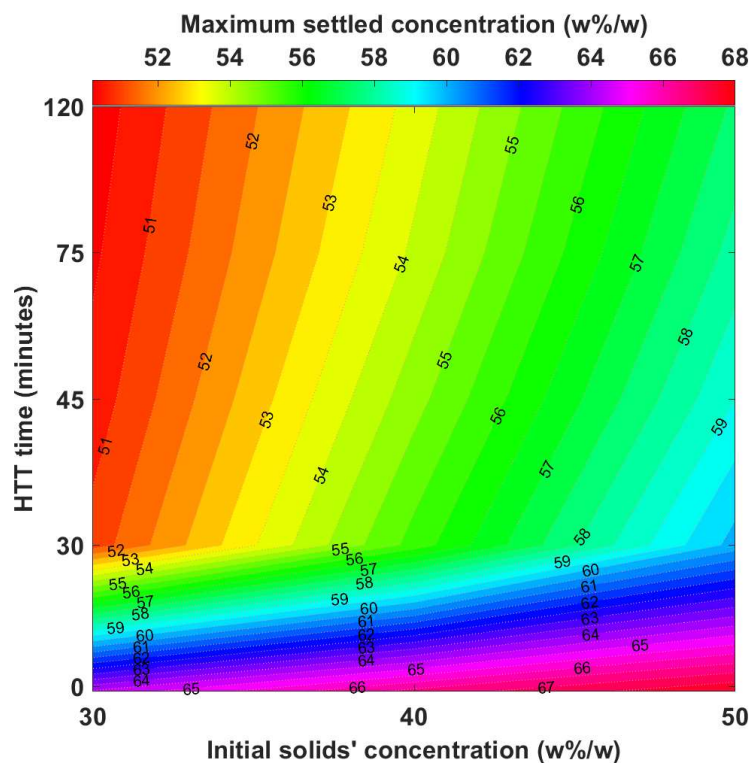


Figure 3.9 Settling characteristics of untreated and HTT coals

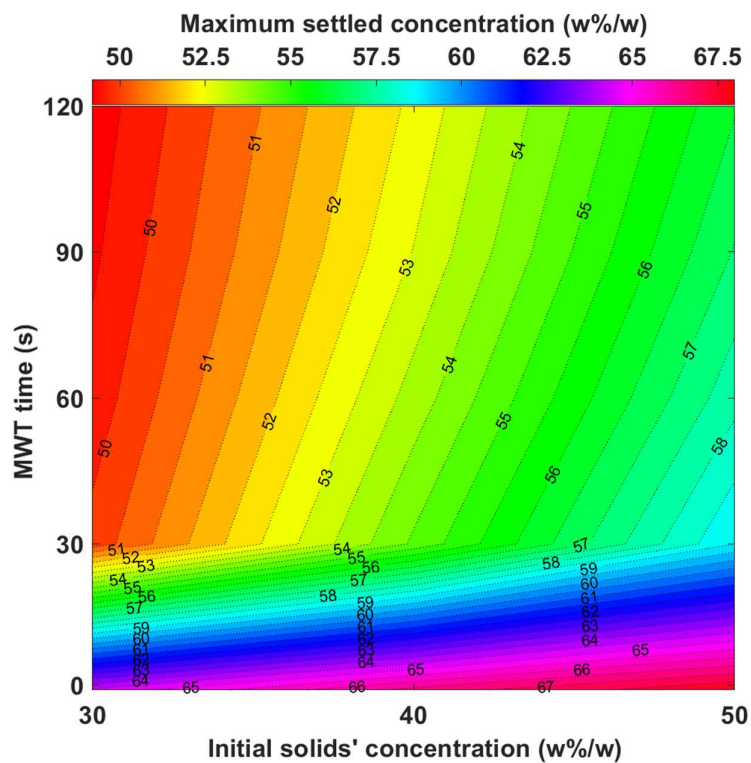


Figure 3.10 Settling characteristics of untreated and MWT coals

A significant reduction in maximum settled concentration is observed upon MWT as presented in Figure 3.10. The maximum settled concentration is found to decrease by 23.83, 21.09 and 16.97% for initial solid concentration of 30, 40 and 50% upon MWT for 120s. Such trends are observed due the increase in the hydrophobicity of the coal. Due to the loss of polar oxygen functional groups, interactions between coal particles and the water molecules decrease so, greater amount of the water is available around the particles. This helps the particle to remain suspended for a longer time. With the increase in the initial solid concentration, the particle-to-particle distance gets reduced. The solid particles possess a charge on their surface and with decrease in the particle-to-particle distance, the repulsive forces between two adjacent particles are significantly increased due to which the particles are in a random motion for a longer time and take more time to settle.

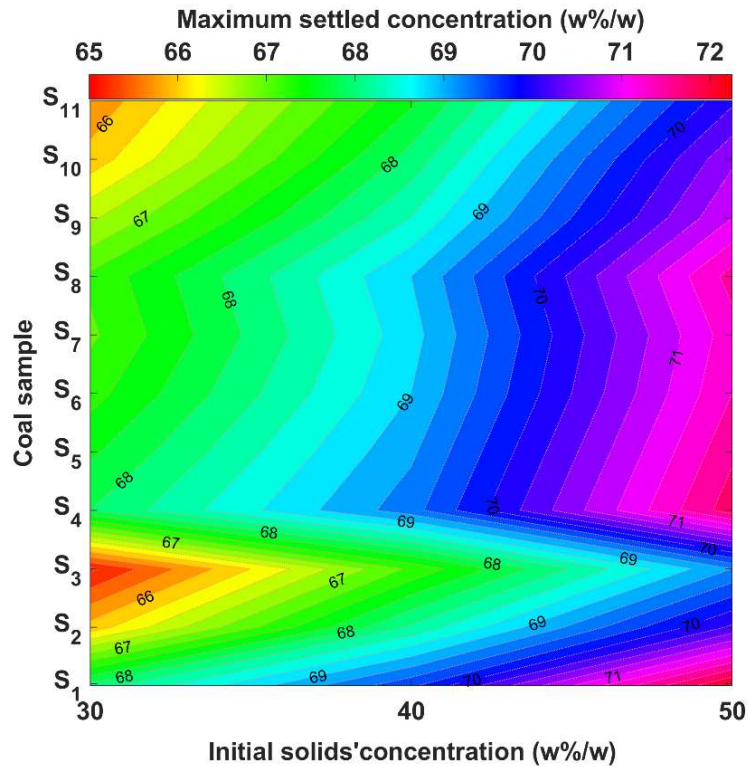


Figure 3.11 Settling characteristics of monomodal and bimodal coal samples

The results from the settling analysis of the fine and blended samples have been represented in Figure 3.11. It is observed that the -53 μm sized fine particles show maximum settling. With the addition of -75+53 μm sized coarse particles in -53 μm sized fine particulate slurry, the stability of slurry shows an improvement as the C_{ss} value for bi-modal samples S4-S7 decreases. Similar trends of enhanced stability can be observed with the addition of -106+75

μm coarse particles in the fine particulate slurry. It is observed that $-106+75 \mu\text{m}$ sized coarse coal particles are more effective as compared to the $-75+53 \mu\text{m}$ sized coarse coal particles. Out of bi-modal samples S_4 - S_7 the maximum stability can be observed in the case of bi-modal sample S_7 , whereas out of bi-modal samples S_8 - S_{11} , the maximum stability can be seen in the case of sample S_{11} . It is observed that the maximum settled concentration values are dependent on the initial solid concentration.

Usually, in case of dilute slurries or at low solid concentrations, the settling behaviour depends upon the different Stoke's parameters; viscosity, particle diameter and relative particle velocity. But in the present case the slurries are so concentrated that the Stoke's law fails to explain the settling behaviour of the solid in the slurries. In highly concentrated slurries, hindered settling behaviour is observed, which is greatly influenced by solid concentration (C_M) in spite of stoke's parameters. In concentrated slurry, the particles are quite over crowded, due to which there is interference in the particle motion as a result the settling rate lowers as compared to the normally expected.

In the present case of mono and multi-sized particulate slurries, the settling rates are decided by particle interaction effects, consisting of collision between the particles and non-hydrodynamic forces [176]. So in the present case, as the initial solids' concentration is increased, the particle to particle interactions increase proportionally, as a result, the maximum settled concentration shows an increase.

The maximum settled concentration of the untreated coal is observed to be 66.92, 68.9 and 70.7% at initial solid concentration of 30, 40 and 50% respectively as represented in Figure 3.12. When these investigations are performed for the treated coal, a sharp decrease in the maximum settled concentration is observed, irrespective of the initial solid concentration. The maximum settled concentration is observed to decrease by 18.1, 13.4 and 8.8% when the untreated coal is blended with 10% treated coal. The maximum settled concentration follows a decreasing trend with the percentage of addition of treated coal. The maximum settled concentration values of MWT40 and MWT50 coals are almost similar. The maximum decrease in the maximum settled concentration is observed to be 19.8, 16.6 and 13.08% in the case of 50% blend of treated coal, i.e., MWT50 coal sample. These results clearly witness the improvement in the stability characteristics of the slurry. Also, the stability shows insignificant change post 40% proportion of the treated coal. So, 40% proportion of treated coal is optimum for a stable slurry.

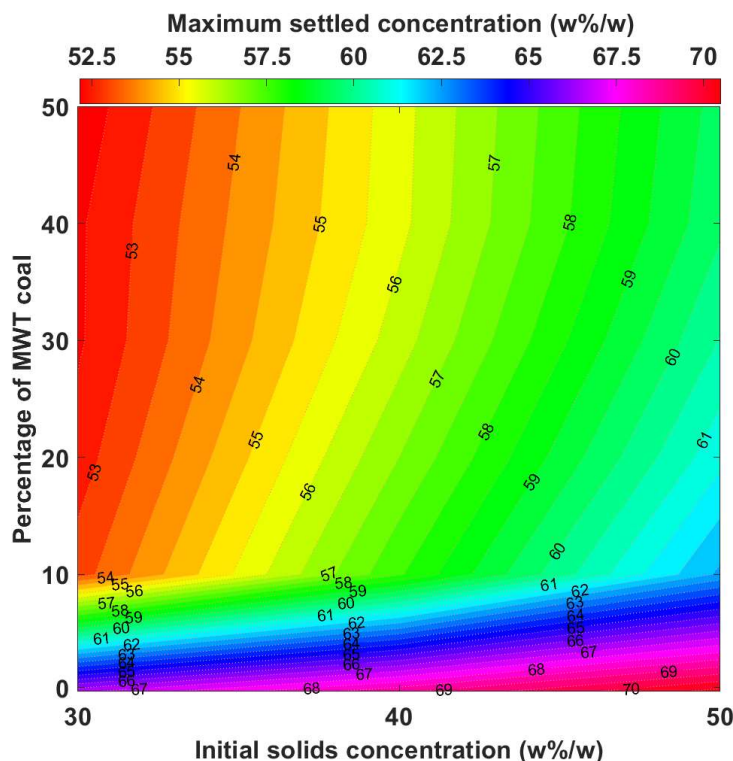


Figure 3.12 Settling characteristics of untreated and the blended coals

The reason behind this improvement is the destruction of the hydrophilic oxygen functional groups, due to which the coal becomes more hydrophobic. The increased hydrophobicity does not allow the water to penetrate the pores of the particle. So, the treated coal particles have greater volume of water in their surroundings, which help them to remain suspended for considerably longer duration of time. So, now the settled slurry has a greater volume water in it. Due to this a decrease in the maximum settled concentration is observed. The increased hydrophobicity helps in balancing the solid particle in the suspending medium.

3.4 Chemical characterization

The chemical characteristics of the untreated and treated coals are analysed using the bench scale test by following the respective standard procedures. The following bench scale tests are carried out on all the coal samples:

- i. Proximate analysis
- ii. Ultimate analysis
- iii. Energy dispersive X-ray spectroscopy (EDS)

3.4.1 Proximate and ultimate analysis:

The chemical characteristics of coal are investigated using proximate and ultimate analysis. These tests are carried out as per the standards mentioned in the Table 3.4.

Table 3.4 The standards/methods for the proximate and ultimate analysis of coal

Proximate analysis (air dried basis)	
Constituent	Standard procedure
Moisture (M)	ASTM D3173M-17a
Volatile Matter (VM)	ASTM D3174-12
Ash (A)	ASTM D3175-17
Fixed carbon (FC)	By difference
Ultimate analysis (air dried & ash free basis)	
Element	Standard procedure
Carbon (C)	ASTM D5373-16
Hydrogen (H)	ASTM D5373-16
Nitrogen (N)	ASTM D5373-16
Sulphur (S)	ASTM D4239-17
Oxygen (O)	By difference

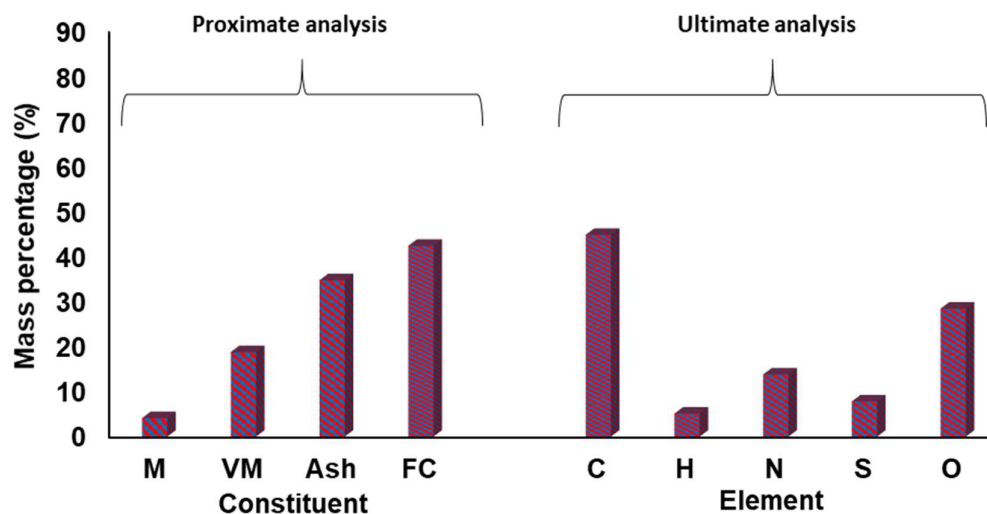


Figure 3.13 Proximate and ultimate analysis of the untreated coal.

The proximate and ultimate analysis of the untreated coal sample is presented in Figure 3.13. The proximate analysis of the untreated coal shows that the present coal sample is rich in ash and contains almost 30% ash by mass. The fixed carbon and volatile matter in the coal are 42.4% and 17.2% respectively. This signifies that the coal source is immature or young in nature. The ultimate analysis indicated the presence of significant amount of oxygen in the form of hydrophilic oxygen functional groups. The elemental carbon in the coal is close to 44.81%.

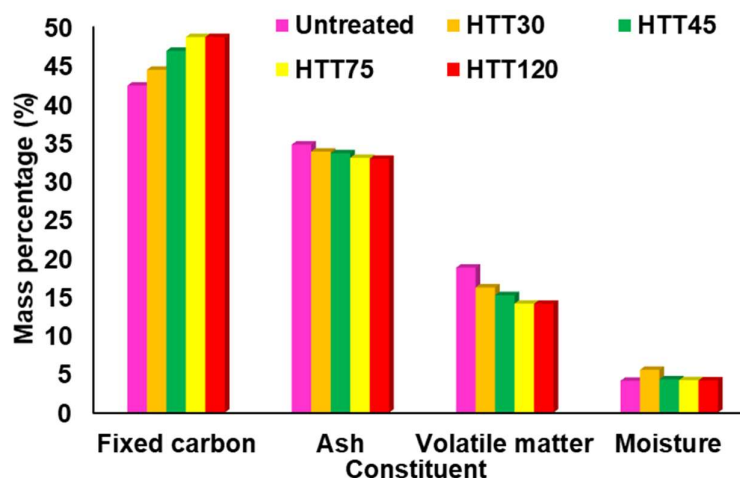


Figure 3.14 Proximate analysis of the HTT coals.

From the proximate analysis, as represented in Figure 3.14, it is observed that there is a significant increase in the fixed carbon fraction upon HTT. The fixed carbon fraction shows an increasing trend with treatment time. An increase of 14.84 and 14.86% in the fixed carbon fractions is observed after a treatment time of 75 and 120 minutes respectively. The ash and the volatile fraction show a decreasing trend up to the treatment time of 75 minutes and remains constant afterward. Whereas the moisture fraction remains almost constant with treatment time. It is found that there is almost no difference in the proximate analysis of HTT75 and HTT120 coal samples.

The ultimate analysis of the HTT coals is presented in Figure 3.15. The HTT leads to a significant decline in the oxygen fraction. The other elemental components viz nitrogen, sulphur and hydrogen also show a decrease with the HTT time. The HTT initiates the process in the which due to high temperature, the loosely bound light volatiles start escaping the coal particle. Majority of these light volatiles escape the coal particles until treatment time of 75 minutes. An almost insignificant change in the elemental composition of coal is observed upon HTT for 120 minutes. The decrease in the fractions of oxygen, nitrogen, sulphur and hydrogen

leads to relative increase in the carbon fraction upon HTT.

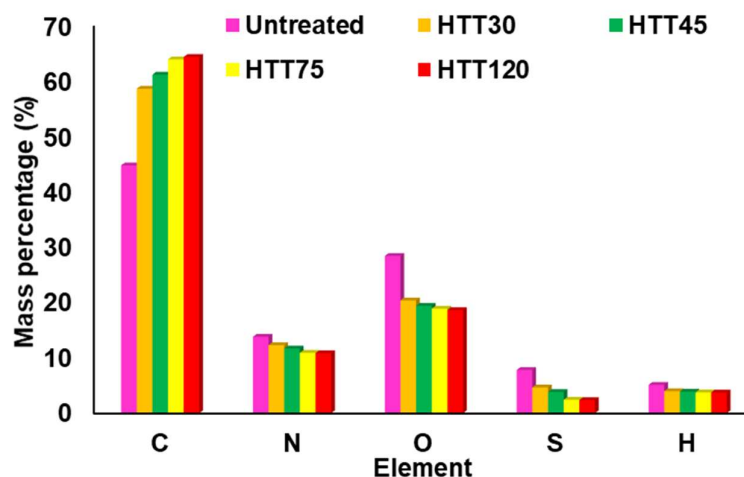


Figure 3.15 Ultimate analysis of HTT coals

Figure 3.16 represents the proximate analysis of the MWT samples. The fixed carbon fractions are observed to be increasing significantly with heating time. The moisture fraction shows a decrease of ~1.5 wt% with MWT. The volatile fractions are observed to be decreasing by ~5% up to a treatment time of 90 seconds. The loss of volatiles is in the form of light volatiles composed of CO₂, and SO₂, that can escape the solid fuel at a temperature >50°C. There is a rise of 17.79 & 18.29% in the FC (fixed carbon) fraction, for the heating time of 90 and 120 seconds. This increase in the FC is relative and mostly due to the loss mass in the form of moisture and light volatiles. The proximate analysis results of MWT90 and MWT120 coals are observed to be almost similar

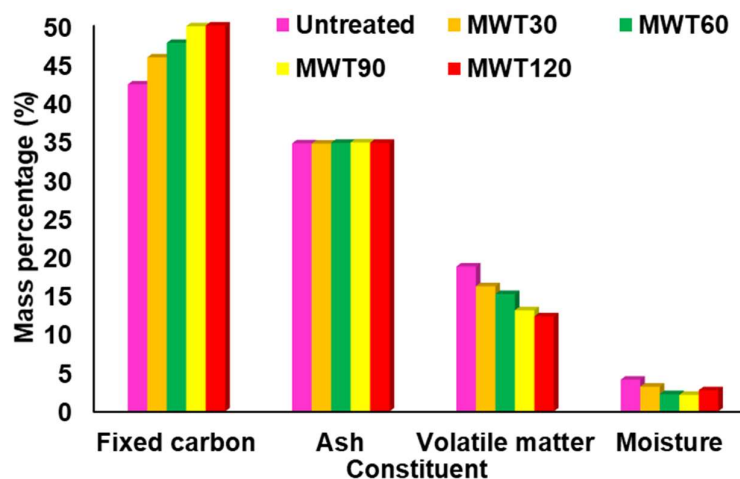


Figure 3.16 Proximate analysis of the MWT coals.

The results of ultimate analysis performed according to the standard procedure mentioned in Table 3.3 are presented in Figure 3.17. The ultimate analysis shows that carbon levels are significantly upgraded by MWT and the oxygen levels are found to fall sharply. There is a moderate reduction in the nitrogen, hydrogen and sulphur fraction upon MWT. The sample, treated for 120 s, showed a maximum increase of 51.1% in the carbon fraction.

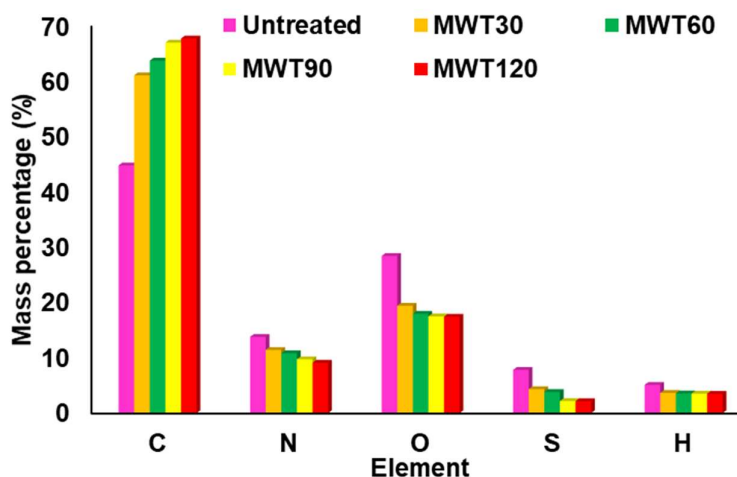


Figure 3.17 Ultimate analysis of MWT coals

The increase in the carbon fraction is purely due to the mass loss in the form of other components viz. nitrogen, hydrogen, oxygen and sulphur. The oxygen and hydrogen fractions are observed to decrease by 38.6 and 31.4% respectively. This indicates that upon MWT, the hydrophilic -OH functional groups are eliminated from the coal. The nitrogen and sulphur fractions are observed to decrease by 33.9 and 72.3% upon treatment for 120s. This indicates the loss of volatiles in the form of SO₂.

3.4.2 Energy dispersive X-ray spectroscopy (EDS):

The chemical characterization of coal is also carried out using Energy Dispersive X-ray Spectroscopy (EDS), which is an integral part of the scanning electron microscope. The spectrum generated by the reflected radiations gives the measurement of the different elements. The energy dispersive spectrum gives an approximate idea about the chemical composition and the minerals present in the sample. It can easily detect the elements having atomic number > 4 but, it has a drawback that it cannot detect the elements having atomic number ≤ 3 , i.e. H, He & Li, as these elements do not have X-rays. So, in order to detect the hydrogen fraction in the coal, the ASTM D5373-16 standard method is used and represented as a part of ultimate analysis in section 3.3.1.

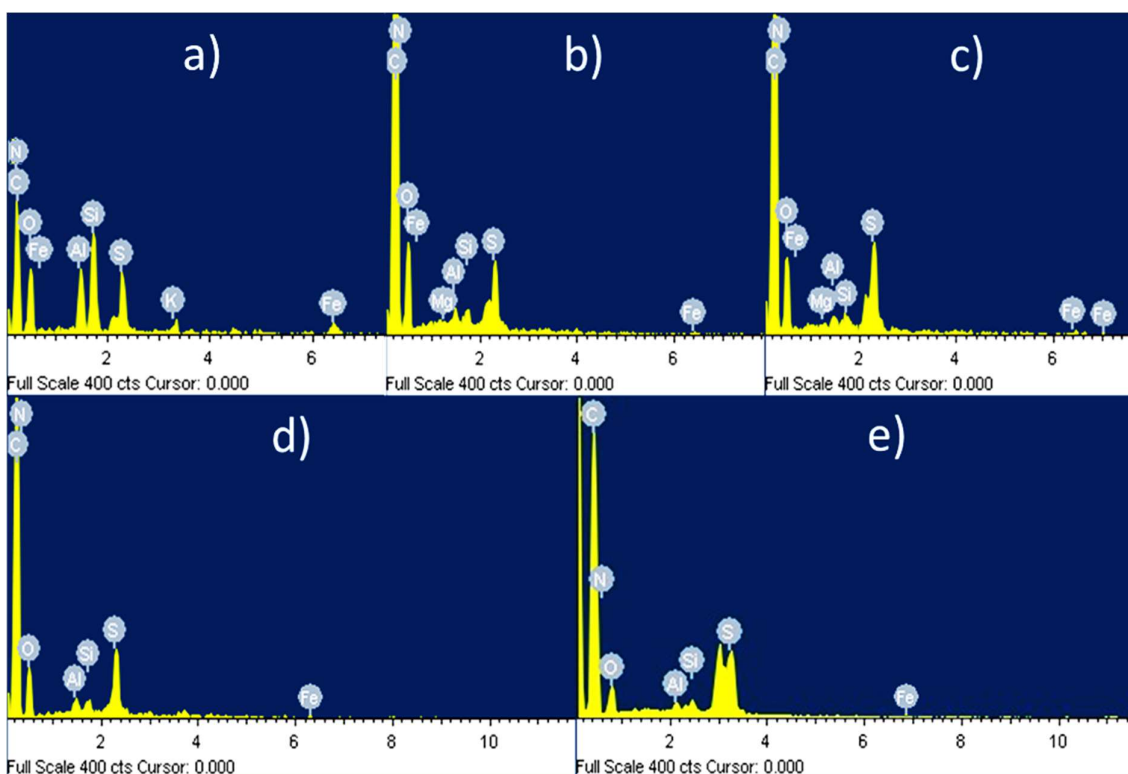


Figure 3.18 Energy dispersive spectra of a) Untreated, b) HTT30, c) HTT45, d) HTT75 and e) HTT120 coals

The EDS spectrums obtained for different coal samples are represented in Figure 3.18. It is observed that there is an increase in the carbon and nitrogen fractions whereas the oxygen shows a decrease with treatment time. The EDS spectrums of d) HTT75 and e) HTT120 coal samples are observed to resemble each other, which indicates almost similar chemical structure of the two samples. The chemical composition interpreted from the EDS spectrums is represented in Figure 3.19.

From Figure 3.19, it is observed that the carbon mass percentage increases up to treatment time of 75 minutes, and a negligible change is observed after a treatment time of 120 minutes. Similarly, nitrogen mass percentage is found to be increasing up to treatment time of 75 minutes and remains constant afterward. Whereas, an inverse trend is observed for oxygen mass percentage, which decreases up to treatment time of 75 minutes and a very small decrease is observed for HTT120 coal sample. The elements Al, Si and Fe, which are initially present in a small fraction are almost eliminated upon HTT. The Mg mass levels evolve with treatment time, but the change is quite insignificant. The sulphur levels decrease by a constant fraction for all HTT samples.

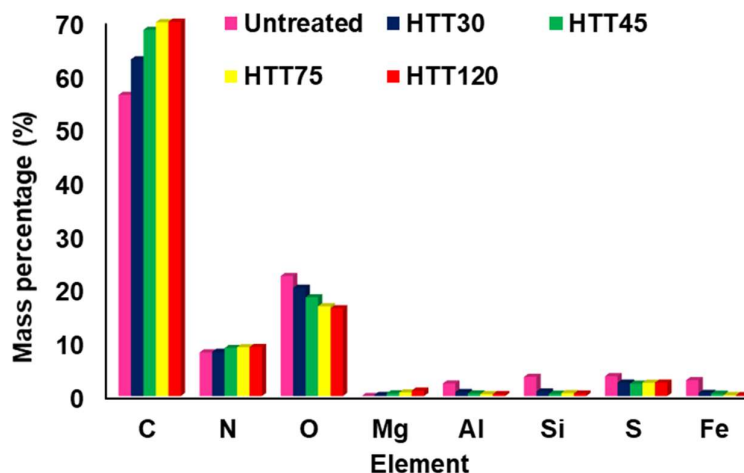


Figure 3.19 Chemical composition of untreated and HTT coals, interpreted from EDS spectra

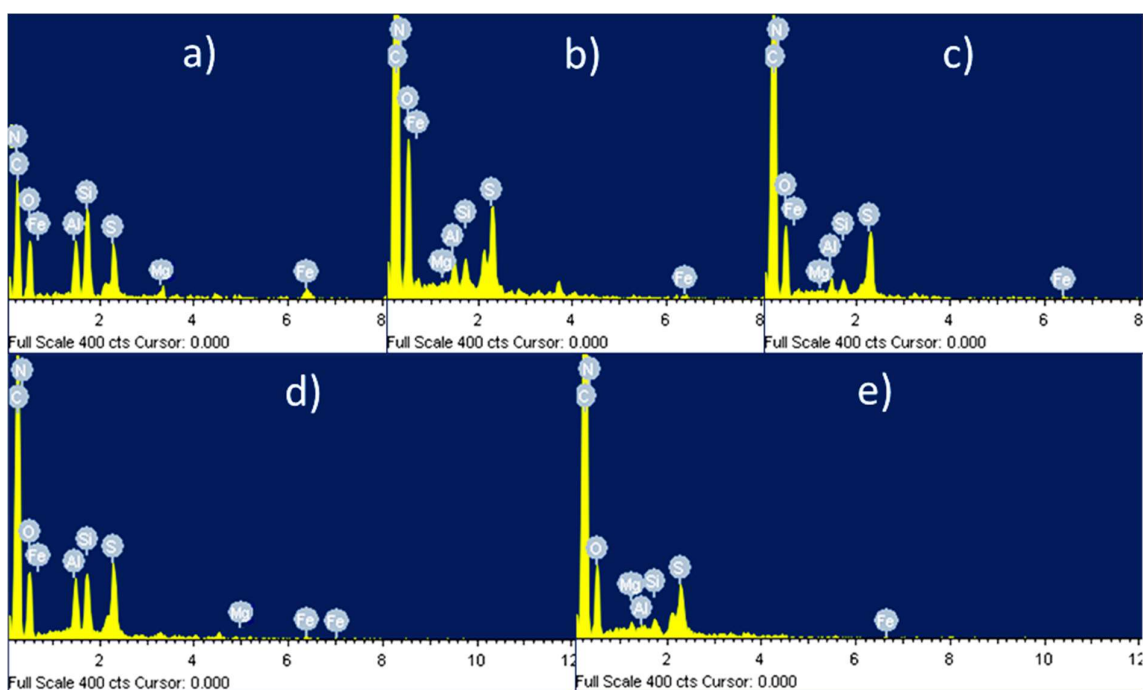


Figure 3.20 Energy dispersive spectra of a) Untreated, b) MWT30, c) MWT60, d) MWT90 and e) MWT120 coals.

The energy dispersive spectra (EDS) of the untreated and the MWT coals are presented in Figure 3.20. The EDS results show that upon MWT, there is a moderate decrease in the oxygen, nitrogen, magnesium, aluminium, silicon, sulphur and iron levels but a significant rise in the carbon fraction. The increase in the carbon content in the treated coals is relative and

mostly due to the loss of elements comprising the moisture and light volatiles viz. hydrogen, oxygen sulphur and some fractions of nitrogen. The MWT is observed to be effective up to a treatment time of 90 s because there is almost no variation in the EDS spectra of MWT90 and MWT120 coals. The EDS results are also represented in the mass percentage of the individual element, interpreted from the spectra in Figure 3.20. The carbon fraction is found to increase sharply for treatment time of 30s and a gradual change is observed afterward up to 90s and almost no change for 120s. For this change, the loss of moisture and light volatiles are responsible.

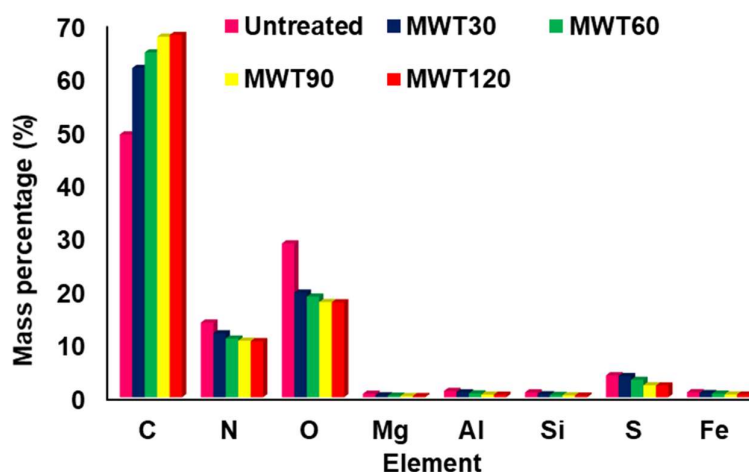


Figure 3.21 Chemical composition of untreated and MWT coals, interpreted from EDS spectra

The EDS results are also represented in the mass percentage of the individual element, interpreted from the spectra in Figure 3.21. The carbon fraction is found to increase sharply for treatment time of 30s and a gradual change is observed afterward up to 90s and almost no change for 120s. For this change, the loss of moisture and light volatiles are responsible. For a treatment time of 30s, the carbon fraction is observed to increase by 25.29% whereas the maximum increase in carbon fraction is 39.36% observed upon treatment for 120s. Unlike carbon fraction, the oxygen in the treated coals is found to decrease sharply upon treatment for 30s and a gradual decrease is observed afterward. The oxygen fraction is found to decrease by 31.93% upon treatment for 30s and a maximum reduction of 39.69% is observed for MWT120 coal. That indicates loss of moisture and light volatiles. The maximum reduction in the nitrogen fraction is 28.38%. Magnesium, aluminium, silicon and iron, are almost eliminated upon MWT. The sulphur mass fraction is reduced by 46.84% upon MWT for 120s as a component of light volatiles in the form of SO_2 .

3.5 Morphological characterization

The morphological characteristics of the coal samples are analyzed using JOEL JSM-6510LV model of scanning electron microscope (SEM). Coal being non-conducting is initially coated with a thin layer of gold; this helps in getting sharp images of the particles. The SEM micrographs of untreated and HTT treated coals are represented in Figure 3.22. It is observed that the untreated coal sample has a very less fraction of fines and the particles $>50\mu\text{m}$ are present in a greater amount as compared to HTT samples.

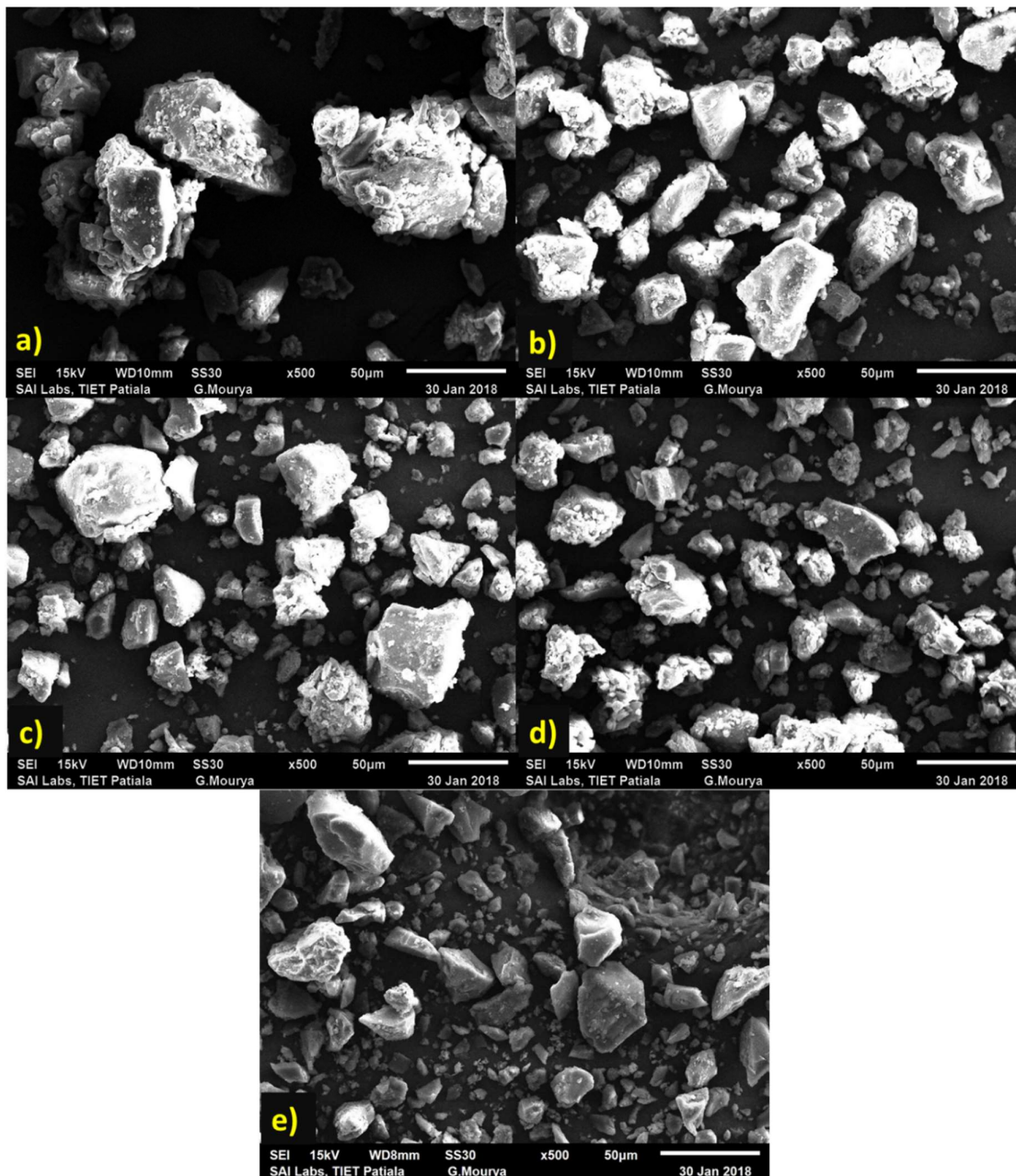


Figure 3.22 Scanning electron micrograph of a) untreated, b) HTT30, c) HTT45, d) HTT75 and HTT120

Upon HTT, an increase in the fraction of fines is observed in the treated samples. This is due to the thermal fracture/cracking of the coal particles. The SEM micrograph of a) untreated coal indicates agglomeration of the coal particles. Whereas in the SEM micrographs of the HTT samples, there is a significant reduction in the agglomeration and the particles appear to be well dispersed.

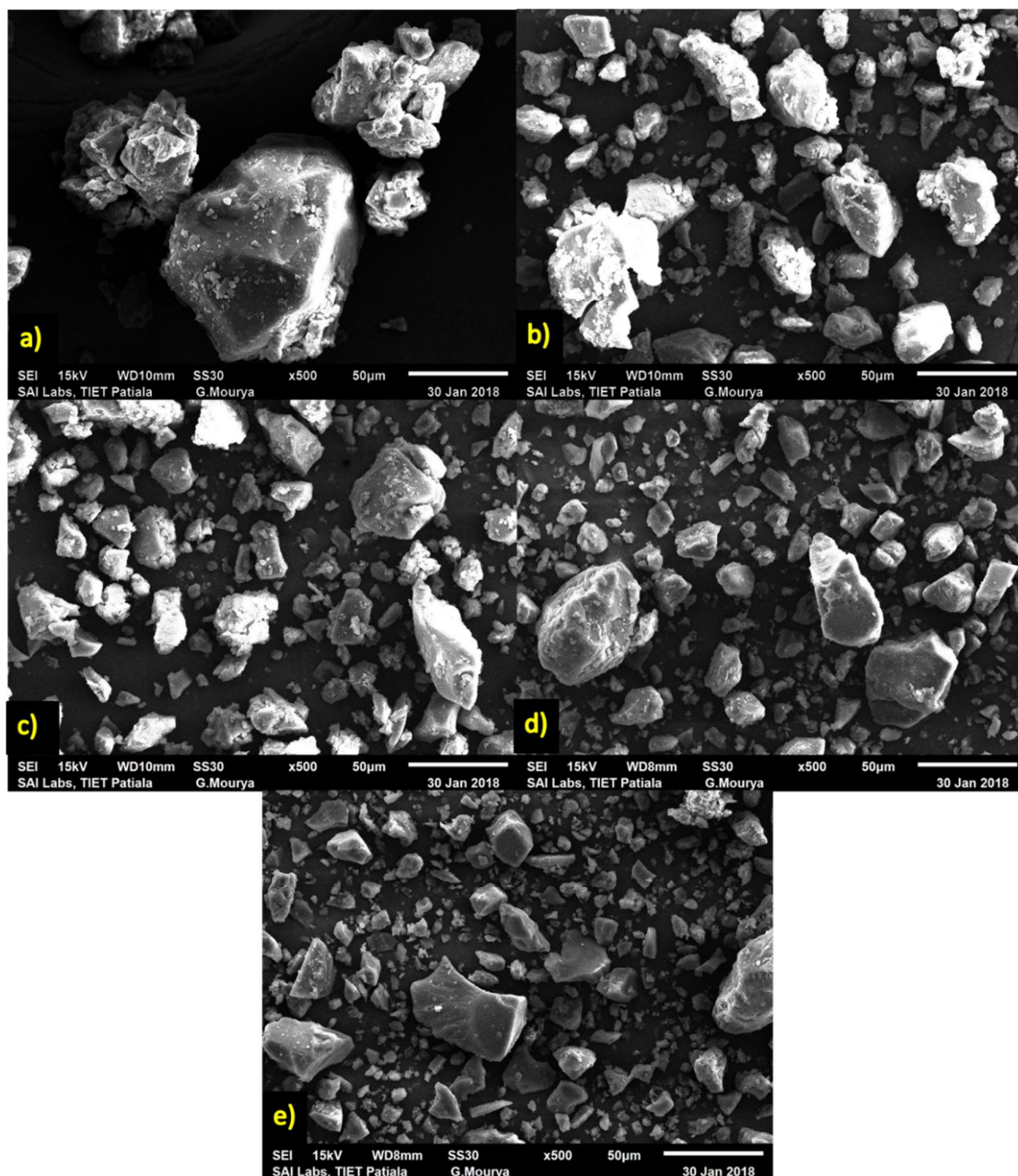


Figure 3.23 Scanning electron micrograph of a) untreated, b) MWT30, c) MWT60, d) MWT90 and MWT120

It is observed that the microwave treatment (Figure 3.23) of the coal leads to the development of cracks and fractures in the particles (explosion of the pores by the locked moisture), due to this, there is an increase in the fine fractions in MWT coals. The SEM micrograph; a) of the untreated coal shows a cluster of the coal particles. The MWT leads to the elimination of the clusters and the particles are observed to be well dispersed. The micrographs of d) MWT90 and e) MWT120 coals are almost similar, which represents that the MWT has a small influence on the coal sample post 90s of treatment.

3.6 Concluding remarks:

In the present chapter, the physico-chemical characteristics of Indian high ash coal have been studied using different bench scale tests. These properties serve as the base for the design of slurry pipelines meant for different applications. The effect of hydrothermal treatment and microwave treatment on the physico-chemical and morphological characteristics of the coal. The fine particulate slurries are desirable for the combustion application due to high surface area. So, the effect of addition of coarse particles on the stability of the blended slurry is also investigated. The following conclusions can be drawn from the results:

1. The present coal sample has high ash (34.73%) and high volatile matter (18.78%). This indicates that the coal sample has high fraction of hydrophilic matter.
2. The median diameters of the untreated, HTT120 and MWT120 coals are 51.7 μm , 32.2 μm , and 22.4 μm respectively. So, hydrothermal treatment and the microwave treatment processes lead to the generation of the fine particles due to thermal breakage of the particles.
3. The hydrothermal and microwave treatment processes lead to the elimination of light volatiles by 24.9% (HTT120) and 34.3% (MWT120) respectively. Similarly, the elemental oxygen is eliminated by 34.43% and 38.65% by hydrothermal and microwave treatment processes.
4. The maximum settled concentration of the hydrothermal and microwave treated samples are reduced by 22.2 and 23.8%. Similarly, the maximum settled concentration for the fine sample blended with MWT coal is reduced by 27.6%.
5. The 30% proportion of -75+53 μm coal or 20% proportion of -106+75 μm coal are observed to be optimum for increasing the stability of the suspended particles. The addition coarse particles beyond the above values lead to setting of the coarse particles.

Chapter 4

Rheological characterization of coal water slurry

4.1 Introduction

The dispersion of a solid powder into a liquid/conveying media, significantly alters the rheological characteristics of the complex fluid (the slurry). These characteristics are a strong function of the physico-chemical properties of the solids that have been studied extensively in Chapter 3. The coal water slurry is a complex liquid fluid used in coal gasification and combustion applications. The combustion applications include both internal (marine engines) and external combustion engines. These sophisticated applications require coal of -106 μm size (major fraction of fine -53 μm sized particles) for the preparation of the coal water slurry fuel. The fines (-53 μm sized particles) have a higher surface area, better for fast and efficient combustion. The previous limited investigations have indicated a significant increase in the viscosity of the CWS with reduction in the particles size. The transportation of such highly viscous fuel may require additional pumping power. The effect of various treatment and blending processes on the rheological characteristics of fine untreated coal is hence analysed. The following methodology is followed for these series of investigations:

4.2 Experimental procedure

4.2.1. Slurry preparation for rheological characterization:

The rheological characterization is performed on the solid-liquid (slurry) samples prepared with the help of the normal water, to match the industrial standards. A calculated mass of sample is mixed in 35 ml water. The slurry is then homogenized using stirrer for sufficient time ranging from 15-20 minutes. The concentration of solid particles (untreated and treated coal) is taken on mass basis and the range lies between 10-60%. The slurry is then used to perform the rheological characterization.

4.2.2. Rheological characterization using the RheolabQC rheometer:

The RheolabQC model of the rheometer supplied by Anton Paar, GmbH, Germany is used for the present rheological characterization. The specifications of the RheolabQC are presented in Table 4.1

Table 4.1 Specification of the RheolabQC rheometer.

Parameter	Specifications
Speed	0.01 to 1200 RPM
Torque	0.20 to 75 mNm
Shear stress	0.5 to 30000 Pa
Shear rate	10^{-2} to 4000 s^{-1}
Viscosity measuring range	1 to 109 mPa.s
Temperature range	-20 to 180 °C
Mass	14 kg

**Figure 4.1 RheolabQC rheometer and the attached PC for the data acquisition**

Figure 4.1 presents the photograph of the RheolabQC rheometer, that is connected to a PC via LAN port for the in-situ acquisition of the data into the system. The Windows® based Rheoplus software is used to visualize the data synchronized between the rheometer and the PC. The ToolMaster™ module of the Rheoplus is responsive option as it automatically adjusts the measuring methods according the measuring system used in a particular case/experiment.

This eliminates the chances of error originating due to the improper choice of the measuring system and the measurement methods.

4.3 Rheological characterization

4.3.1. Rheological characteristics of -75 μm untreated coal-water slurry:

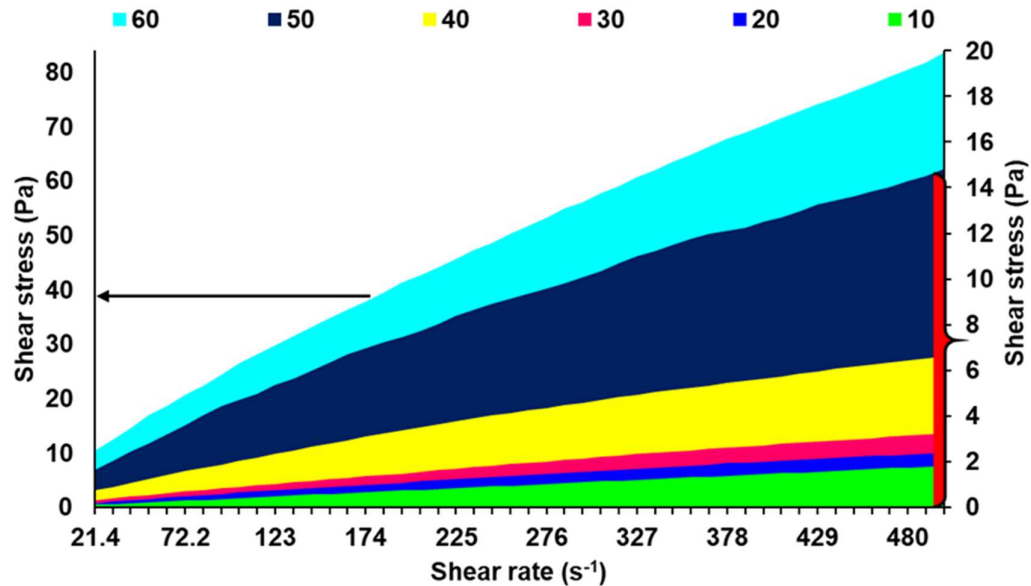


Figure 4.2 Variation of shear stress with shear rate of CWS at 10-60% coal concentration by mass

The variation of the shear stress with shear for the coal water slurry is presented in Figure 4.2. The results indicated that the shear stress is observed to increase with shear rate and the concentration of the coal dispersed in the water. When the shear stress vs shear rate data is fitted using non-linear regression, it is observed that the coal water slurry behaves like a Newtonian fluid up to 30% concentration of the coal (by mass). When the concentration is further increased to 40%, the shear stress follows an increasing trend with the shear stress. However, the increase is no longer linear as the relative increase in the shear stress with shear stress is found to diminish gradually. So, the behavior of the slurry is eventually changed from Newtonian behavior to non-Newtonian or power law fluid. When the concentration of the coal particles is further increased to 50%, high shear stress is observed at low shear rates, indicating the presence of yield stress. This indicates the start of the overcrowding of the particles in the dispersing media. The further increase of concentration to 60% shows a critical change in the rheological characteristics of the coal water slurry. There is an abrupt increase in the shear stress at 60% solid concentration and the slurry almost approaches to a highly viscous fluid structure.

At solid concentrations of 10-30%, the coal particles are well dispersed in the water and there are very less interactions between the coal particles. The water between the coal particles served as a lubricant whenever a coal particle tries to come in contact with the other. However, the further increase in the concentration of the coal particles, slightly changes the behavior of the CWS. The slurry now behaves like a shear thinning fluid. As indicated earlier, the overcrowding of the particles starts at 50% solid concentration. Now at this concentration, the coal particles actually come in contact with each other, due to lesser availability of the water. The coal-water shearing is now replaced by coal-coal shearing, as a result, the yield stress comes into picture. Due to the overcrowding of the particles, the slurry along with some water makes a compact and stable structure, that is hard to break. The shear thinning behavior is also exhibited by CWS at higher shear rates. The further addition of coal particles by 10% leads to increased interactions between the coal particles. The pore structure in the coal absorbs most of the water available around the coal particle. As a result, there is a deficiency of water in the slurry. So, the yield stress is highly increased at 60% concentration. Further at higher shear rates, the shear stress is also observed to magnify to almost 6 times of the value at 50% solids concentration.

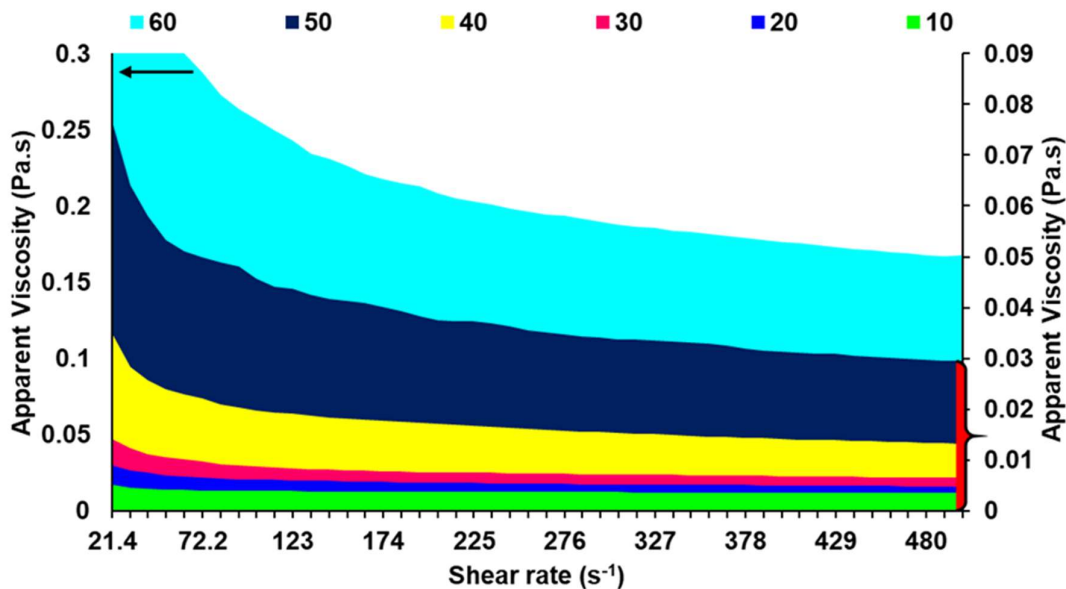


Figure 4.3 Variation of apparent viscosity with shear rate of CWS at 10-60% coal concentration by mass

Figure 4.3 represents the variation of apparent viscosity with shear rate at 10, 20, 30, 40, 50 and 60% solid concentration. As the slurries behave like a Newtonian fluid, up to 30% solids' concentration, so the apparent viscosity of such slurries is observed to be a constant

value, irrespective of the shear rate. The apparent viscosity of the slurries at 10, 20 and 30% concentration of coal are 0.0037, 0.0051 and 0.0069 Pa.s respectively. The fitting rheological parameters of the CWS at different solids' concentration are given in Table 4.2.

Table 4.2 rheological parameters of different coal water slurries.

Solids' concentration (w%/w)	Yield stress τ_0 (Pa)	Consistency index μ (Pa.s)/K (Pa.s ⁿ)	Flow index n	R ²
10	0	0.0037	1	0.9997
20	0	0.0051	1	0.9967
30	0	0.0069	1	0.9954
40	0	0.027	0.8988	0.9964
50	0.901	0.0571	0.8930	0.9942
60	6.015	0.3227	0.8864	0.9983

The consistency index of the slurry is observed to increase with concentration of coal dispersed in the water. Due to the change in the flow behavior of the slurry at 40% coal concentration, there is sudden increase in the consistency index of the slurry. At 50% a yield stress of 0.901 Pa is observed along with increased consistency index. The excessive overcrowding of the particles at 60% solids concentration, the yield stress and the consistency index are found to increase significantly. The flow index is found to be up to 30% concentration of the coal, indicating Newtonian nature of the slurry. However, the flow index is found to fall at higher concentrations of coal, indicating a shear thinning nature of the slurry.

4.3.2. Rheological characteristics of hydrothermally treated coals:

The rheological characteristics of the untreated along with HTT coals are presented in Figure 4.4. The solid concentration is varied from 10 to 60% by mass. From the rheological characteristics, it observed that all the slurries behave like a Newtonian fluid for solid concentration in the range of 10-30%. The apparent viscosity varies linearly with shear rate up to solid concentration of 30%. Whereas at 40% solid concentration, there is a significant change in the rheological characteristics of all the slurries. The apparent viscosity starts decreasing with shear rate. Such type of behavior is a characteristic property of pseudoplastic or shear thinning fluids. The apparent viscosity shows an increasing trend with solid concentration in untreated as well as HTT coal slurries.

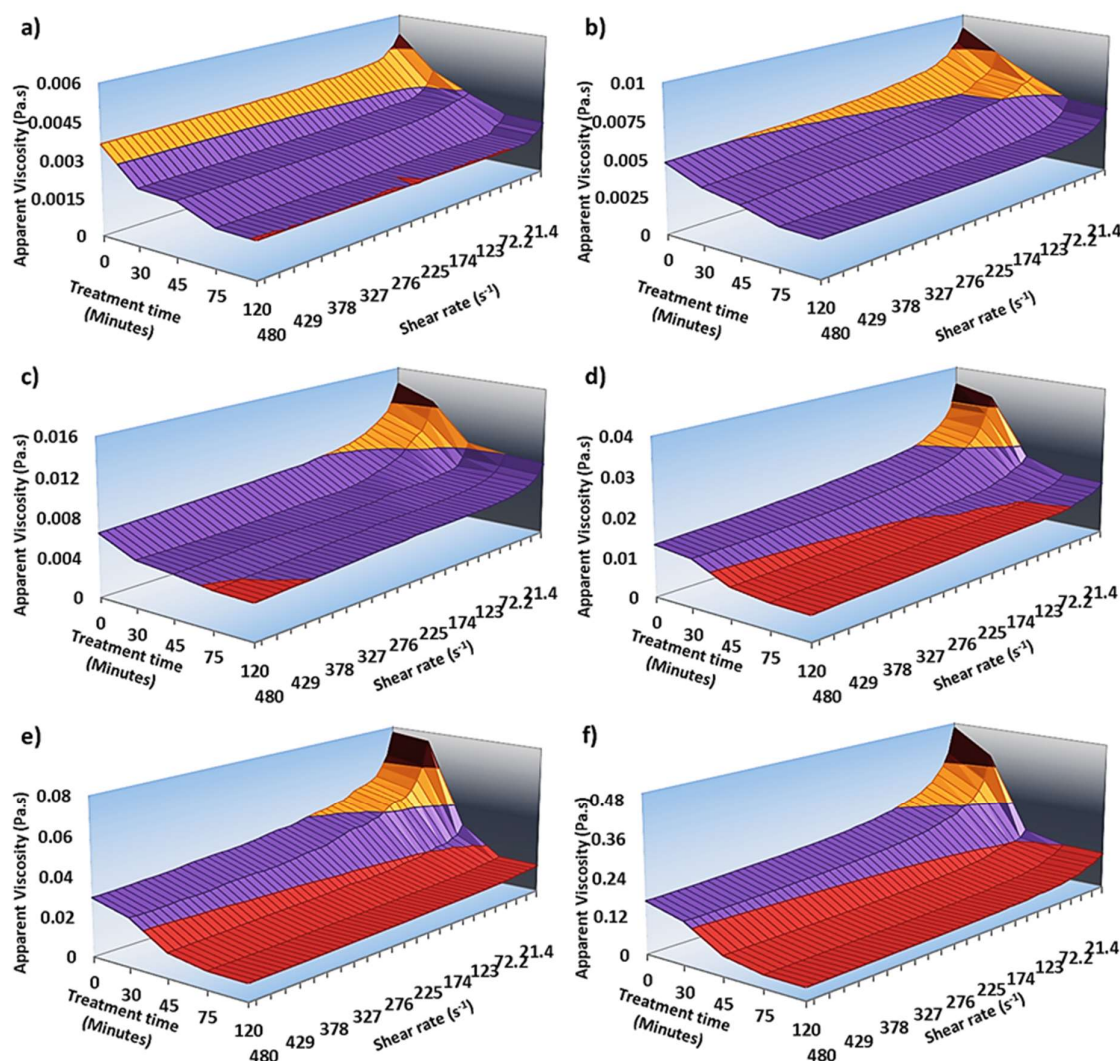


Figure 4.4 Rheological characteristics of slurries prepared from untreated and HTT coals at a) 10, b) 20, c) 30, d) 40, e) 50 and f) 60% coal concentration by mass.

The apparent viscosity is found to be decreasing with treatment time. There is a significant decrease in the apparent viscosity of the CWS after a treatment time of 30 minutes and this effect diminishes with treatment time. From the proximate analysis and EDS, it has been found that there is a significant decrease in the oxygen containing hydrophilic groups upon HTT with treatment time. The hydrophobicity of the coal increases with treatment time. Due to the increase in hydrophobicity, the water is unable to penetrate into the coal particle, as a result, there is a greater amount of water surrounding the particle. So, there is a reduction in the shear stress needed to deform the compact slurry structure. The apparent viscosity shows a decrease with treatment time. Similar results have been reported by the authors [61,65,128,177].

4.3.3. Rheological characteristics of microwaves-treated coal:

The rheology of the CWS prepared from untreated and treated coals is represented in Figure 4.5. All the mixtures are observed to behave like a Newtonian fluid in the case of solid concentrations ranging from 10-30% by mass. The slurries exhibit a pseudoplastic non-Newtonian behavior for solid concentration of 40-60% by mass. The MWT has a significant effect on the rheological characteristics of the CWSs. The rheological characteristics are significantly improved irrespective of the concentration.

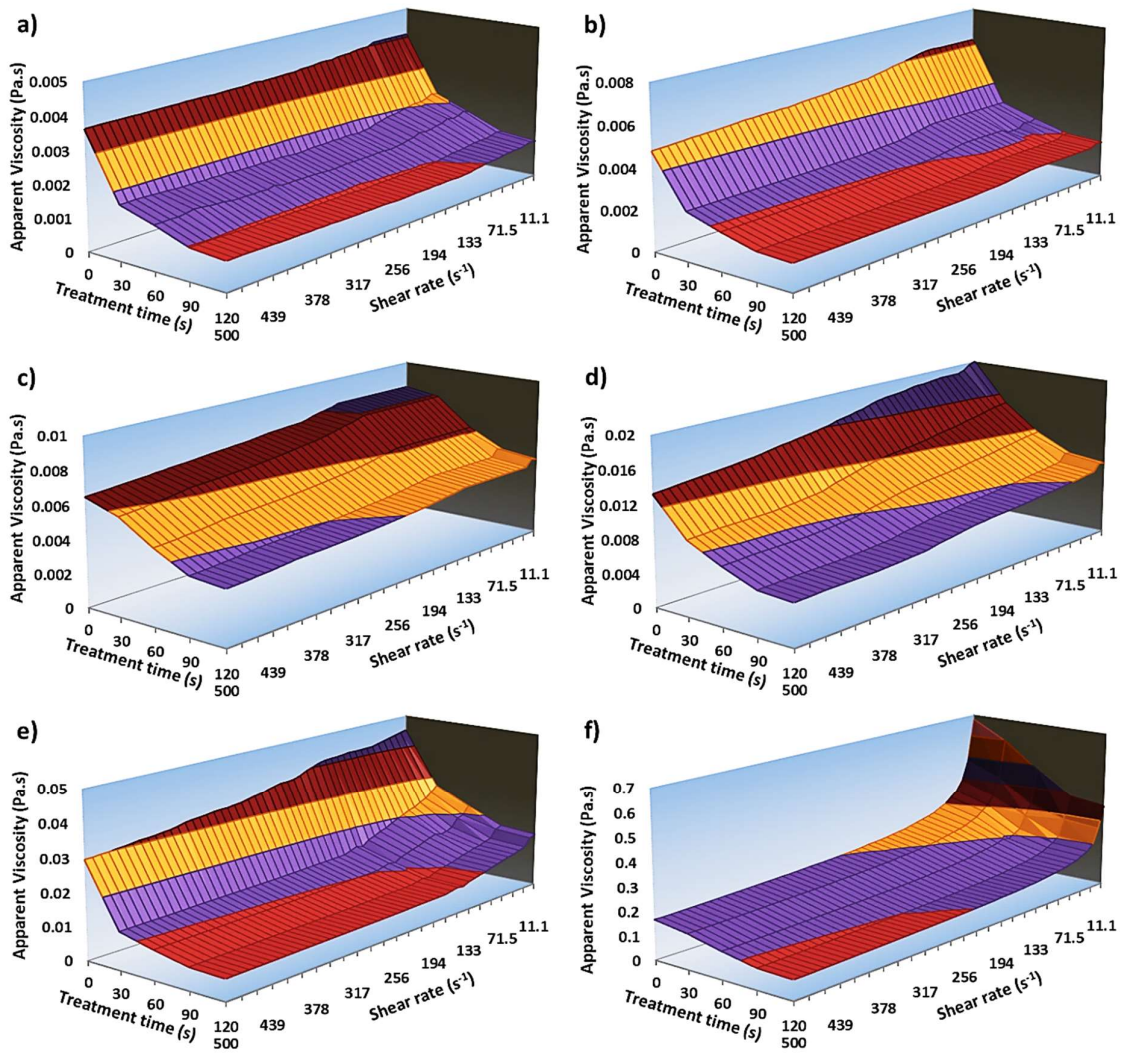


Figure 4.5 Rheological characteristics of untreated and treated coals at a) 10, b) 20, c) 30, d) 40, e) 50 and f) 60% solid concentration by mass.

The apparent viscosity is observed to be decreasing sharply for MWT30 sample at all the solid concentrations. This trend is observed to be diminishing with treatment time and the rheological characteristics of MWT90 and MWT120 coals are almost alike. The slurries having

solid concentration $\geq 40\%$ are observed to show a pseudoplastic behavior, and these slurries possess some yield stress. Yield stress characteristics is a very important property of any slurry to be transported through pipelines. When a prime mover tries to bring a static and stable slurry to motion (during system start-up), a significant amount of power is required to break the strong slurry structure. This power requirement is decided by the yield stress characteristics that can be obtained from the rheological data. If yield stress is reduced, then the starting and restarting power consumption can be greatly reduced. The MWT leads to the reduction in the yield stress and this is clearly indicative from the rheographs d), e) and f).

The aforementioned trends are observed due to the increase in the hydrophobicity of the treated coals. Due to the loss of polar oxygen functional groups, interactions between coal particles and the water molecules decrease so, greater amount of the water is available around the particles. Due to this the particles experience greater lubrication by the surrounding water, thereby reducing the drag among each other. On an average a maximum decrease of 75.55, 72.55, 46.23, 59.96, 77.51 and 49.66% at 10, 20, 30, 40, 50 and 60% solid concentration is observed. These values are observed for MWT120 coal sample.

4.3.4. Rheological characteristics of blended samples of fine (-53 μm) and coarse (-75+53 μm) coal samples:

Figure 4.6 shows the variation of the apparent viscosity with shear rate and particle size distribution (samples) for samples S₁, S₄-S₇. It is observed that the slurries of all the samples viz. S₁, S₄-S₇ behave like a Newtonian fluid up to the solid concentration of 30%, i.e. 10-30%; and a pseudoplastic behaviour is observed for the slurries of solid concentration ranging from 40-60%. When the coarse particles are added to the fine particulate slurry, the coarse particles break the compact structure of the solids by replacing the fine particles. This phenomenon is observed up to the 30% proportion of -75+53 μm sized coarse particles.

When the fraction of the coarse particles (sample S₂) is increased to 40%, the slurry becomes over crowded by the coarse particles and the fine particles start filling the voids between the coarse particles, leading to the formation of a compact structure, which requires a greater shear force to deform. The particle size distribution of sample S₆ can be considered optimum in the case of bi-modal slurries of -53 μm sized fine and -75+53 μm sized coarse coal particles as it has the lowest apparent viscosity in all the cases.

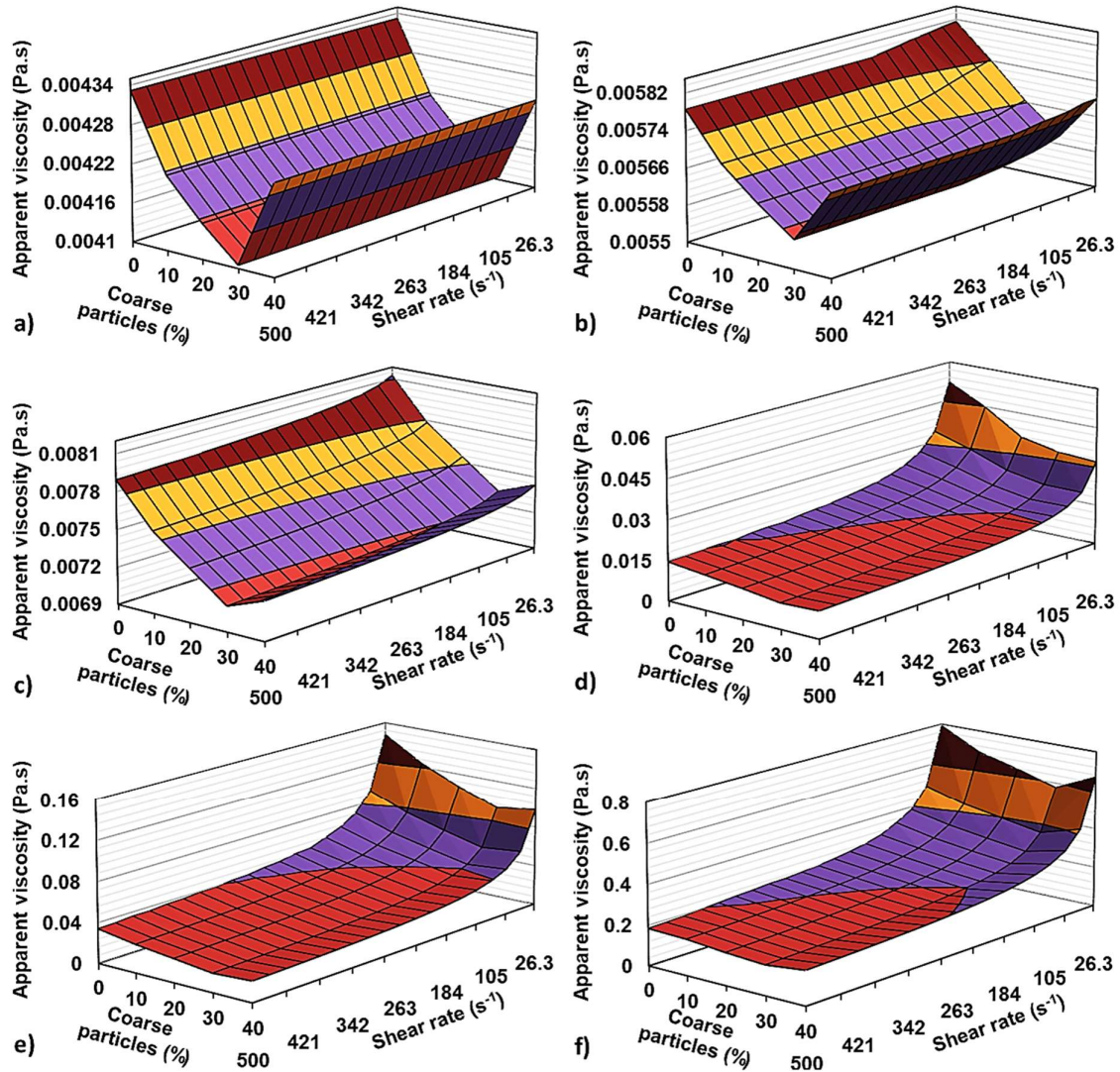


Figure 4.6 The rheological characteristics blended samples of the fine ($-53\ \mu\text{m}$) and coarse ($-75+53\ \mu\text{m}$) coal particles at a) 10, b) 20, c) 30, d) 40, e) 50 and f) 60% concentration of blended coal by mass

4.3.5. Rheological characteristics of blended samples of fine ($-53\ \mu\text{m}$) and coarse ($-106+75\ \mu\text{m}$) coal samples:

The rheological characteristics of the fine and blended samples S₈-S₁₁ at solids' concentration ranging from 10-60% are presented in Figure 4.7. It is observed that the slurries of the all the samples behave like a Newtonian fluid for solid concentration ranging from 10-30%. Whereas for solid concentration ranging from 40-60%, the slurries behave like a shear thinning or pseudoplastic fluid. The apparent viscosity of the blended samples is observed to decrease up to 20% fraction of coarse $-106+75\ \mu\text{m}$ sized particles.

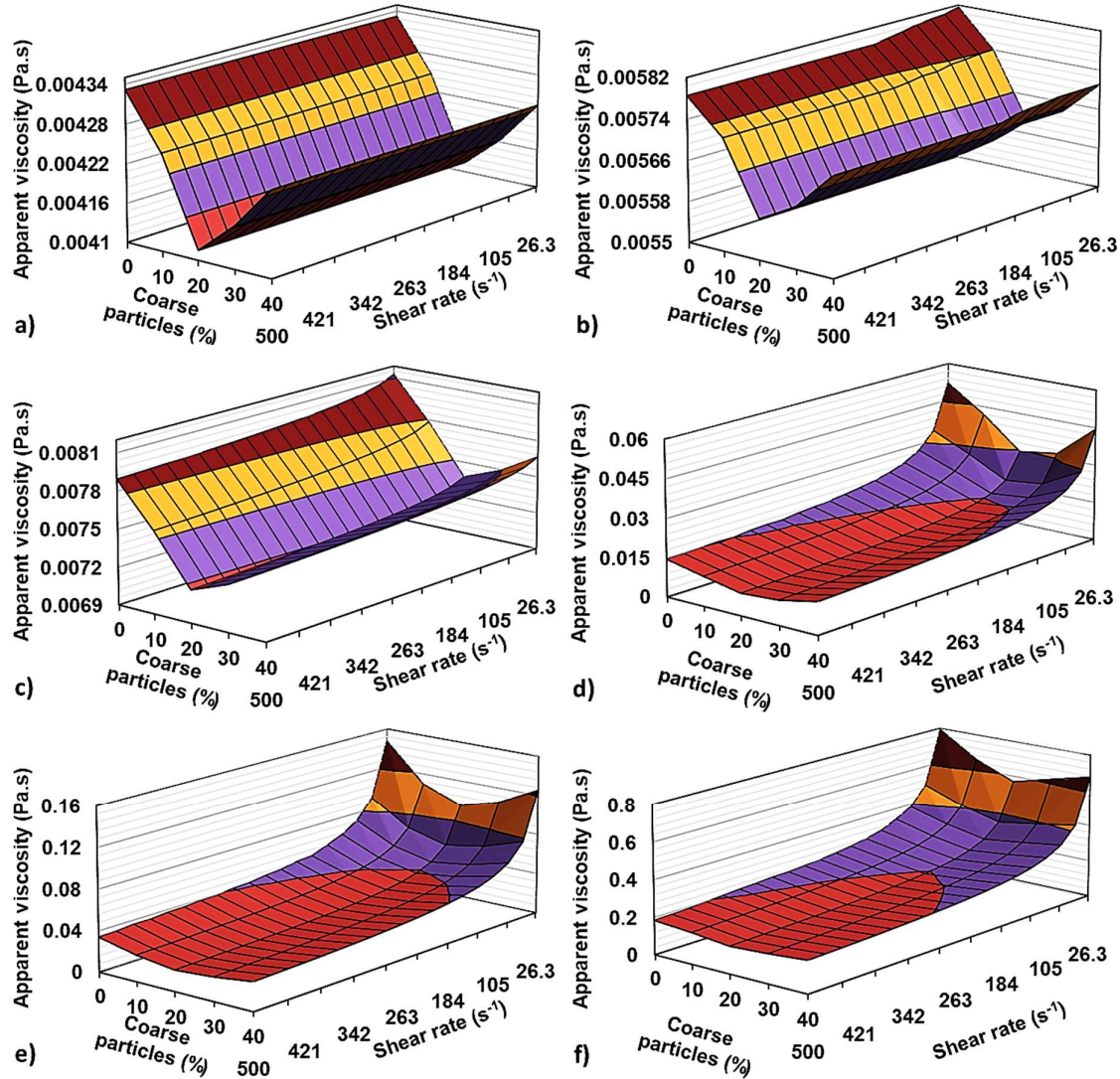


Figure 4.7 The rheological characteristics blended samples of the fine ($-53 \mu m$) and coarse ($-106+75 \mu m$) coal particles at a) 10, b) 20, c) 30, d) 40, e) 50 and f) 60% concentration of blended coal by mass

Similar to the previous investigation, the $-106+75 \mu m$ sized coarse particles are effectively able to break the compact slurry structure up to 20% proportion of (coarse) particles. The further increase in the fraction of coarse particles leads to an increase in the apparent viscosity of the slurry. The overcrowding of the coarse particles in this case is observed beyond 20% proportion of $-106+75 \mu m$ sized coarse particles, i.e. in the case of samples S_{10} & S_{11} , due to the greater size of the $-106+75 \mu m$ sized coarse particles. It can be concluded that the particle size distribution of the bi-modal sample S_9 is optimum in the case of bi-modal slurries of $-53 \mu m$ and $-106+75 \mu m$.

4.3.6. Rheological characteristics of blended samples of fine untreated and fine MWT coals:

The rheological characteristics of the fine ($-53\ \mu\text{m}$) untreated coal and its blended samples are presented in Figure 4.8. The fine ($-53\ \mu\text{m}$) coal is added in 10, 20, 30, 40 and 50% proportion by mass. The results indicate that the apparent viscosity of the untreated fine ($-53\ \mu\text{m}$) coal is significantly higher as compared to the untreated $-75\ \mu\text{m}$ sized coal. This is because the fine coal the particles are distributed over a narrow range of size, that makes a stronger slurry structure, that is even more difficult to break. However, in the case of $-75\ \mu\text{m}$ sized coal, the bigger particles help in breaking the compact slurry structure, thereby reducing the apparent viscosity of the slurry.

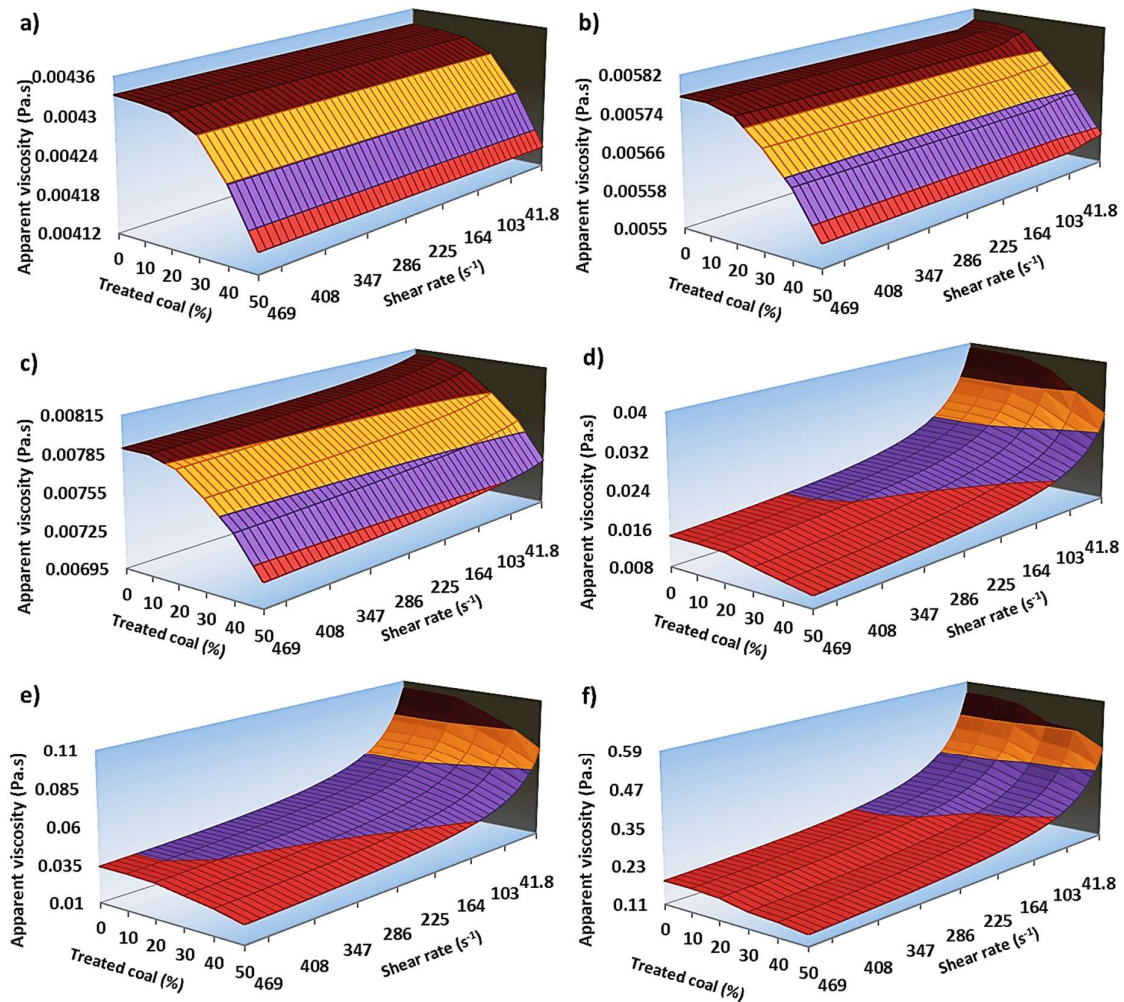


Figure 4.8 Rheological characteristics of untreated coal and the blended coals at a) 10, b) 20, c) 30, d) 40, e) 50 and f) 60% concentration of coal by mass.

All the blended samples behave like a Newtonian fluid at 10-30% solids' concentration. However, pseudoplastic behavior is exhibited at higher solids' concentration. The addition of fine MWT treated coal to the untreated coal leads to the improvement in the rheological characteristics of the blended sample. The addition of 10% treated coal slightly reduces the apparent viscosity of slurry, irrespective of the concentration of the coal dispersed in the water. The further addition of the treated coal has a magnifying effect in the reduction of the apparent viscosity. The relative improvement in the apparent viscosity is higher in the cases where greater fraction of treated coal is available. The treated coal is more hydrophobic as compared to the untreated one. As a result, the treated coal increases the water availability in the slurry. More, the fraction of the treated coal, more will be the water available in the particles' surroundings, that provides extra lubrication for the flow of the slurry.

As the addition of treated coal goes on reducing the apparent viscosity of the slurry, the optimum fraction of the treated coal can be limited to 40%. The excessive addition of treated coal may be economically expensive. Maximum reduction of apparent viscosity at 40% fraction of treated coal is observed to be 14.58, 20.57, 20.36, 8.16, 5.09 and 2.13% at 60, 50, 40, 30, 20 and 10% concentration of blended coal.

4.4 Concluding remarks:

The rheological characteristics of the different coal samples are investigated using the RheolabQC rheometer. The rheological characteristics of untreated coal, HTT coal and MWT coal are analysed to study the effect of thermal treatment on the slurryability and rheology of the treated samples. The fine particles' coal water slurry is desirable for gasification and combustion application, the effect of addition of a small fraction of coarse particles and the MWT fine coal, on the rheological characteristics of the blended sample have also been studied. Based on the results from all the above investigations, the following conclusions have been made:

1. The coal water slurry is a complex fluid that behave like a Newtonian fluid up to 30% concentration of the coal, while the flow behavior shifts to a shear thinning behavior at higher concentrations
2. The HTT leads to reduction in the apparent viscosity by a maximum of 60.1, 50.71, 39.9, 50.55, 65.69 and 75.41% at 10, 20, 30, 40, 50 and 60% solid concentration respectively.
3. The MWT also leads to reduction in apparent viscosity by a maximum of 75.55, 72.55, 46.23, 59.96, 77.51 and 50% at 10, 20, 30, 40, 50 and 60% solid concentration respectively

-
4. The addition of coarse particles in a suitable proportion leads to significant improvement in the rheological characteristics of the coal water slurry. These fractions are 30 and 20% by mass for -75+53 μm and -106+75 μm sized coarse particles respectively. The maximum reduction in the apparent viscosity is observed to be 43.2% and 41.4% in the case of addition of -75+53 μm and -106+75 μm sized coarse particles respectively.
 5. The addition of the MWT treated fine coal results in the overall improvement of the surface properties of the blended coal. The apparent viscosity is found to continuously decrease with increasing fraction of treated coal.

Chapter 5

Headloss characterization of coal water slurry

5.1 Introduction:

The design of a slurry pipeline is a challenging task and takes into account various parameters of the slurry that is to be transported. These parameters include the physico-chemical and rheological characteristics, minimum and maximum concentration of solids, hydrophobicity/hydrophilicity of the solids, particles' size and particle size distribution of the solids. The coal water slurry fuel has sophisticated applications such as gasification and combustion. The maximum permissible size in such applications is 106 μm however, it is desirable to have majority of the particles of fine ($\sim 53 \mu\text{m}$) size. These fine particulate slurries are usually associated with high headloss and power requirement. This is because the fine particulate slurries have high viscosity due to the formation of compact slurry structure. As observed in chapters 3 and 4, the addition of coarse particles results in the improvement of the stability and rheological characteristics of the coal water slurry prepared from the blended sample. So, the objective of present chapter is to investigate the headloss characteristic of the fine ($\sim 53 \mu\text{m}$) particulate coal water slurry experimentally. The study on effect of solids' concentration and the fraction of coarse coal particles on the headloss characteristics and specific energy consumption of the slurry pipeline is also the objective of the present chapter. The following methodology is followed to investigate the headloss characteristics of the coal water slurry:

5.2 Experimental procedures:

5.2.1 The pilot test loop:

The experimental investigation of the headloss characteristics of the coal water slurry is done on a 60m pilot test loop, installed at the fluid machinery laboratory at Thapar Institute of Engineering and Technology, Patiala. The pilot test loop consists of the parts mentioned in Table 5.1. In all of the present test experiments, a 60 m long 100mm in diameter, mild steel pipe having a density of 7850 kg/m^3 is used. The pilot test loop contains 6 bends of R/d of 1.5. The piezoelectric pressure transducers are used for the instantaneous measurement of gauge pressure at five different locations inside the pipeline. To account for the total flow and instantaneous flow rate through the pipeline, an electromagnetic flow meter, 100mm in diameter is used. The data obtained from the pressure transducers and the electromagnetic

flow meter is recorded and stored every minute. The backbone of the pilot test loop is a progressive cavity pump, that is a positive displacement pump. The pump traps some volume of slurry in the cavity and the rotating worm, delivers it further. This pump is driven by a 11.1kW 3-phase motor, by means of a reduction gear mechanism. The pump can easily deliver slurry at maximum flow rate of $60\text{m}^3/\text{h}$.

Table 5.1 Parts along with their specifications, used in the pilot test loop

Part name	Specifications
Pipe	100mm diameter; MS pipe
Pipe bends	6 No.s; $R/d=1.5$
Pressure transducers	5 No.s ; up to 10 bar
Data Acquisition panel	16 inputs ports; data recorded every minute
Electromagnetic flow meter	100mm diameter
Progressive cavity pump	Maximum flowrate of $60\text{m}^3/\text{h}$
Driving motor	11.1 kW 3-phase motor

The key components of the pilot loop are presented in schematic diagram represented by Figure 5.1.

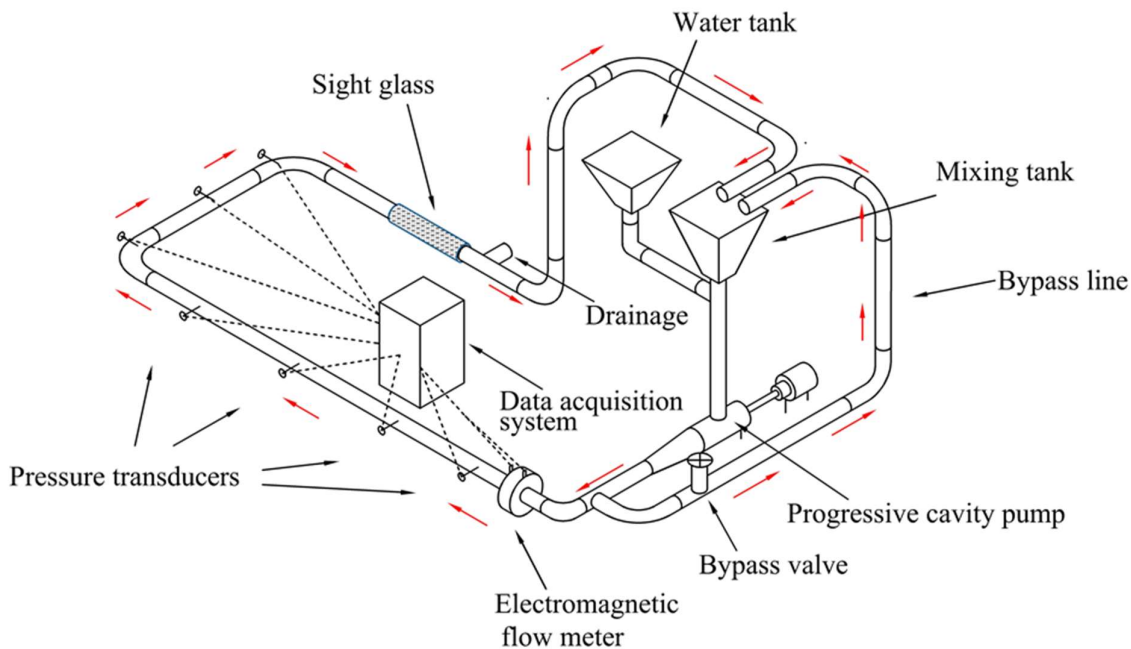


Figure 5.1 Schematic diagram of the pilot test loop

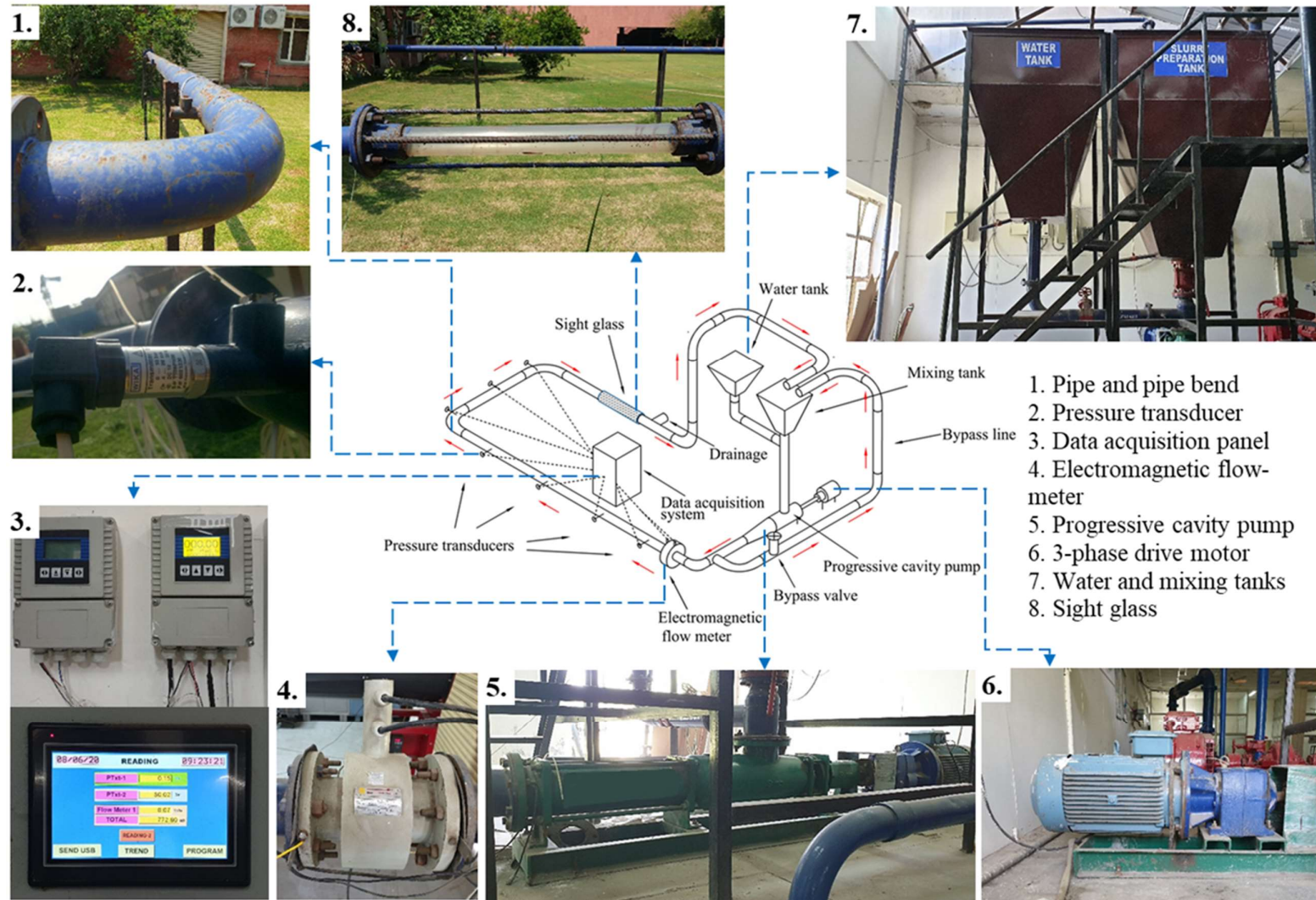


Figure 5.2 Actual photographs of the experimental pilot test loop

Figure 5.2 presents the actual photographs of the components of the pilot test loop. The slurry that is delivered by the pump is transferred back to the slurry preparation tank, as shown in Figure 5.2. The turbulence created by recirculating flow inside the slurry preparation tank ensures proper dispersion of the coal particles into water.

5.2.2 Samples and flow parameters:

The coal sample used initially in the present investigation is the $-53\ \mu\text{m}$ coal sample. The concentration of the coal sample is varied in the range of 10-60% by mass. The coarse particles of $-75+53\ \mu\text{m}$ and $-106+75\ \mu\text{m}$ are added in different proportions to investigate the effect of addition of the coarse particles on the headloss characteristics of the blended coal sample. The velocity of the flow of the slurry is varied from 0.25 to 1.25 m/s in steps of 0.25 m/s.

5.2.3 Preparation of the coal water slurry:

The calculated amount of water required for the preparation of the slurry is transferred to the slurry preparation tank and is allowed to circulate in the loop with the help of the pump. Then the calculated amount of the coal sample is gradually added to the circulating water. The slurry is circulated for at least 20 minutes to ensure uniformity in the composition of the slurry at any location of the test loop.

5.3 Headloss characteristics of slurry:

5.3.1 Headloss characteristics of fine $-53\ \mu\text{m}$ sized coal particles:

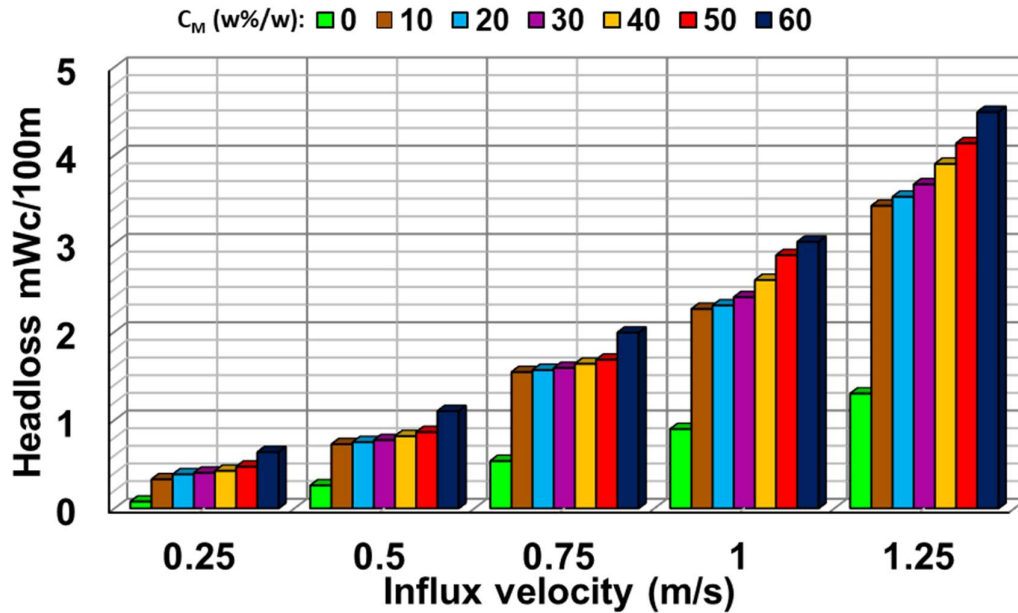


Figure 5.3 Headloss characteristics of water and fine $-53\ \mu\text{m}$ coal particulate slurry

The headloss characteristics of the fine (-53 μm) particulate coal water slurry are presented in Figure 5.3. The headloss characteristics of the fine particulate coal water slurry is also compared with those of normal tap water. The headloss of the water is found to increase with increase in the flow velocity from 0.25 to 1.25 m/s. The increase in the flow velocity is associated with increased shear stress as represented by equation 5.1 in the case of laminar flows. In the case of Newtonian fluids, the parameter μ_e is referred to as the dynamic viscosity of the water/slurry.

$$\tau_{WL} = \mu_e \frac{8u}{D} \quad 5.1$$

Similarly in the case of turbulent flows, the shear stress can be estimated as a function of Darcy's friction factor and the turbulent kinetic energy of the flow as presented in equation 5.2.

$$\tau_{WT} = \frac{f_d \rho u^2}{8} \quad 5.2$$

Where the Darcy's friction factor depends upon the experimental conditions and is found to be a function of Reynolds number, Re and other properties of the pipe such as pipe roughness, ε and diameter D . So,

$$f_d = f\left(Re, \frac{\varepsilon}{D}\right) \quad 5.3$$

$$h = \frac{4\tau_{W(L/T)}}{\rho g D} \quad 5.4$$

The increase in the wall shear stress contributes to the headloss as presented in equation 5.4. Similar results have also been obtained in the case of the flow of coal water slurry at 10 to 30% concentration of coal. However, at 40% concentration, the coal water slurry is no longer a Newtonian fluid. So, the effective viscosity μ_e in the case of power law fluid is estimated as per equation 5.5. Similarly, for the estimation of the Darcy's friction factor, the modified Reynolds number presented in equation 5.6 can be used.

$$\mu = K \left(\frac{1 + 3n}{4n} \right)^n \left(\frac{8V}{D} \right)^{n-1} \quad 5.5$$

$$Re = \frac{\rho V D}{K} \left(\frac{4n}{1 + 3n} \right)^n \left(\frac{D}{8V} \right)^{n-1} \quad 5.6$$

So, in all the cases, the wall shear stress is observed to increase with increase in flow velocity, that results into increase in the headloss. The addition of fine coal in water, leads to a

significant increase in the headloss. This is due to the significant modification and increase in the rheological characteristics of the water upon addition of coal. However, the consequent increase in the headloss with increase in solids concentration is monotonous up to 30% concentration of the coal. The equation 5.1 presents the dependence of the wall shear stress on the dynamic viscosity of the slurry. The increasing concentration of coal leads to increase in the dynamic viscosity and hence the wall shear stress. The headloss is observed to increase at higher rate at higher concentration due abrupt increase in the consistency index, 'K' of slurry and hence the wall shear stress as presented in equation 5.4. Due to the same reason, the headloss is found to increase suddenly at 60% concentration of coal, at all the velocity of the flow. The maximum headloss is observed in the case of 60% concentration of coal and the slurry flowing a 1.25 m/s.

The flow regimes of the different experiments are identified based on the Reynolds number of the flow. The slurry flow experiments are divided into two regimes viz. laminar and turbulent, based on the critical Reynolds number as proposed by Ryan and Johnson (1959) [178]:

$$Re_{cr} = \frac{6464n(n+2)^{(n+2)/(n+1)}}{(1+3n)^2} \quad 5.7$$

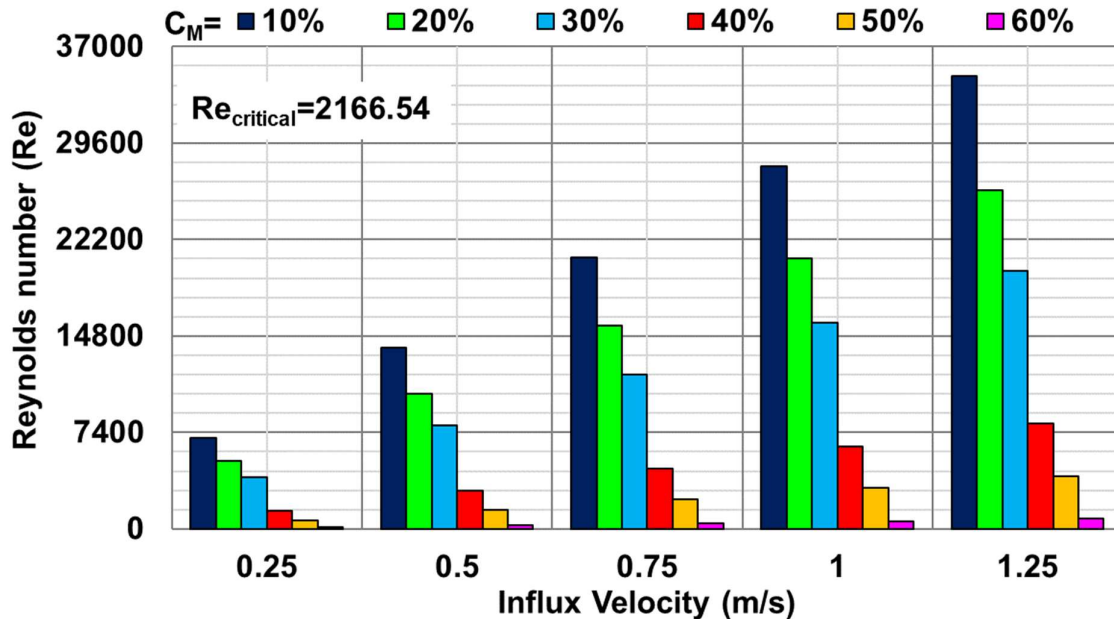


Figure 5.4 Reynolds number of the fine coal water slurry at different influx velocities

The Reynolds number of the flow is calculated using the equation 5.6. The Reynolds number is found to increase with increase in the influx velocity in all the cases. However, it is observed

to reduce with increase in the solid concentration. Based on the critical Reynolds number, the flow of coal water slurry at 10 to 30% solid concentration is purely turbulent, irrespective of the influx velocity of the flow. However, at low influx velocity of 0.25m/s at 40% solid concentration and 0.25-0.5 m/s at 50% solid concentration, the flow becomes laminar. The further increase in the influx velocity in both the cases leads to the transition of the flow from laminar to turbulent. The flow at 60% solid concentration is purely laminar.

5.3.2 Headloss characteristics of blended samples of fine (-53 μm) and coarse (-75+53 μm) coal samples:

The headloss characteristics of the blended coal samples of fine (-53 μm) and coarse (-75+53 μm) coals are presented in Figure 5.5. The headloss is observed to increase with flow velocity, in all the cases and all the concentrations of the coal. The addition of the -75+53 μm sized coarse coal particles leads to reduction in the headloss in the flow. This reducing trend is observed up to 30% proportion of coarse coal particles. The further addition of coarse coal leads to subsequent increase in the headloss in the flow.

When the coarse coal particles are added in small proportions ranging from 10 to 30%, they break the compact slurry structure created by the fine particles. Also, the stability of the blended samples is higher than the fine (-53 μm) coal, that reduces the particles settling. These factors prove to be favorable in the reduction of the headloss in the flow. The further addition of coarse particles is accompanied by significantly magnified drag acting on the coarse that retards the coarse particles, present in greater amount. The maximum reduction in the headloss is observed to be 55.96, 55.21, 53.6, 50.6, 46.69 and 45% at 10, 20, 30, 40, 50 and 60% concentration of the blended coal containing 30% proportion of coarse (-75+53 μm) particles.

The Reynolds number of the slurry at different influx conditions is presented in Figure 5.6. It is observed that the flow of the slurry at 10% solid concentration is highly turbulent, with Reynolds number of as high as 36900. With further increase in the solid concentration to 20 and 30%, the Reynolds number is observed to decrease, due to increase in the viscosity of slurry with solid concentration. At 40% solid concentration, the flow at low influx velocity of 0.25 m/s is found to be laminar for all the cases. However, the increase in the influx velocity leads to increase in the Reynolds number, that crosses the critical Reynolds number and appears to turbulent in nature.

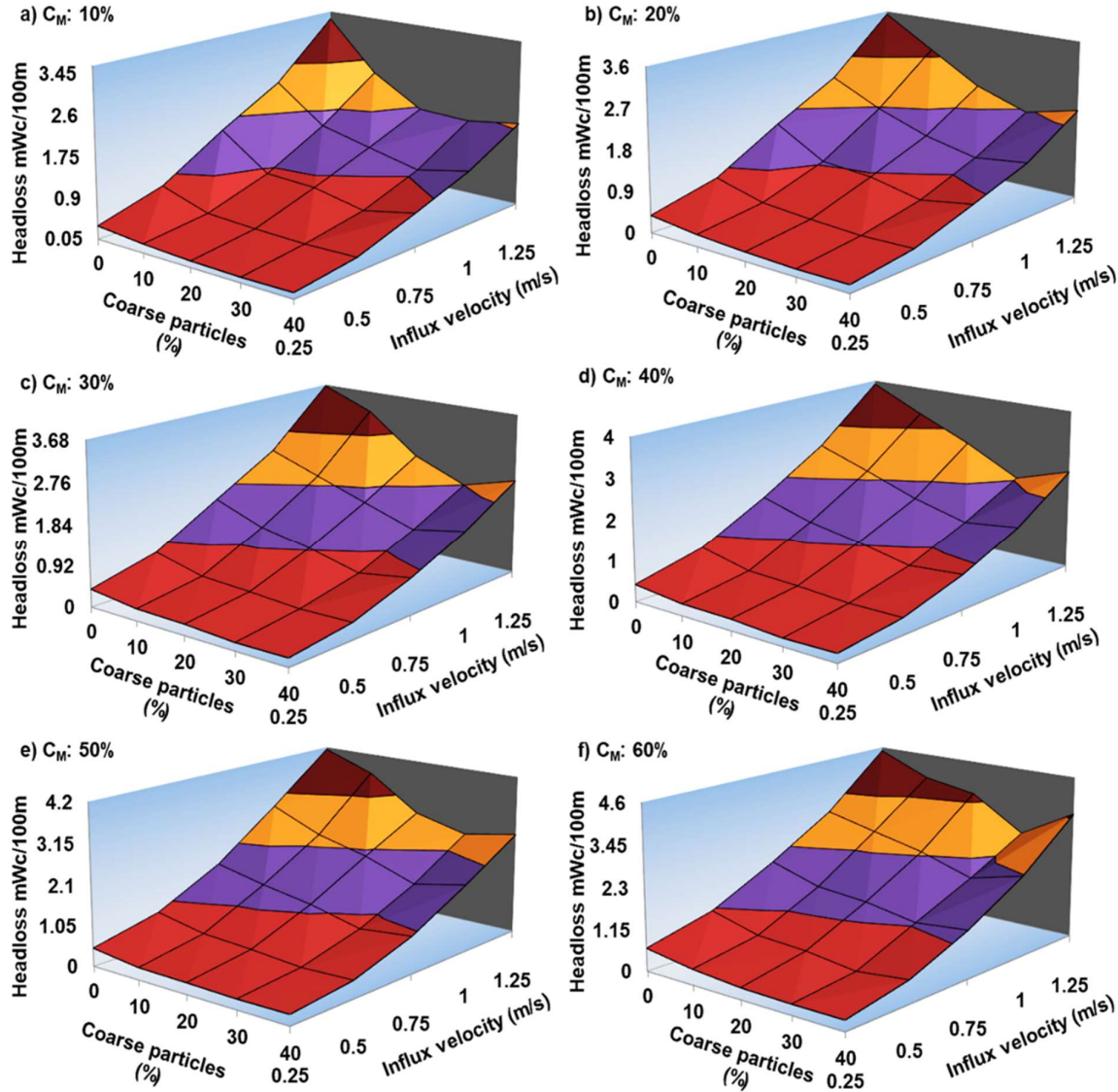


Figure 5.5 Headloss characteristics of slurry prepared from the blended samples of the fine ($-53 \mu\text{m}$) and coarse ($-75+53 \mu\text{m}$) coal particles at a) 10, b) 20, c) 30, d) 40, e) 50 and f) 60% concentration of blended coal by mass.

The flow of CWS at 40% solid concentration and higher velocities is turbulent. Similarly, at 50% solid concentration, and influx velocity of 0.25m/s, the flow is observed to be laminar in all the cases. At an influx velocity of 0.5m/s, the flow is observed to be laminar in almost all the cases, except in the case, where 30% coarse particles are added to the fine particulate slurry. The Reynolds number in this case is observed to be 2407, that crosses the critical mark and can be called as turbulent. The flow at influx velocities higher than 0.5m/s are turbulent in nature. At 60% solid concentration, due to abrupt increase in the dynamic viscosity of the slurry, the Reynolds number is observed to fall significantly. It is observed that the flow of CWS at 60% solid concentration is purely laminar at all influx velocities.

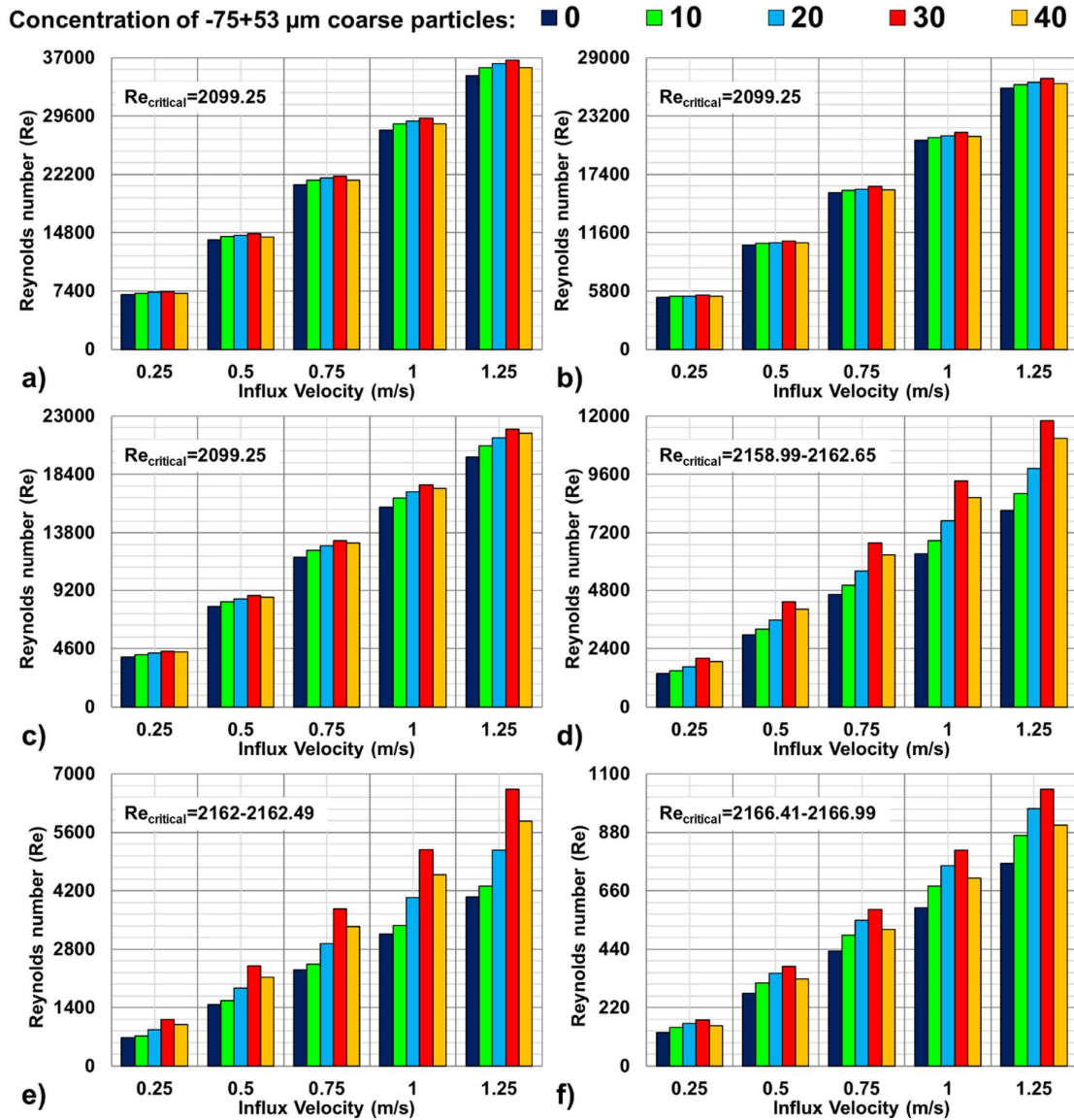


Figure 5.6 Reynolds number of slurry prepared from the blended samples of the fine (-53 μm) and coarse (-75+53 μm) coal particles at a) 10, b) 20, c) 30, d) 40, e) 50 and f) 60% concentration of blended coal by mass.

5.3.3 Headloss characteristics of blended samples of fine (-53 μm) and coarse (-106+75 μm) coal samples:

The headloss characteristics of the blended coal samples of fine (-53 μm) and coarse (-106+75 μm) coals are presented in Figure 5.7. The headloss in the flow is observed to increase with flow velocity, irrespective of the coal sample and concentration of coal sample. Similar to the previous case, the headloss is observed to decrease with addition of coarse coal particles. The reduction in headloss is witnessed up to 20% proportion of the coarse coal particles.

The addition of coarse coal particles in greater proportion leads to increase in the headloss. The reduction of headloss up to 20% proportion of coarse (-106+75 μ m) coal is due to the breakage of the compact slurry structure. But on subsequent addition of coarse coal particles, the slurry as a whole start experiencing additional drag, that is primarily due to presence of the coarse particles in great amount. The maximum reduction in the headloss is observed to be 55.96, 55.21, 53.6, 50.6, 46.69 and 45% at 10, 20, 30, 40, 50 and 60% concentration of the blended coal containing 30% proportion of coarse (-106+75 μ m) particles.

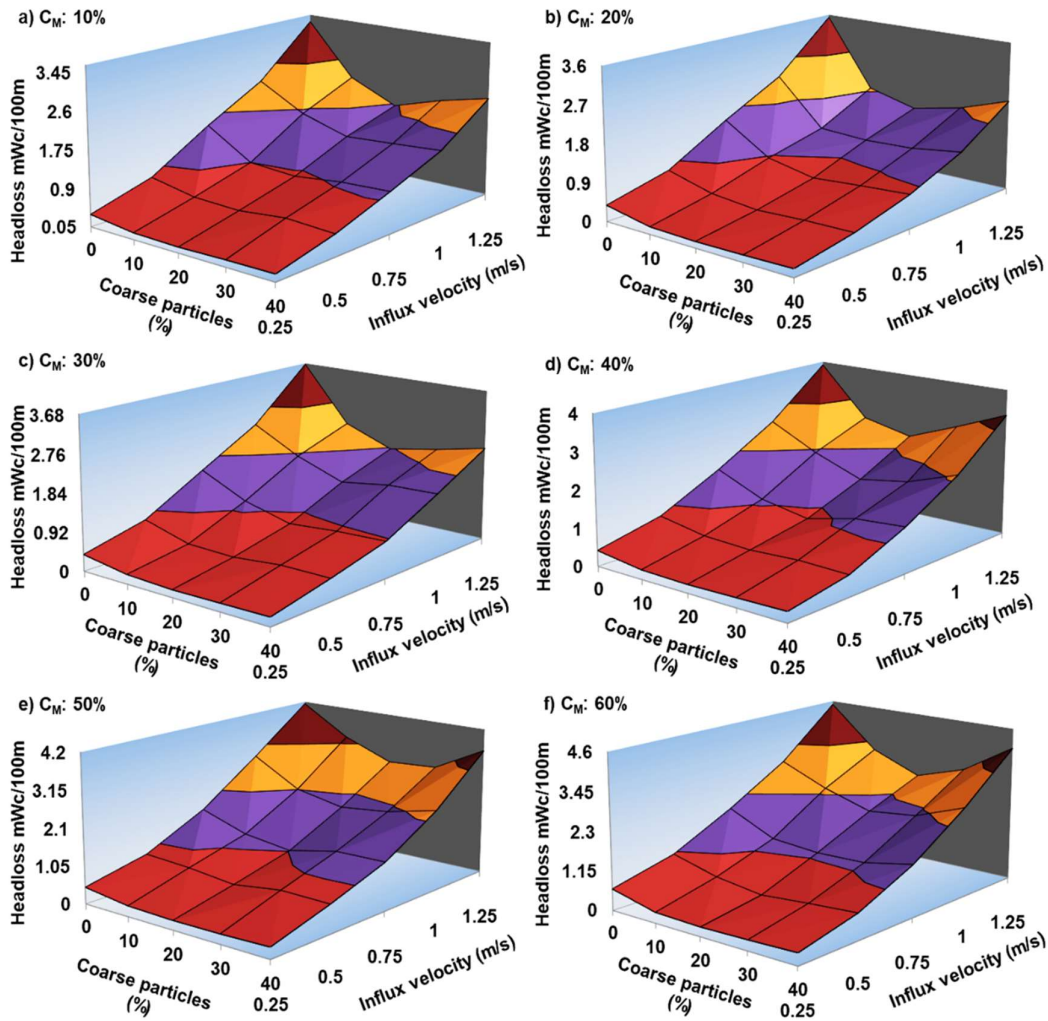


Figure 5.7 Headloss characteristics of slurry prepared from the blended samples of the fine (-53 μ m) and coarse (-106+75 μ m) coal particles at a) 10, b) 20, c) 30, d) 40, e) 50 and f) 60% concentration of blended coal by mass.

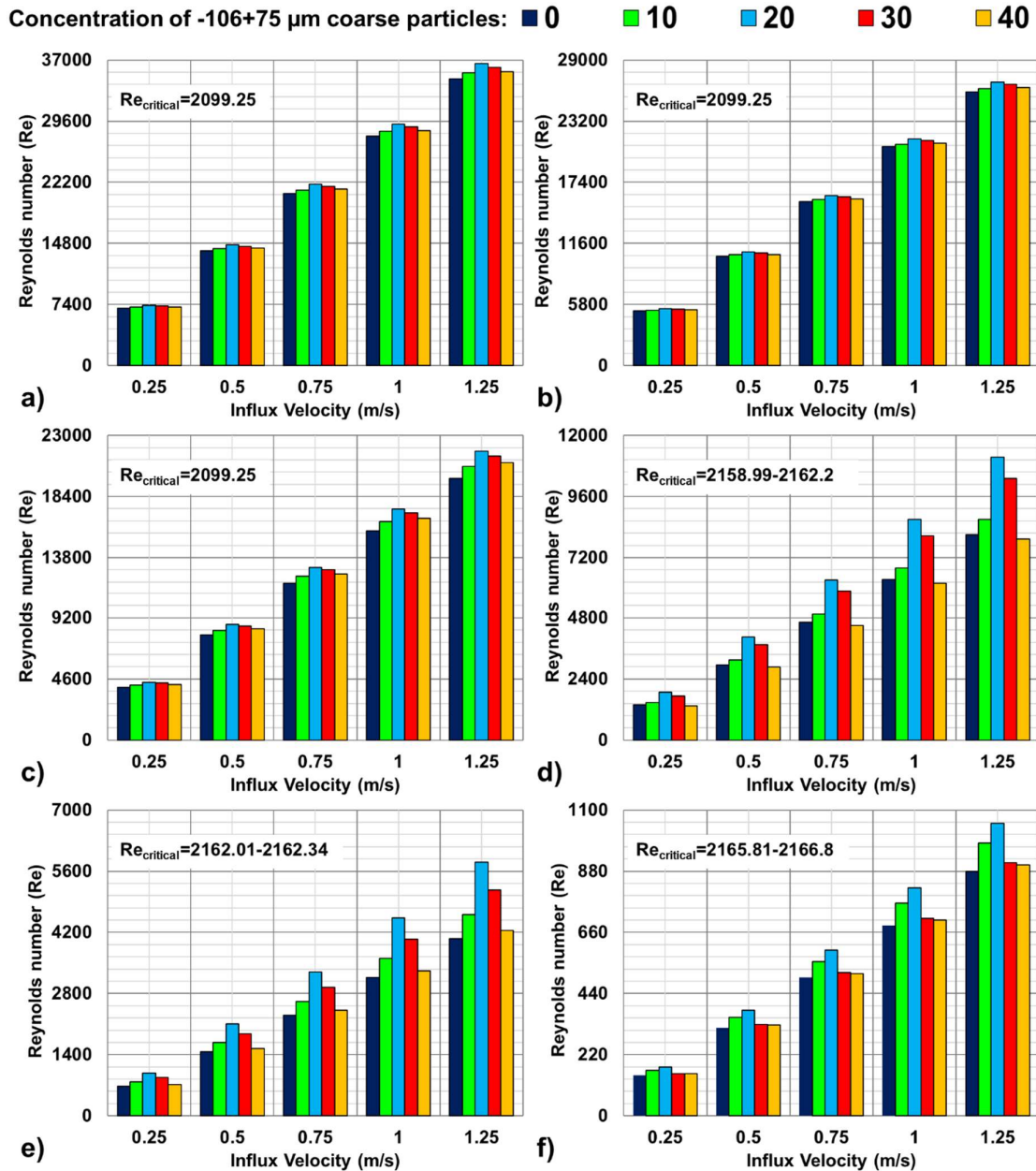


Figure 5.8 Reynolds number of slurry prepared from the blended samples of the fine (-53 μm) and coarse (-106+75 μm) coal particles at a) 10, b) 20, c) 30, d) 40, e) 50 and f) 60% concentration of blended coal by mass.

The Reynolds number of the slurry at different influx conditions is presented in Figure 5.8. It is observed that the flow of the slurry at 10% solid concentration is highly turbulent, with Reynolds number of as high as 36584. With further increase in the solid concentration to 20 and 30%, the Reynolds number is observed to decrease, due to increase in the viscosity of slurry with solid concentration. At 40% solid concentration, the flow at low influx velocity of

0.25 m/s is found to be laminar for all the cases. However, the increase in the influx velocity leads to increase in the Reynolds number, that crosses the critical Reynolds number and appears to turbulent in nature. The flow of CWS at 40% solid concentration and higher velocities is turbulent. Similarly, at 50% solid concentration, and influx velocity of 0.25m/s, the flow is observed to laminar in all the cases. At an influx velocity of 0.5m/s, the flow is observed to be laminar in all the cases. The flow at influx velocities higher than 0.5m/s are turbulent in nature. At 60% solid concentration, due to abrupt increase in the dynamic viscosity of the slurry, the Reynolds number is observed to fall significantly. It is observed that the flow of CWS at 60% solid concentration is purely laminar at all influx velocities.

5.4 Specific energy consumption of the slurry:

5.4.1 *Specific energy consumption of fine -53 μm sized coal particles:*

The specific energy consumption is a dimensionless parameter that has a great practical significance. Specific energy consumption is the measure of energy required to transport a unit weight of bulk solids by a unit distance. In the present study, the specific energy consumption is calculated using equation 5.8 as published in [179]:

$$SEC_w = \left| \frac{\Delta p}{\Delta l} \right| \times \frac{1}{\rho_s V_s g} \quad \text{Units} = \text{J/N} - \text{m} \quad 5.8$$

In practice, the mass is used to quantify the solids instead of the weight. To make the analysis more practical and easier to understand, the equation 5.8 has been modified to equation 5.9. The equation 5.9 presents the SEC as the measure of energy required to transport “a unit mass” of bulk solids by a unit distance. It can be seen that the SEC in terms of mass is “g” times the SEC in terms of weight.

$$SEC_w = \left| \frac{\Delta p}{\Delta l} \right| \times \frac{1}{\rho_s V_s} \quad \text{Units} = \text{J/kg} - \text{m} \quad 5.9$$

In equations 5.8 and 5.9, $\frac{\Delta p}{\Delta l}$ is the pressure loss per unit length and is measured in Pa/m, ρ_s is the density of coal in kg/m^3 , V_s is the volumetric concentration of the coal.

The specific energy consumption for the conveyance of the slurry at different concentrations of coal and at different flow velocities is presented in Figure 5.9. The SEC is observed to increase with flow velocity for all the cases. For a given concentration of coal, the

pressure drop, $\frac{\Delta p}{\Delta l}$ is observed to increase with velocity of flow. Due to this, the SEC is observed to increase with velocity of flow.

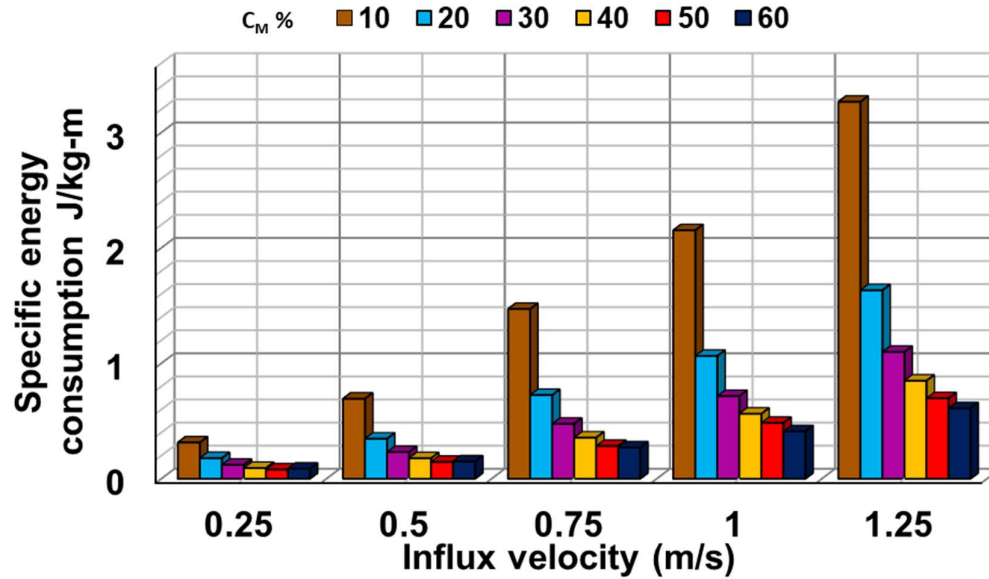


Figure 5.9 Specific energy consumption characteristics of water and fine -53 μm coal particulate slurry

Unlike the headloss, the specific energy consumption is observed to reduce with concentration. The lowest SEC is observed when the coal is transported at 60 concentration. For a given flow velocity, the pressure drop no doubt increases with increase in concentration of coal, but the relative increase in the denominator of equation 5.6 is higher, that leads to significant reduction in the SEC. Practically it can be said that the relative increase in headloss is small as compared to the increase in the mass of solids being transported. So, in spite of increased headloss at 60% concentration of coal, the same is a favourable option to convey coal with least possible SEC.

5.4.2 Specific energy consumption of blended samples of fine (-53 μm) and coarse (-75+53 μm) coal samples:

The specific energy consumption for the conveyance of the slurry of blended samples at different concentrations and at different flow velocities is presented in Figure 5.10

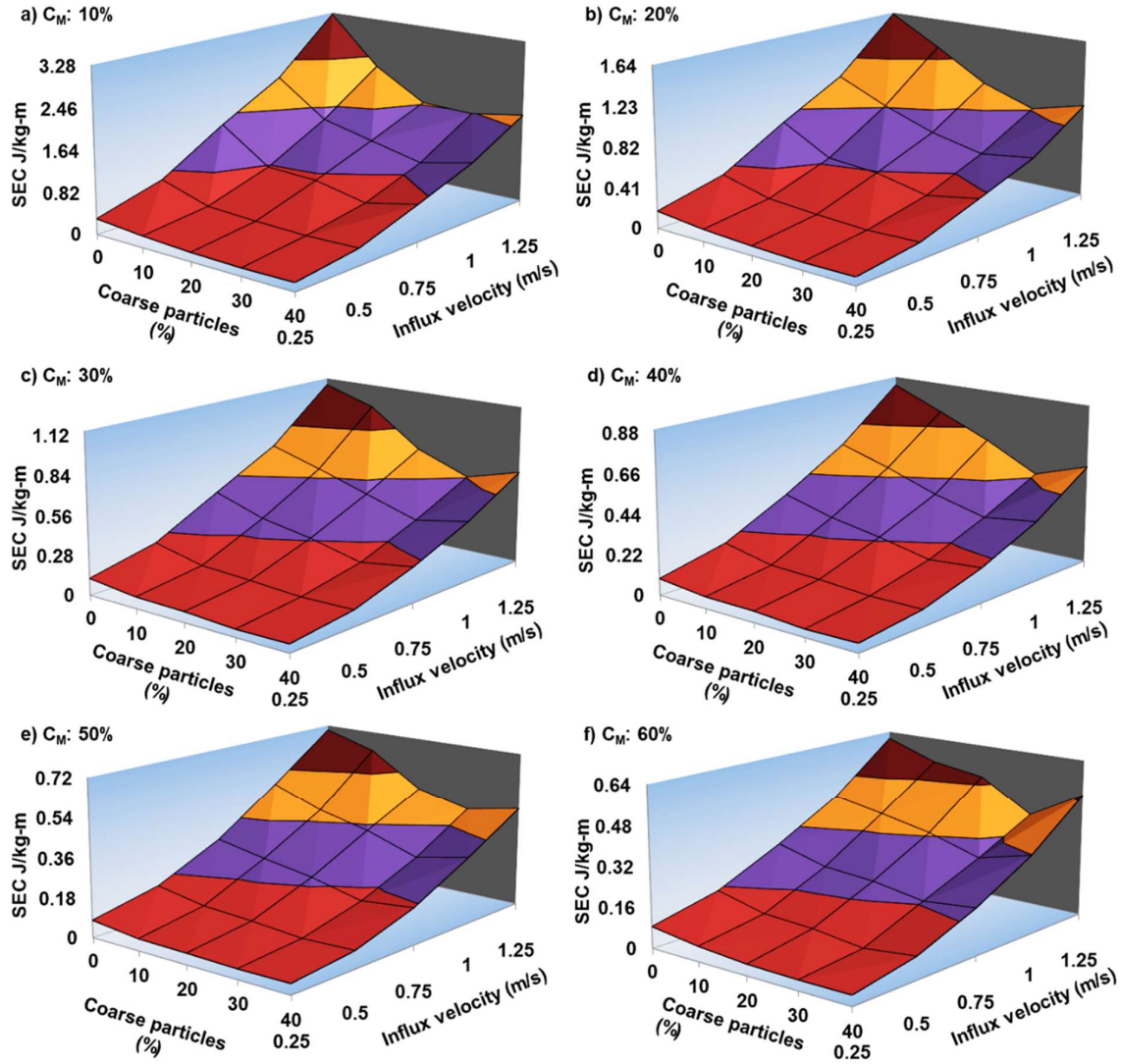


Figure 5.10 SEC characteristics of slurry prepared from the blended samples of the fine ($-53 \mu\text{m}$) and coarse ($-75+53 \mu\text{m}$) coal particles at a) 10, b) 20, c) 30, d) 40, e) 50 and f) 60% concentration of blended coal by mass

The surface plots of Figure 5.10 resemble those presented in Figure 5.5 as the SEC data is derived from the headloss data presented in Figure 5.5. The SEC is observed to increase with velocity of flow, irrespective of the concentration of coal. This is due to the increase in the pressure drop with flow velocity for a given concentration of coal. However, the SEC is observed to fall with concentration of coal. This is due to higher relative increase in mass of solids being transported at higher concentration. The addition of coarse ($-75+53 \mu\text{m}$) coal leads to reduction in the SEC for the conveyance of blended coal. The lowest SEC is observed in the case when 30% proportion of coarse ($-75+53 \mu\text{m}$) coal is added to the fine coal and conveyed at 60% concentration.

5.4.3 Specific energy consumption of blended samples of fine ($-53\ \mu\text{m}$) and coarse ($-106+75\ \mu\text{m}$) coal samples:

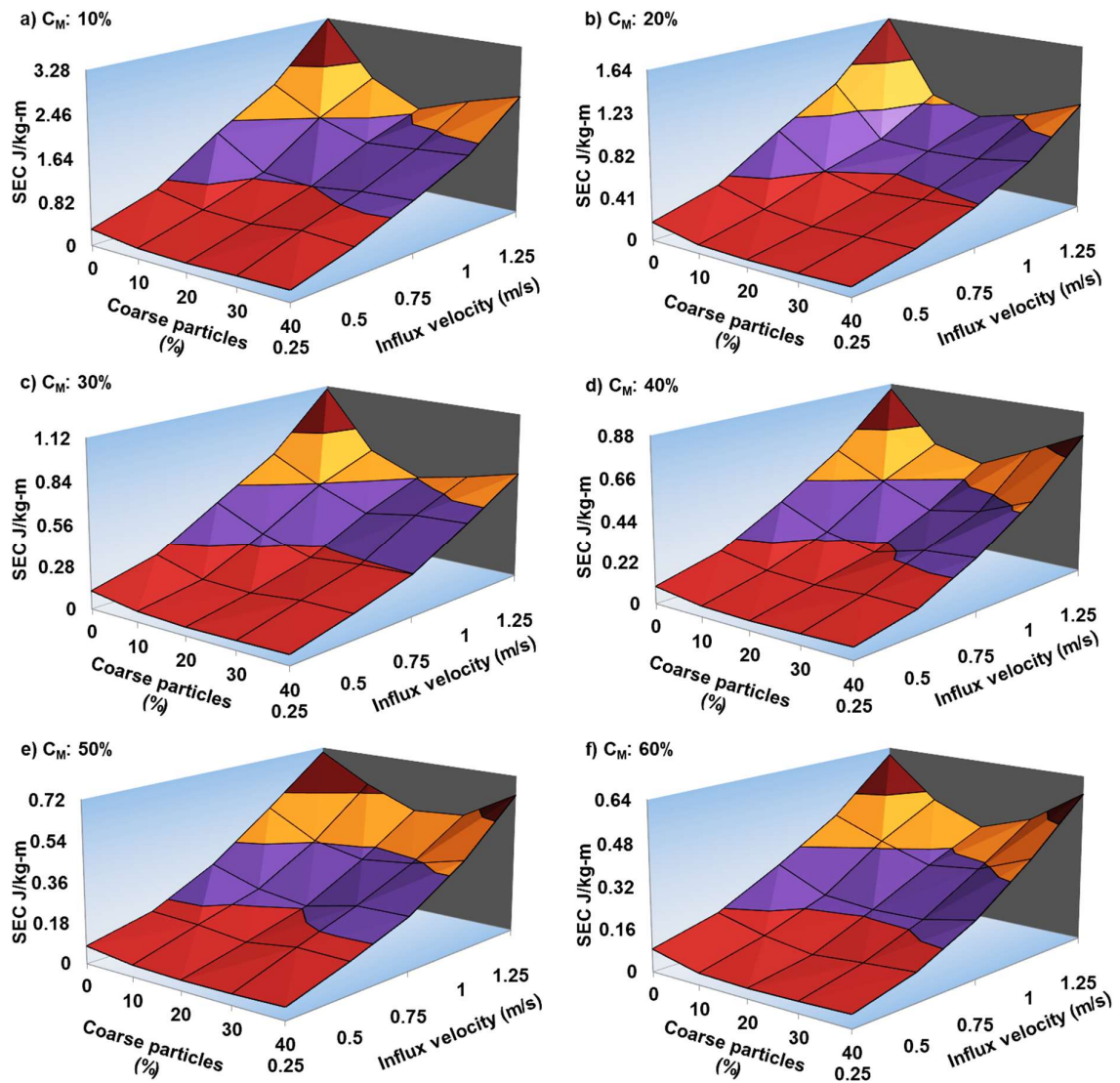


Figure 5.11 SEC characteristics of slurry prepared from the blended samples of the fine ($-53\ \mu\text{m}$) and coarse ($-106+75\ \mu\text{m}$) coal particles at a) 10, b) 20, c) 30, d) 40, e) 50 and f) 60% concentration of blended coal by mass

The specific energy consumption for the conveyance of the slurry of blended samples at different concentrations and at different flow velocities is presented in Figure 5.11. The SEC is observed to increase with flow velocity at all concentrations. The addition of coarse ($-106+75\ \mu\text{m}$) coal in fine coal leads to reduction of SEC only up to 20% proportion of coarse coal. This trend is in accordance to the headloss of the blended samples. The lowest SEC is observed

when 20% proportion of coarse (-106+75 μm) coal is added to the fine coal and conveyed at 60% concentration.

5.5 Comparison of the experimental results with some existing fine-slurry models:

Efforts have been made to compare the experimental headloss results with some previously proposed theoretical models. These models are:

5.5.1 The Irvine (1988) model:

Irvine et al. [180] in 1988, proposed a new generalized approach to estimate the fanning friction factor for turbulent flow of pseudoplastic fluids. In this approach, the fanning friction factor is estimated as:

$$f_n = \frac{F'(n)}{Re^{[1/(3n+1)]}} \quad 5.10$$

Here, $F'(n)$ is a function of flow index, n , that is computed from the rheology of the CWS at given conditions. The Reynolds number, Re is calculated using equation 5.6. The function $F'(n)$ is estimated using the following equation:

$$F'(n) = \frac{2}{8^{n-1}} \left(\frac{8n^n}{77n(1+3n)^n} \right)^{1/(3n+1)} \quad 5.11$$

The Darcy's friction factor is estimated using the fanning friction factor as:

$$f_d = 4f_n \quad 5.12$$

For laminar flow conditions such as at 60% influx concentration, the Darcy's friction factor is estimated as per equation 5.13

$$f_d = \frac{64}{Re} \quad 5.13$$

The above model is used to predict the headloss characteristics of fine (-53 μm) coal slurries, slurries of blended samples of the fine (-53 μm) and coarse (-75+53 μm) coal particles and slurries of blended samples of the fine (-53 μm) and coarse (-106+75 μm) coal particles. The results obtained using the above model are presented in Figure 5.12. The results predicted by the Irvine (1988) model are in moderate to good agreement with the experimental results. The statistical analysis of the results shows that around 87.23% of the results fall in the confidence interval as the R^2 value comes out to be 0.8723.

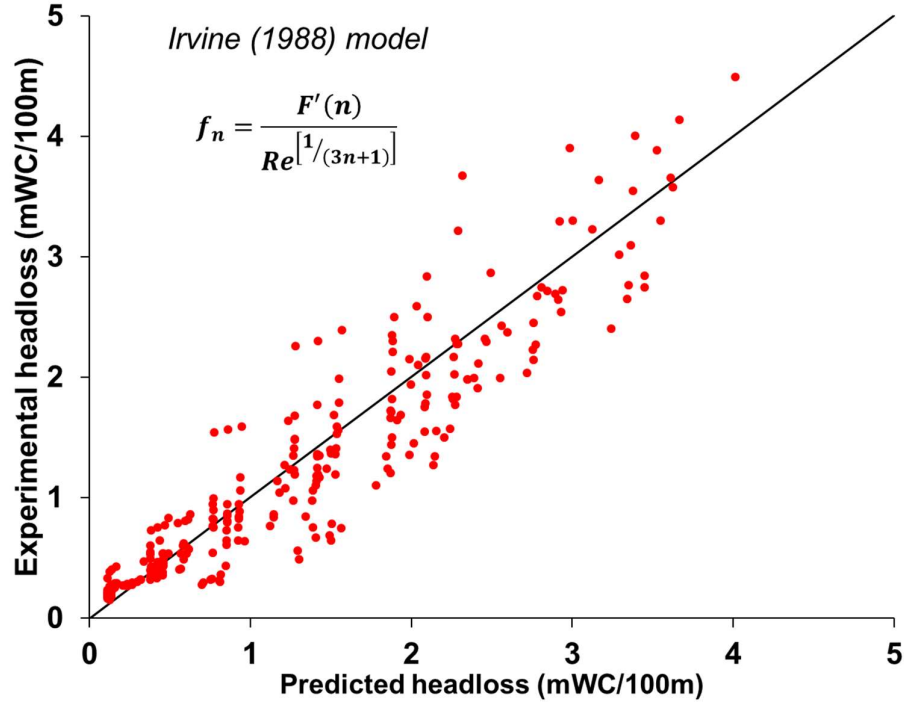


Figure 5.12 Comparison of the predicted headloss (Irvine (1988) model) with experimental results

5.5.2 The Wood (1966) model, modified by Mukhtar (1991):

Wood [181] in 1966 proposed an empirical approach to estimate the Fanning friction factor for turbulent flow as:

$$f_n = (a + b \cdot Re^{-c}) \quad 5.14$$

The parameters a , b and c are estimated using the roughness of the pipe material, K_R and particle diameter, d_s . These are calculated as:

$$a = 0.026 \left(\frac{K_R}{d_s} \right)^{0.225} + 0.133 \left(\frac{K_R}{d_s} \right) \quad 5.15$$

$$b = 22 \left(\frac{K_R}{d_s} \right)^{0.44} \quad 5.16$$

$$c = 1.62 \left(\frac{K_R}{d_s} \right)^{0.134} \quad 5.17$$

However, in 1991, Mukhtar [182] proposed a small modification in the estimation of the Fanning friction factor as:

$$f_n = (a + b \cdot Re^{-c}) \cdot (1 - 0.33 \cdot C_M) \quad 5.18$$

Here, C_M is the concentration of the particles by mass in the slurry.

The above model is also used to predict the headloss characteristics of fine (-53 μ m) coal slurries, slurries of blended samples of the fine (-53 μ m) and coarse (-75+53 μ m) coal particles and slurries of blended samples of the fine (-53 μ m) and coarse (-106+75 μ m) coal particles. The results obtained using the above model are presented in Figure 5.13.

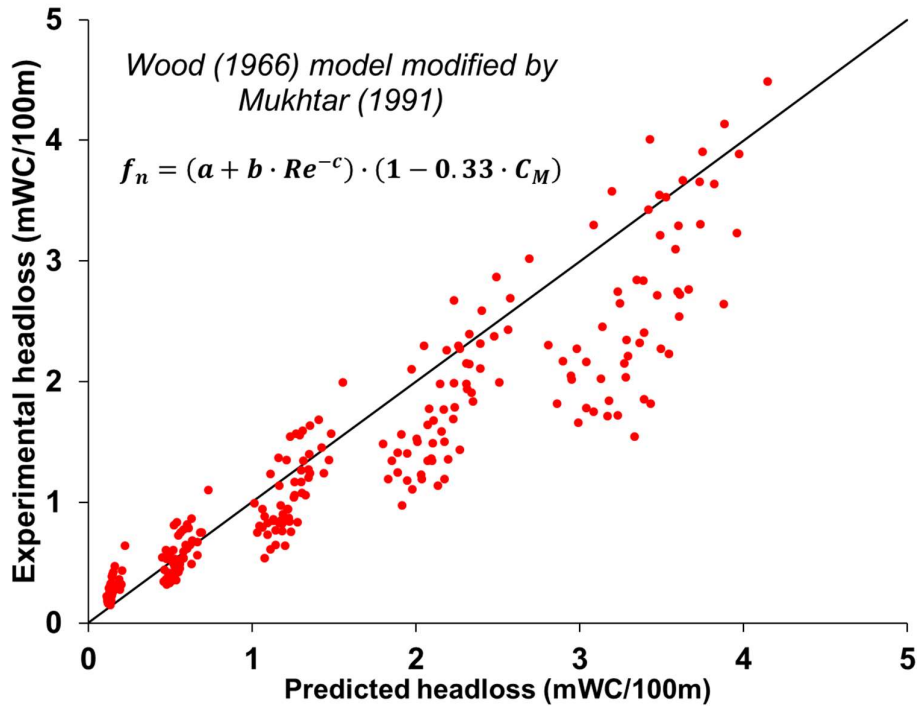


Figure 5.13 Comparison of the predicted headloss (Modified Wood (1966) model) with experimental results

The results predicted by the modified Wood (1966) model are in moderate agreement with the experimental results. This model is observed to mostly overpredict the results. The improper inclusion of the pipe surface roughness into the model might be reason for the overprediction of the results. Around 80% of the results predicted by this model lie in confidence interval as the R^2 value for this set of statistical data is found to be 0.803

The Irvine (1988) model shows better agreement with the experimental results as compared to the Modified Wood (1966) model.

5.6 Concluding remarks:

The headloss and SEC characteristics of the fine and the blended sample are investigated using the 60m pilot test loop. The headloss of the fine and the blended samples are analysed to study the effect of addition of coarse particles on the headloss of the flow. Based on the results obtained from the pilot test loop, the following conclusions have been made:

1. The addition of coal to the water makes it a complex fluid, as a result the headloss of the flow increases to 2.7 to 3.6 times of the headloss for water.
2. The headloss is observed to rise non linearly with influx velocity.
3. The addition of -75+53 μm particles in 30% proportion to the fine particulate slurry leads to reduction in the headloss by 45% (C_M : 60%) to 55.9% (C_M : 10%). Similarly, the addition of -106+75 μm in 20% proportion leads to reduction in the headloss by 42.3% (C_M : 60%) to 55.7% (C_M : 20%)
4. The SEC is observed to decrease by around 78%, when the concentration of coal in the slurry is increased from 10 to 60%.
5. The -75+53 μm particles in 30% proportion and -106+75 μm in 20% proportion leads to the maximum reduction in the SEC.
6. The Irvine (1988) model shows better agreement with the experimental results as compared to the Modified Wood (1966) model.

Chapter 6

CFD modelling of coal water slurry flow through pipe fittings

6.1 Introduction

The computational fluid dynamics tool has been extensively used to predict the dynamics of the single and multiphase fluid systems with high accuracy. In the earlier ages of its development, the two-layer model was widely used to model a variety of flow problems [183,184]. But the further in-depth studies indicated some error associated to the near wall flow formulation. To address this issue additional term was suggested, that accurately predicts the flow in the near wall region. This flow is then blended with the developed turbulent flow away from the wall using a blending function that is represented as:

$$u^+ = e^\Gamma u_{lam}^+ + e^{\frac{1}{\Gamma}} u_{turb}^+ \quad 6.1$$

Here u^+ is the dimensionless velocity, and expressed as the ratio of velocity perpendicular to the wall and the frictional velocity. u_{lam}^+ is the dimensionless velocity in the viscous sublayer, that is located adjacent to the wall. The velocity varies linearly with the dimensionless wall distance, y^+ in this region. u_{turb}^+ is the dimensionless velocity in the logarithmic region, where the velocity varies in a logarithmic way. This region is located adjacent to the developed free stream. In addition to these two regions, there is a third region, known as the buffer layer, where the velocity distribution is neither linear, nor logarithmic. This region can be considered as the transition region from the linear velocity distribution to the logarithmic velocity distribution. To closely model the velocity distribution in the buffer layer, the blending is achieved using equation 6.2. The blending function, Γ can be expressed as:

$$\Gamma = -\frac{a(y^+)^4}{1 + by^+} \quad 6.2$$

The constants a and b have values 0.01 and 5 respectively. This approach enhances the validation of the two-layer model throughout the flow domain. The dispersed phase of two-phase system can be formulated using two approaches viz. Euler approach or Lagrange while the primary phase or the carrier medium is formulated using Euler approach. Both

these approaches can be effectively used to predict the dynamics of the flow, while these are usually selected based on the application and the experimental conditions.

The selection of the modelling scheme is mostly based on the type to regime, of the slurry. So, the slurry can be either homogenous or heterogenous. For homogenous suspensions such as salt-water, sugar-water, etc. the best modelling scheme is the Mixture model. But in case the suspension/slurry is heterogenous, such as sand-water, coal-water, ash-water, sand-oil, etc. Eulerian model is used for the carrier medium for sure. However, to track the particles, there are further two options viz: Granular sub-model in the Eulerian model or the Lagrange model. Both these models provide accurate results but their accuracy is dependent on the parameters such as phase/density of the carrier fluid and solids' concentration. The Lagrangian model is preferred where, the presence of the solid phase does not influence the trajectory of the carrier medium and this is possible only if the density of the carrier medium is very small as compared to the solids being transported, such as pneumatic conveying of the solid particles or the in the case where the concentration of the solid phase is very small in a carrier medium of comparable density such as in case of very dilute solid-liquid slurries. In addition, the Lagrangian modelling can be attractive where primary focus is on the study of the microscopic details of the flow. The Euler-Euler approach when used for dense slurries is very stable as the equations for both the phases are solved simultaneously, while in Euler-Lagrange approach, first the trajectory of the carrier medium is traced and accordingly based on the interactions of the particles with the continuous phase, the particles' trajectory is traced so, sometimes the solution shows fluctuations in values.

The elbow, converging section and diverging section, are important pipe fittings, usually associated with a significant unsteadiness in the flow of the fluid as compared to a similar parallel pipe. This is due to the formation of eddies and separation of flow in some cases. These are usually linked to a significant head-loss and consequently energy loss as compared to a parallel pipe.

The design of these fittings comprises of some important factors such as pressure recovery coefficient, head-loss across the section, friction factor and volumetric efficiency. Each parameter has its importance, which can be seen in detail in the present study.

6.2 Numerical Model for a horizontal pipe:

6.2.1 The Euler-Euler approach:

The Euler modelling approach is used to model a fixed control volume. The modelling scheme can be steady-state or transient, depending upon the considered case. In the present study, the steady-state scheme is followed. The present research involves modelling two immiscible phases, that constitute a heterogenous system. For this Euler-Euler approach has been used to model both the phases viz. liquid and solid. The liquid phase is considered as the primary phase and the solid phase is considered as the secondary phase. To consider the particle-fluid interaction forces and inter particle forces, the secondary phase is selected to be granular in nature.

At the inlet, the both the phases have same velocity and the secondary phase is homogenously dispersed in the liquid phase. The liquid phase experiences a no-slip condition near the wall. The Euler model is based on the solution of three interdependent systems of equations viz. continuity equations, momentum balance equations and the turbulence scheme. In the present research, four different turbulence schemes are used to get the preliminary results for the validation purpose. These models are Standard $k - \epsilon$ with enhanced wall treatment, Standard $k - \epsilon$ with scalable wall functions, SST $k - \omega$, and Standard $k - \omega$. In a following section, it will be seen that the SST $k - \omega$ model shows the best agreement to the previously published experimental results.

The Euler model used in the present research is described as follows:

6.2.1.1 Continuity equation:

As discussed earlier, the continuity equations are solved for each phase separately and simultaneously. The equation is:

$$\nabla \cdot (\rho_i V_i u_i) = 0 \quad 6.3$$

ρ_i , V_i and u_i represents the density, volumetric concentration and velocity of either phase respectively and i can be either phase viz. water or coal.

6.2.1.2 Momentum balance equation:

The momentum equation considers all the forces originating due to fluid-wall, fluid particle, particle-wall and particle-particle interactions.

For either phase, the momentum equation takes the form:

$$\nabla \cdot (\rho_i V_i \vec{u}_i \vec{u}_i) = -V_i \nabla P_t + F_i \quad 6.4$$

P_t is thermodynamic pressure and F_i is the collective term for the forces acting on either phase.

The forces acting on the primary phase “P”, F_P is represented as:

$$\begin{aligned} F_P = & \underbrace{\rho_P V_P \vec{g}}_{\text{Body force}} + \underbrace{\nabla \cdot \tau_P}_{\text{Viscous force}} + \underbrace{C_{SP}(\vec{u}_S - \vec{u}_P)}_{\text{Inter-phase drag force}} + \underbrace{C_{vm} V_P \rho_P (\vec{u}_S \cdot \nabla \vec{u}_S - \vec{u}_P \cdot \nabla \vec{u}_P)}_{\text{Virtual mass force}} \\ & + \underbrace{C_L V_S \rho_P (\vec{u}_S - \vec{u}_P) \times (\nabla \times \vec{u}_P)}_{\text{Lift force}} \end{aligned} \quad 6.5$$

Where $\vec{g}$ is acceleration due to gravity and τ_P , liquid phase stress tensor is described as:

$$\tau_P = \mu_P [\nabla \vec{u}_P + (\nabla \vec{u}_P)^T] - \frac{2}{3} \mu_P (\nabla \cdot \vec{u}_P) I \quad 6.6$$

C_{PS} or C_{SP} is the inter-phase exchange coefficient, given as:

$$C_{SP} = C_{PS} = \frac{\rho_S V_S f}{t_r} \quad 6.7$$

t_r the particle relaxation time is given as:

$$t_r = \frac{\rho_S d_s^2}{18 \mu_P} \quad 6.8$$

In equation 6.8, f is the drag function as a function of C_D , the drag coefficient and Re_S , the Reynolds no. based on the relative motion of the two phases such that:

$$f = \frac{C_D Re_S V_P}{24 v_t^2} \quad 6.9$$

From here, the inter-phase exchange coefficient can be written as:

$$C_{SP} = C_{PS} = \frac{3 \rho_P V_P V_S}{4 v_t^2 d_S} C_D \left(\frac{Re_S}{v_t} \right) |\vec{u}_S - \vec{u}_P| \quad 6.10$$

Here, Re_S is the Reynolds no. based on the relative motion of the two phases.

$$Re_S = \frac{\rho_P d_S |\vec{u}_S - \vec{u}_P|}{\mu_P} \quad 6.11$$

v_t is the terminal velocity correlation of coal particles in water such that:

$$v_t = 0.5 \left(C_1 - 0.06 Re_S + \sqrt{(0.06 Re_S)^2 + 0.12 Re_S (2C_2 - C_1) + C_1^2} \right) \quad 6.12$$

$$C_1 = V_P^{4.14} \text{ \& } C_2 = 0.8 V_P^{1.28} \quad 6.12 \text{ (a)}$$

C_{vm} and C_L represent the virtual mass and lift coefficients, taken as 0.5 each for the present study.

The term $\mathbf{F}_S$ is a collective term for different forces acting on the secondary/solid phase (coal).

$\mathbf{F}_S$ is represented as:

$$\begin{aligned} F_S = & \underbrace{\rho_S V_S \vec{g}}_{\text{Body force}} + \underbrace{\nabla \cdot \tau_S}_{\text{Viscous force}} + \underbrace{C_{PS}(\vec{u}_P - \vec{u}_S)}_{\text{Inter-phase drag force}} + \underbrace{C_{vm} V_S \rho_P (\vec{u}_P \cdot \nabla \vec{u}_P - \vec{u}_S \cdot \nabla \vec{u}_S)}_{\text{Virtual mass force}} \\ & + \underbrace{C_L V_S \rho_P (\vec{u}_P - \vec{u}_S) \times (\nabla \times \vec{u}_P)}_{\text{Lift Force}} \end{aligned} \quad 6.13$$

Where τ_S , solid phase stress tensor is described as:

$$\tau_S = (\zeta_S \nabla \cdot \vec{u}_S - P_S) I + \mu_S \left\{ [\nabla \vec{u}_S + (\nabla \vec{u}_S)^T] - \frac{2}{3} \mu_S (\nabla \cdot \vec{u}_S) I \right\} \quad 6.14$$

The bulk solids viscosity is denoted by ζ_S and described as:

$$\zeta_S = \frac{4}{3} V_S \rho_S d_S Y_r (1 + e_c) \sqrt{\frac{\Theta_c}{\pi}} \quad 6.15$$

Where, d_S is the coal particle diameter. Y_r is the radial distribution function and basically reflects the chances of interparticle contact and e_c is coefficient of restitution, taken as 0.9 in the present case. Y_r is described as:

$$Y_r = \left(1 - \frac{V_S}{V_{SC}} \right)^{-2.5 V_{SC}} \quad 6.16$$

Here, V_{SC} is the maximum settled concentration of the secondary phase

In equation 6.16, P_S is the bulk solids pressure and is given as:

$$P_S = \rho_S V_S \Theta_S + 2 \rho_S V_S^2 (1 + e_s) Y_r \Theta_S \quad 6.17$$

μ_P and μ_S are the shear viscosities of water and coal respectively. The solid shear viscosity is expressed as:

$$\mu_S = \mu_{Kinetic} + \mu_{Friction} + \mu_{Collision} \quad 6.18$$

Where $\mu_{Kinetic}$ is the kinetic viscosity, $\mu_{Friction}$ is the frictional viscosity and $\mu_{Collision}$ is the collisional viscosity of the solid particles. These viscosities are formulated as:

$$\mu_{Kinetic} = \frac{V_S d_S \rho_S \sqrt{\Theta_S \pi}}{6(3 - e_C)} \left[1 + \frac{2}{5} (1 + e_S)(3e_S - 1) Y_r V_S \right] \quad 6.19$$

$$\mu_{Friction} = \frac{P_S \sin \phi}{2\sqrt{I_{2D}}} \quad 6.20$$

$$\mu_{Collision} = \frac{4}{5} V_S Y_r d_S \rho_S (1 + e_S) \sqrt{\frac{\Theta_S}{\pi}} \quad 6.21$$

Θ_S is the granular temperature, ϕ is angle of attack and is taken as 30° in the present case.

6.2.1.3 Turbulence modelling for solid and liquid phases:

Different turbulence models viz. Standard $k - \epsilon$ with enhanced wall treatment, Standard $k - \epsilon$ with scalable wall functions, SST $k - \omega$, and Standard $k - \omega$ have been used initially to identify the most accurate model.

i. Standard $k - \epsilon$ model:

The standard $k - \epsilon$ model is a two-equation model used for the modelling of k , the turbulence kinetic energy and ϵ , turbulence dissipation rate. These equations are

$$\frac{\partial}{\partial x_i} (\rho k u_i) = \frac{\partial}{\partial x_j} \left[\left(\mu + \frac{\mu_T}{\sigma_k} \right) \frac{\partial k}{\partial x_j} \right] - \rho \overline{u_i' u_j'} \frac{\partial u_j}{\partial x_i} - \rho \epsilon \quad 6.22$$

$$\frac{\partial}{\partial x_i} (\rho \epsilon u_i) = \frac{\partial}{\partial x_j} \left[\left(\mu + \frac{\mu_T}{\sigma_\epsilon} \right) \frac{\partial \epsilon}{\partial x_j} \right] - 1.44 \frac{\epsilon}{k} \left(\rho \overline{u_i' u_j'} \frac{\partial u_j}{\partial x_i} \right) - 1.92 \rho \frac{\epsilon^2}{k} \quad 6.23$$

The wall functions can be modelled as “Standard wall functions”, “Scalable wall functions” and “Enhanced wall treatment”. These are discussed below.

Standard wall functions:

For rough pipes, the near wall flow is modeled as:

$$\frac{u_{pw}u^*}{\tau_w/\rho} = \frac{1}{\kappa} \ln(Ey^*) - \Delta R \quad 6.24$$

u_{pw} is the mean particle velocity at the wall-adjacent cell centroid, τ_w is the wall shear stress, ρ is the density of the mixture/slurry. The Von Karman constant κ has a value of 0.4187, E , is the empirical constant and has a value of 9.793.

For computationally inexpensive near wall boundary layer modelling, Ansys Fluent uses y^* , the dimension-less wall distance, that is a function of the turbulence kinetic energy (at wall adjacent cell centroid), k_a only. Unlike the formulation of y^+ , the formulation of y^* does not requires any kind of iteration as the latter can be calculated from the turbulence kinetic energy, k_a ; however, the former is the function of an unknown quantity viz. τ_w (as $y^+ = \frac{\rho y_a \sqrt{\tau_w/\rho}}{\mu}$). So, to find out two unknowns from one equation, a separate iterative method is needed, that makes the formulation computationally expensive. The formulation of y^* is done as:

$$y^* = \frac{\rho u^* y_a}{\mu} \quad 6.25$$

Here y_a is the distance from the centroid of the wall adjacent cell to the wall. μ is the dynamic viscosity of the slurry and is modelled as a power law fluid. u^* is the frictional velocity and is defined as the function of turbulence kinetic energy at wall adjacent cell centroid, k_a as:

$$u^* = \sqrt{\beta^{*1/2} k_a} \quad 6.26$$

In equation 6.24, ΔR accounts for the flow losses due to pipe roughness and is formulated as:

$$\Delta R = \frac{1}{\kappa} \ln \left(1 + \frac{C_R K_R \rho u^*}{\mu} \right) \quad 6.27$$

Here C_R and K_R are the roughness constant ($C_R = 0.5$) and physical roughness of the pipe surface in meters.

Scalable wall functions:

The scalable wall functions are used to force the usage of log-law along with the standard wall functions. This requirement is fulfilled using a limiting function such that:

$$\widetilde{y}^* = \text{Max}(y^*, 11.225) \quad 6.28$$

So, now in the scalable wall functions, the y^* is replaced by $\widetilde{y}^*$

Enhanced wall treatment:

In this approach, the whole vicinity is divided into two regions viz. full turbulent region and viscosity-affected region. For the identification these regions, a wall-distance based Reynolds number, Re_y is calculated as:

$$Re_y = \frac{\rho y_a \sqrt{k_a}}{\mu} \quad 6.29$$

Re_y for fully turbulent is >200 . So, the regions where Re_y is <200 fall in the viscosity affected regions. In such regions, the equation 6.22 holds good however, the turbulent viscosity, μ_T is modelled as:

$$\mu_{T,vis} = \rho C_\mu l_\mu \sqrt{k_a} \quad 6.30$$

$$l_\mu = y C_l^* \left(1 - e^{-Re_y/A_\mu} \right) \quad \left| \quad C_l^* = \kappa C_\mu^{-0.75}; A_\mu = 70 \quad 6.31$$

The blending of the μ_T in the turbulent region with the $\mu_{T,vis}$ in the viscosity region is done using a blending function λ_ε such that it has value unity away from walls and zero in the viscosity affected regions. So, throughout the near wall region, $\mu_{T,enh}$ can be defined as:

$$\mu_{T,enh} = \lambda_\varepsilon \mu_T + (1 - \lambda_\varepsilon) \mu_{T,vis} \quad 6.32$$

The turbulence dissipation rate ε in the viscosity affected regions is calculated as:

$$\varepsilon = \frac{k^{3/2}}{l_\varepsilon} \quad 6.33$$

$$l_\varepsilon = y C_l^* \left(1 - e^{-Re_y/A_\varepsilon} \right) \quad \left| \quad A_\mu = 2 C_l^* \quad 6.34$$

ii. Standard $k - \omega$ model:

The $k - \omega$ model is a two-equation model; these two equations are used to predict the turbulence in the fluid flow. In $k - \omega$ model, the transport equations for turbulence kinetic energy, k and specific dissipation rate ω are given as:

$$\frac{\partial}{\partial x_i}(\rho k u_i) = \frac{\partial}{\partial x_j} \left[\left(\mu + \frac{\mu_T}{\sigma_k} \right) \frac{\partial k}{\partial x_j} \right] - \rho \overline{u'_i u'_j} \frac{\partial u_j}{\partial x_i} - \Omega_k \quad 6.35$$

$$\frac{\partial}{\partial x_i}(\rho \omega u_i) = \frac{\partial}{\partial x_j} \left[\left(\mu + \frac{\mu_T}{\sigma_\omega} \right) \frac{\partial \omega}{\partial x_j} \right] - \alpha \frac{\omega}{k} \left(\rho \overline{u'_i u'_j} \frac{\partial u_j}{\partial x_i} \right) - \Omega_\omega \quad 6.36$$

The turbulent eddy viscosity is modelled as:

$$\mu_t = \alpha^* \frac{\rho k_a}{\omega} \quad 6.37$$

The coefficient α is given by:

$$\alpha = \frac{0.52}{\alpha^*} \left(\frac{1/9 + Re_t/2.95}{1 + Re_t/2.95} \right) \quad 6.38$$

$$\alpha^* = \left(\frac{0.216/9 + Re_t/6}{1 + Re_t/6} \right) \quad 6.39$$

$$Re_t = \frac{\rho k_a}{\mu \omega} \quad 6.40$$

iii. SST $k - \omega$ model:

The standard $k - \omega$ model is highly sensitive to the freestream condition and performs very accurately in the regions close to the wall. The freestream conditions can be easily modelled by the $k - \epsilon$ model. So, the SST $k - \omega$ takes the advantage of accurate flow prediction in the near wall and free stream regions using both the models and the blending function. The blending function assumes the value unity in the near wall regions and turns off the $k - \epsilon$ model, while in the freestream regions, assumes value of zero to turn off the $k - \omega$ model.

The turbulent Prandtl numbers σ_k and σ_ω for k and ω are used to achieve the blending using the blending function, CF . σ_k and σ_ω are evaluated as:

$$\sigma_k = \frac{1}{CF/2 + (1 - CF)} \quad 6.41$$

$$\sigma_\omega = \frac{1}{CF/2 + (1 - CF)/1.168} \quad 6.42$$

$$CF = \tanh \left[\left\{ \min \left\{ \max \left(\frac{\sqrt{k}}{0.09 \omega y}, 500 \frac{\nu}{\omega y^2} \right), \frac{3.42 \rho k}{y^2} \times 10^{10} \right\} \right\}^4 \right] \quad 6.43$$

The turbulent eddy viscosity in SST $k - \omega$ model is evaluated as:

$$\mu_t = \frac{\rho k}{\omega} \frac{1}{\max \left[\frac{1}{\alpha^*}, \frac{S \cdot CF_2}{0.31\omega} \right]} \quad 6.44$$

$$CF_2 = \tanh \left(\max \left[2 \frac{\sqrt{k}}{0.09\omega y}, 500 \frac{\nu}{\omega y^2} \right] \right) \quad 6.45$$

As proposed by Messa and co-workers in [185,186] No-slip condition has been imposed on both the phases as the wall boundary condition in the present study. The near wall flow is modelled as standard wall functions as per equations 6.24-6.27.

6.2.2 Flow domain modelling:

6.2.2.1 Geometry:

The geometry of the flow domain, the horizontal pipe is modelled in the DesignModeler module of the Ansys™ Workbench 2020 R1. The diameter of the pipe is 100mm and has a length of 1500mm. The simulation results indicate a full flow development at 1000mm distance from the pipe inlet. So, the length of the pipe is sufficient for a fully developed flow. The pipe material considered is mild steel of density of 7850 kg/m³. The geometrical model of the horizontal pipeline developed in the DesignModeler is presented in Figure 6.1.

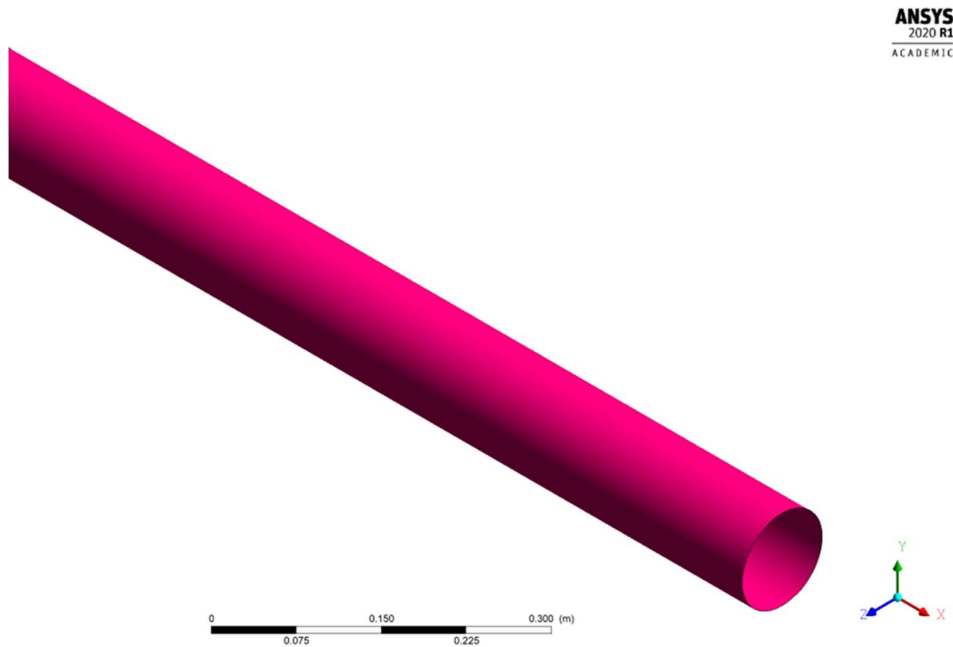


Figure 6.1 Geometry of the horizontal pipe modelled in the DesignModeler™ module of Ansys™ Workbench

6.2.2.2 Meshing:

The discretization of the horizontal pipe is performed using the inbuilt meshing module of AnsysTM Workbench 2020 R1. The key parameters of the meshing system are presented in Table 6.1.

Table 6.1 Details of the meshing system

Parameter	Untwisted pipe
Cell size (mm)	10 to 6 (0.5mm step)
Cell type	Tetrahedrons
No. of Inflation layers	8
Inflation transition ratio	0.237
Inflation growth rate	1.15

The size of grid is varied from 10 to 6mm to achieve the grid independence. The grid independence study is carried out based on the physical and rheological characteristics of coal water slurry at 60% concentration. The flow velocity of 0.25 to 1.25 m/s is selected for the present study. The results obtained using different sized grids is presented in Figure 6.2.

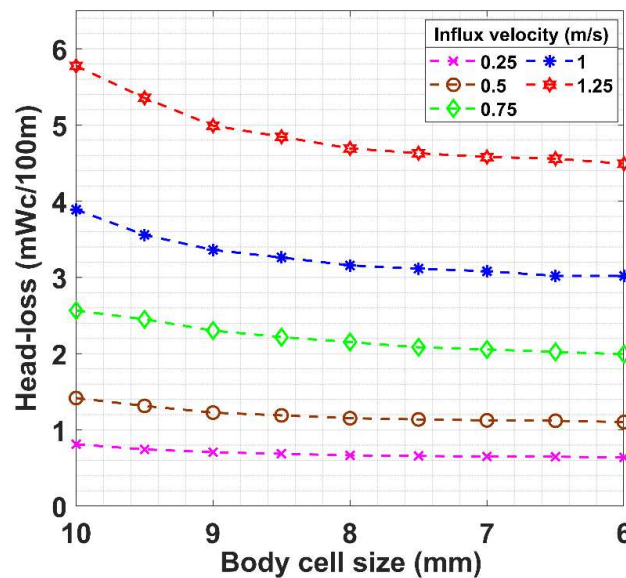


Figure 6.2 Static head-loss predicted by the different sized grids of size ranging from 10 to 6mm with step size of 0.5mm.

The results clearly indicated the presence of significant amount of error in the larger grids. The headloss is observed to fall with reduction in the grid size. The relative change in the headloss is observed to decrease with grid size. The results obtained at 6.5mm and 6mm sized grids are almost similar. So, it can be said that the grid independence has been achieved at 6.5mm size of the grid. The final mesh obtained at 6.5 mm size of the grid is shown in Figure 6.3.

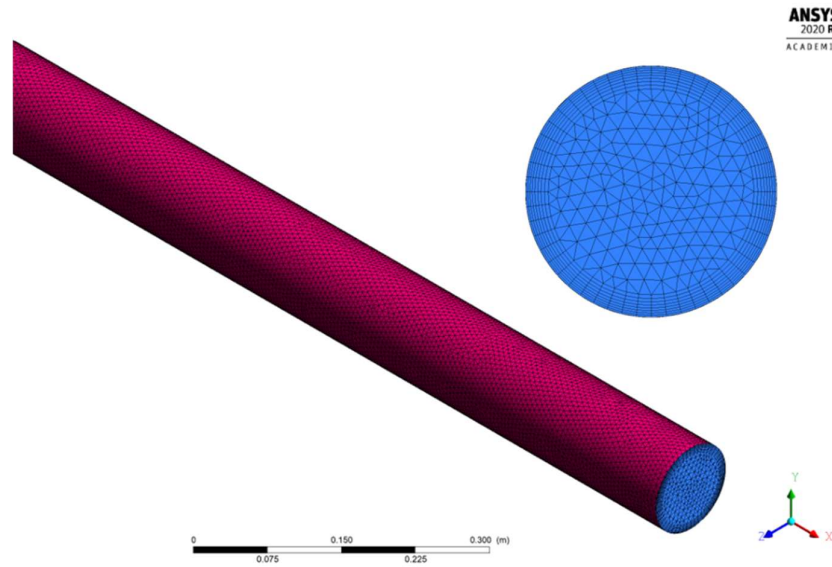


Figure 6.3 Complete mesh of the flow domain showing the wall and the pipe outlet

6.2.3 Boundary and operating conditions:

The flow domain is assigned three different boundaries viz. Inlet, outlet and wall. These boundary conditions are presented in Table 6.2.

Table 6.2 Boundary conditions

Boundary	Inlet	Outlet	Wall
Boundary type	Velocity inlet	Pressure outlet	Wall
Velocity (m/s)	Variable; 0.25-1.25	Not to be specified	0, due to no slip condition
Pressure (Pa)	Operating pressure of 101325	Operating pressure of 101325	Operating pressure of 101325

Turbulence (medium intensity)	Specified as Turbulence intensity of 4%, and hydraulic diameter of 100mm	Specified as Turbulence intensity of 4%, and hydraulic diameter of 100mm	Not to be specified
Volume fraction	Variable; 0.07411- 0.5193	Not to be specified	Not to be specified
Roughness (m)	Not to be specified	Not to be specified	6.3×10^{-5}

6.2.4 Model validation:

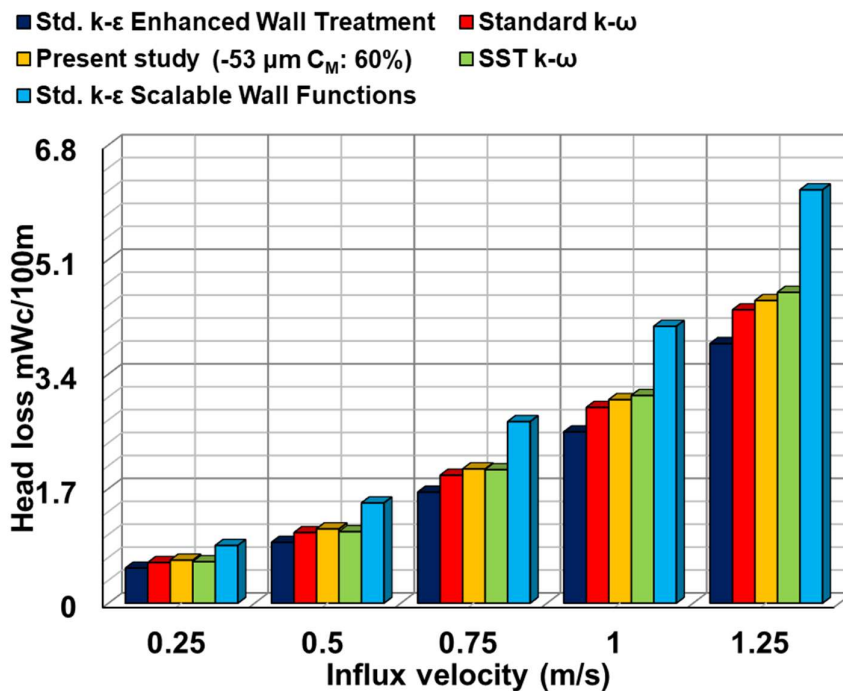


Figure 6.4 The results predicted by different turbulence models viz. Standard $k - \varepsilon$ with enhanced wall treatment, Standard $k - \varepsilon$ with scalable wall functions, SST $k - \omega$, and Standard $k - \omega$ compared with present experimental results

Different turbulence models viz. Standard $k - \varepsilon$ with enhanced wall treatment, Standard $k - \varepsilon$ with scalable wall functions, SST $k - \omega$, and Standard $k - \omega$ have been used initially to identify the most accurate model. As the theory of the standard $k - \varepsilon$ model indicates, is best for completely turbulent flows and highly accurate flow prediction away from the wall. The enhanced wall treatment and scalable wall functions are used on the top of the $k - \varepsilon$ model to make it compatible with the present case(s).

The results predicted by various turbulence schemes are presented in Figure 6.4. When compared with the present experimental results, (Figure 6.4) the results predicted by Standard $k - \epsilon$ model with enhanced wall treatment underpredicted the results with error ranging from 12 to 18%. Similarly, the Standard $k - \epsilon$ model with scalable wall function overpredicted the results. The error in this case lies in the range of -33 to -37%. The Standard $k - \epsilon$ model despite the use of the special wall functions is found to fail in the accurate prediction of the results. The Standard $k - \omega$ is based on the original Wilcox model, that is known for the precise flow prediction close to the wall. The results predicted by this model are better than those predicted by the $k - \epsilon$ models but are not acceptable as the results are underpredicted.

The SST $k - \omega$, model is an advanced variant of the Standard $k - \omega$ model. This model uses the original Wilcox model (Standard $k - \omega$ model) to accurately predict the flow close to the wall and the Standard $k - \epsilon$ model for the accurate flow formulation away from the wall, thus covering a broad spectrum of flow. In the present case, this model is found to give satisfactory results and the error in results' prediction range from -2.71 to 3.69%. As indicated by Figure 6.4, the SST $k - \omega$ model gives the best performance and hence used in the present study.

Flow modelling through pipe fittings:

The pre-validated SST $k - \omega$ model is used to predict the flow through the common pipe fittings viz. 90° elbow/mitter bend, converging section and diverging section. The flow parameters for these pipe fittings are; flow velocity of 0.5 to 5m/s and influx concentration of coal ranging from 10 to 60% by mass.

6.3 Flow modelling through a 90° elbow/mitter bend:

6.3.1 Modelling of the flow domain

6.3.1.1 Geometry:

The geometry of the 90° elbow is modelled in the Design Modeler module of the Ansys™ Workbench 2020 R1. Figure 6.5 shows the geometry of the 90° elbow. The geometry consists of two pipes of 0.1m internal diameter, connected at 90° to each other. The pipe located before the elbow is upstream pipe and the other located after the elbow is the downstream pipe. In the present case, due to the limitation of the problem size of the educational software, only the lower half of the pipe-bend assembly is considered and modelled.



Figure 6.5 Geometry of the 90° pipe elbow

6.3.1.2 Meshing:

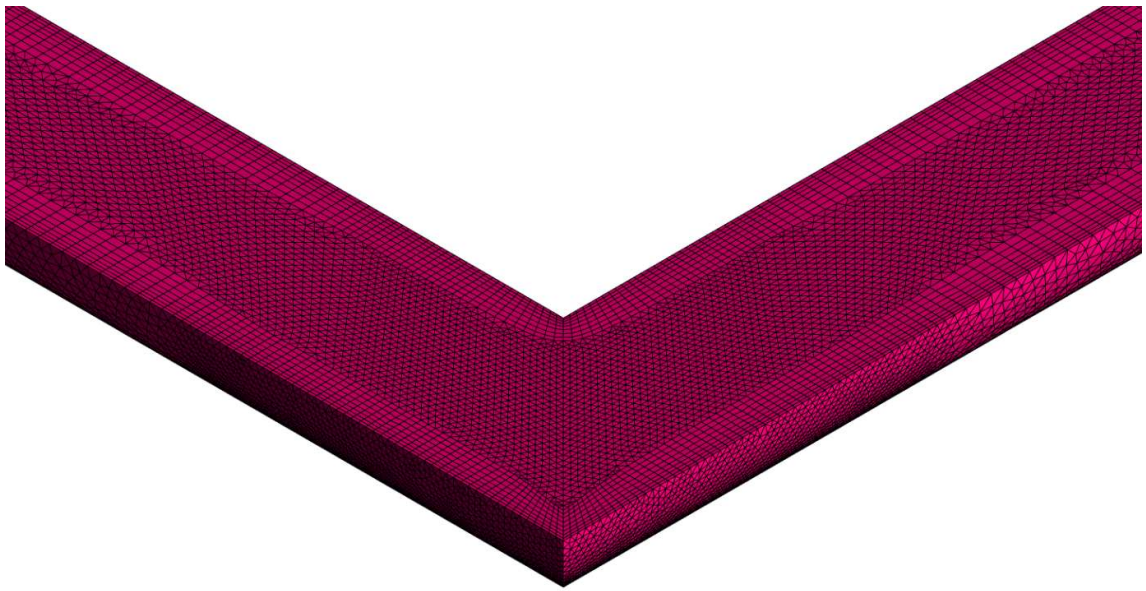


Figure 6.6 Meshing system of the 90° pipe elbow

The Meshing module of Ansys™ Workbench 2020 R1 is used to discretize the fluid domain in the present case. The meshing systems of the pipe fitted with a 90° elbow are represented in Figure 6. Tetrahedron shaped cells are used in the present case, using the patch conforming method. The inflation layers also help in achieving the y^+ values of ~ 30 -35. The size of each cell is decided based on the results produced by different sized cells and represented in Figure 6.3. The mesh near the 90° elbow is significantly refined to get a better flow prediction in the vicinity of the bend. A uniform mapped mesh is then created with the

help of the face meshing command. To have precise near wall flow prediction, the inflation layers are applied adjacent to the walls with a growth rate of 1.2.

6.3.1.3 *Boundary conditions:*

The fluid domain is resolved into four boundaries viz. inlet, wall, outlet and symmetry. The boundary conditions at these boundaries are given in Table 6.3 The symmetry boundary condition is applied on the sectioned face of the pipe-elbow assembly. The cell-zone is assigned the fluid, that is a mixture of coal and water.

Table 6.3 Boundary conditions

Boundary	Inlet	Outlet	Wall
Boundary type	Velocity inlet	Pressure outlet	Wall
Velocity (m/s)	0.5-5	Not to be specified	0, due to no slip condition
Pressure (Pa)	Operating pressure of 101325	Operating pressure of 101325	Operating pressure of 101325 Pa
Turbulence (medium intensity)	Specified as Turbulence intensity of 4%, and hydraulic diameter of 100mm	Specified as Turbulence intensity of 4%, and hydraulic diameter of 100mm	Not to be specified
Volume fraction	Variable; 0.07411-0.5193	Not to be specified	Not to be specified
Roughness (m)	Not to be specified	Not to be specified	6.3×10^{-5}

Results:

6.3.2 *Visualization of flow separation:*

The results generated by the Fluent module of Ansys™ Workbench 2020 R1 software are further analyzed using the CFD post Module of Ansys™ Workbench 2020 R1. The flow is visualized using vector plots to know the flow structure at any location in the upstream or downstream pipe. This analysis is carried out in order to check any case of flow separation inside the bend. The vector plot inside the bend is presented in Figure 6.7.

The flow separation is observed as soon as the flow enters the downstream pipe. This is due to the adverse pressure gradient close to the interior wall of the elbow. The region adjacent to the separated region experiences fluid flow at a very high velocity. The flow in this region experiences local acceleration. After travelling some distance in the downstream pipe, the flow is observed to reattach at distance of less than 0.25m from the interior corner of the elbow.

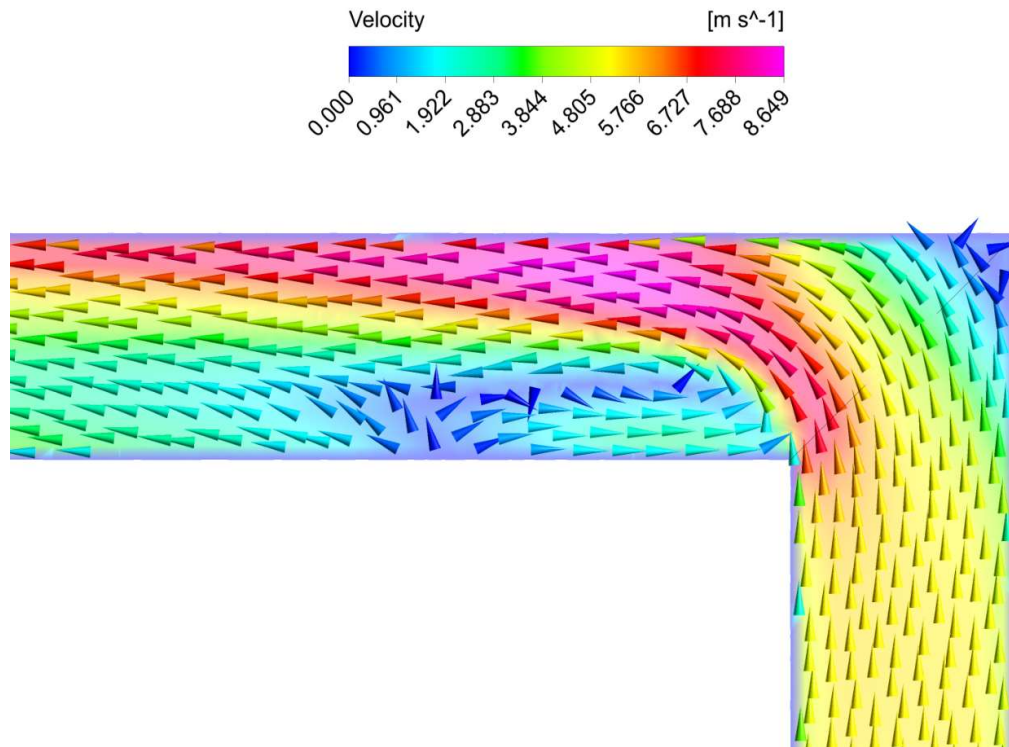


Figure 6.7 Flow separation visualized at the upstream $Re = 39230$. The vectors facing in the direction opposite to the streamwise direction belong to the separated flow.

The turbulence in the flow is analysed using the turbulence kinetic energy at different locations in the flow. The turbulence kinetic energy at the symmetry boundary is presented in Figure 6.8. The flow separation occurs at the interior corner of the elbow. Travelling some distance downstream, a region of very high turbulence is observed. At this location, an interface between the separated and unseparated flow is formed. As, the direction of flow on the either side is opposite to each other, so a high shear zone is formed at the interface.

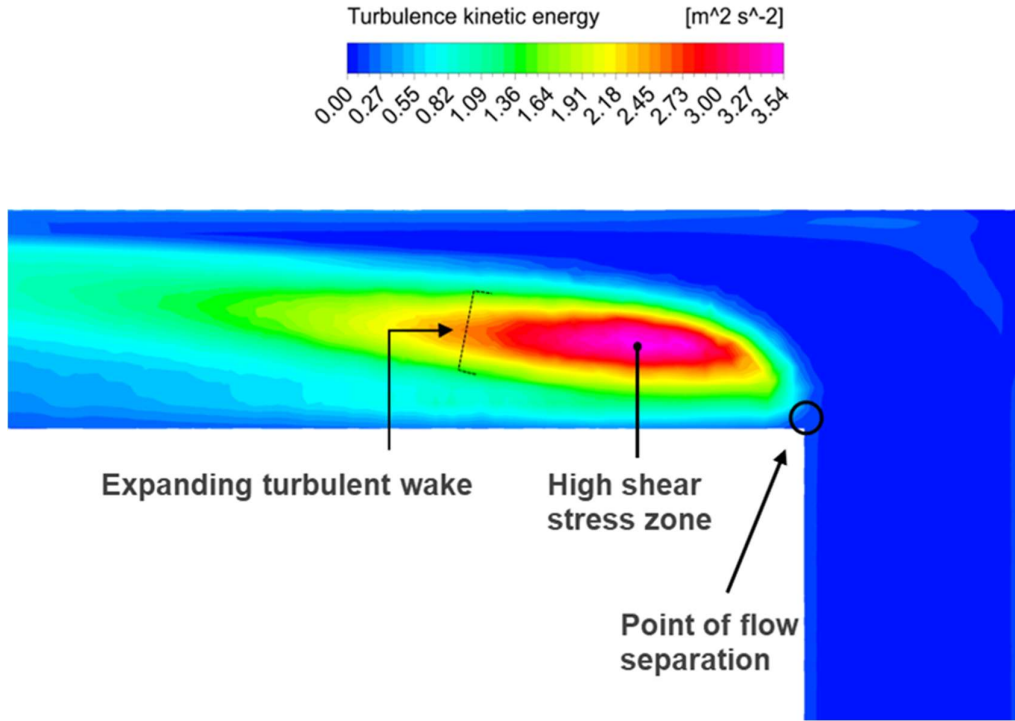


Figure 6.8 Turbulence in the flow describing the flow separation, high shear stress zone and expanding turbulent wake

The turbulence kinetic energy is a direct function of the shear stress. So, the turbulence kinetic energy is very high at the interface. The turbulent wake formed at the interface is observed to expand in the streamwise direction. At some distance downstream the turbulent wake impinges the pipe wall, where the flow is found to reattach.

6.3.3 Performance of the elbow:

As the flow travels from the upstream pipe, to the downstream pipe via the elbow, the flow is found separate inside the elbow. This flow separation is always associated with very high headloss. Based on this headloss, the bendloss coefficient is also evaluated. Both these parameters serve as the performance parameters for the elbow.

6.3.3.1 Headloss associated with the elbow:

The headloss across the elbow is investigated by comparing it with a straight pipe of same length. The idea behind this investigation is presented in Figure 6.9.

Mathematically, the headloss across the elbow can be calculated as:

$$H_L = \frac{1}{\rho \cdot g} \left[\left(P_{inlet} - l \cdot \frac{\Delta P}{\Delta l} \right) - P_{outlet} \right] \quad 6.46$$

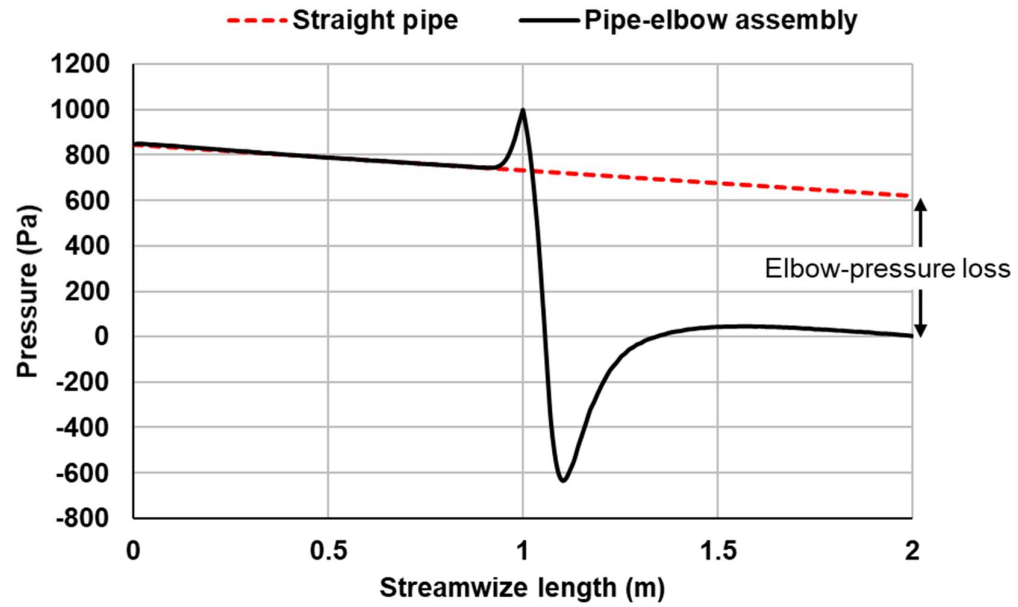


Figure 6.9 Methodology followed for the investigation of headloss across the elbow

Here H_L is the headloss across the elbow, P_{inlet} is the pressure at the inlet of the upstream pipe, l is the total length of the upstream and downstream pipes, $\frac{\Delta P}{\Delta l}$ is the pressure gradient in the straight pipe of length l and P_{outlet} is the pressure at the outlet of the downstream pipe.

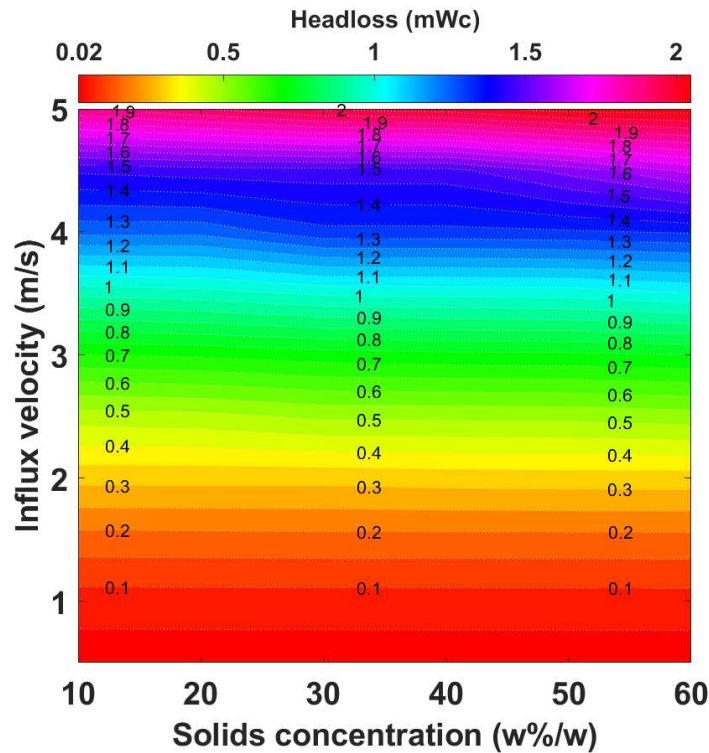


Figure 6.10 Headloss across the elbow.

The results obtained using the equation 6.29 are presented in Figure 6.10. The headloss across the elbow is observed to increase significantly with influx velocity. At higher velocities, the wall shear stress in the regions close to the wall is higher, so the pressure loss is also observed to increase with influx velocity. The head loss is observed to increase with concentration of coal present in the slurry. This trend is more clearly visible at higher influx velocities. So, maximum headloss is observed when coal water slurry containing 60% coal is conveyed at a velocity of 5 m/s.

6.3.3.2 Bendloss coefficient:

The bendloss coefficient is based on the headloss obtained in the previous subsection and is calculated as:

$$H_L = K_B \frac{\bar{V}^2}{2 \cdot g} \quad 6.47$$

Here K_B is the bendloss coefficient and $\bar{V}$ is the mean velocity of the flow in the upstream pipe.

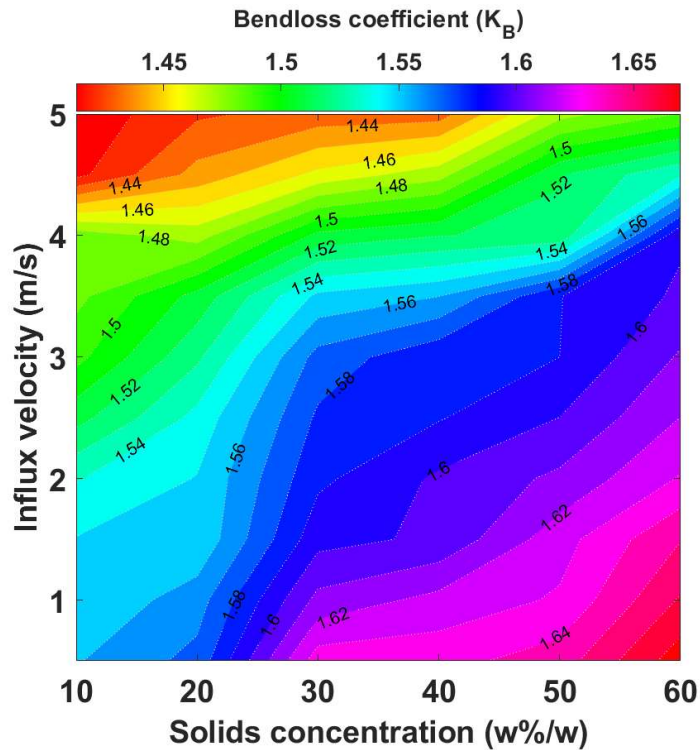


Figure 6.11 Bend loss coefficient of the elbow at different influx velocities and concentration of coal

The bend loss coefficient is observed to decrease with the influx velocity. Comparing equation 6.47 with the Darcy-Weisbach equation, the bendloss coefficient can be written as:

$$K_B = \frac{f_d \cdot l}{d} \quad 6.48$$

Here, f_d is the Darcy's friction factor, which is always observed to decrease with Reynolds No. or in this case the influx velocity. Similarly, the increase in solids concentration leads to decrease in the Reynolds number of the flow, so the friction factor is observed to increase at higher concentrations and so is the bend loss coefficient.

So, it can be concluded that the headloss is observed in increase at higher concentration and higher velocities of the flow.

6.4 Flow modelling through a converging section:

6.4.1 Modelling of the flow domain:

6.4.1.1 Geometry:

The geometries of the converging sections are modelled in the DesignModeler module of Ansys™ Workbench 2020 R1. Figure 6.12 shows the geometries of the converging sections tested in the present study.



Figure 6.12 Modelled fluid domain for the 0.05m converging section

The section consists of two cylindrical pipe sections of diameters 100mm and 50mm respectively and a converging section. The 100mm diameter pipe section is the upstream section and the 50mm diameter pipe section, located after the converging section, is the downstream section. The horizontal length, N of the diverging section is varied from 0.05m to 0.6m, ($N = 0.05, 0.1, 0.15, 0.2, 0.25, 0.3, 0.4, 0.5, 0.6$ m).

6.4.1.2 Meshing:

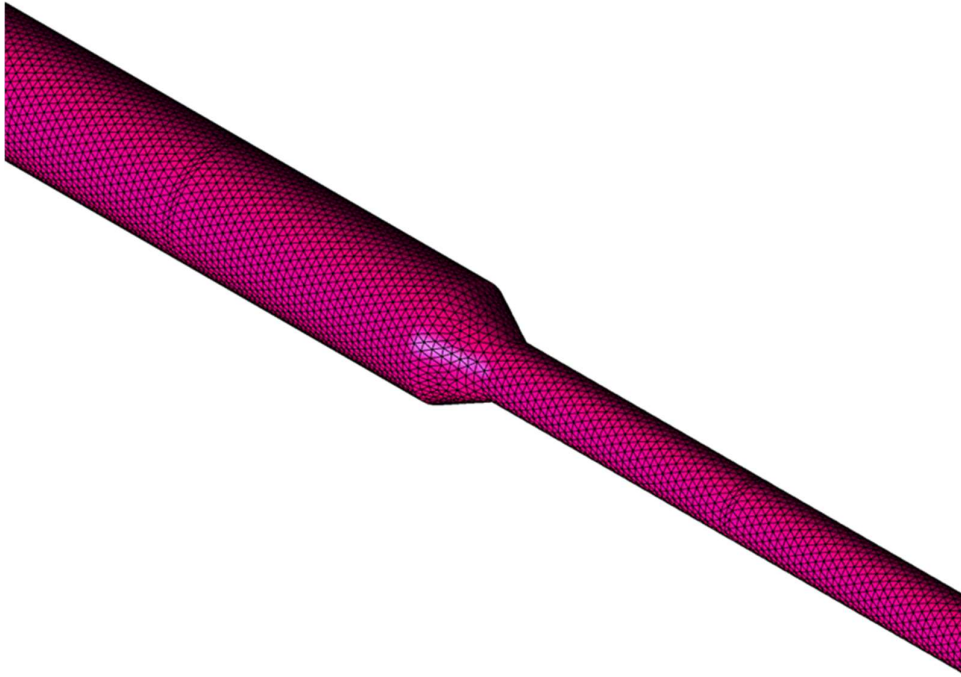


Figure 6.13 Meshed fluid domain for the 0.05m converging section

The Meshing module of AnsysTM Workbench 2020 R1 is used to discretize the fluid domain in the present case. The meshing systems of the pipe fitted with a converging section are represented in Figure 6.13 Tetrahedron shaped cells are used in the present case, using the patch conforming method. The inflation layers also help in achieving the y^+ values of ~ 30 -35. The size of each cell is decided based on the results produced by different sized cells and represented in Figure 6.3.

6.4.1.3 Boundary conditions:

The fluid domain is resolved into three boundaries viz. inlet, wall and outlet. The pictorial representation of the domains is shown in Figure 6.14. The boundary conditions at these boundaries are given in Table 6.4.

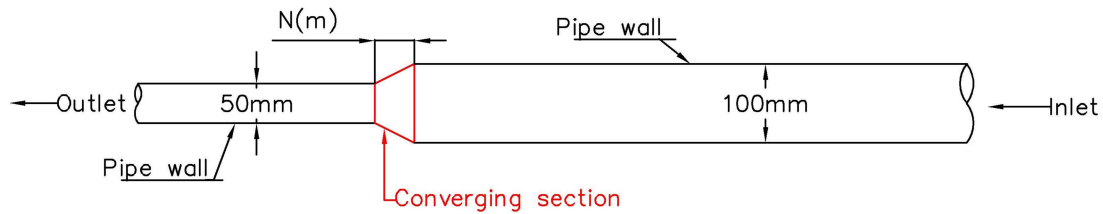


Figure 6.14 Schematic of the converging pipe section. The inlet upstream pipe has a 100 mm diameter, that transits into a 50 mm diameter pipe with help of a 50-100 mm converging section.

Table 6.4 Boundary conditions

Boundary	Inlet	Outlet	Wall
Boundary type	Velocity inlet	Pressure outlet	Wall
Description	Cross-section of $D_1 = 100\text{mm}$	Cross-section of $D_2 = 50\text{mm}$	Cylindrical walls of 50mm and 100mm diameter pipes and conical wall of the converging pipe section
Velocity (m/s)	Variable; 0.5-5	Not to be specified	0, due to no slip condition
Pressure (Pa)	Operating pressure of 101325	Operating pressure of 101325	Operating pressure of 101325
Turbulence (medium intensity)	Specified as Turbulence intensity of 4%, and hydraulic diameter of 100mm	Specified as Turbulence intensity of 4%, and hydraulic diameter of 50mm	Not to be specified
Volume fraction	Variable; 0.0741-0.5193	Variable; 0.0741-0.5193	Not to be specified
Roughness (m)	Not to be specified	Not to be specified	6.3×10^{-6}

Results

6.4.2 Visualization of the flow and turbulence in the flow:

The results generated by the Fluent module of Ansys™ Workbench 2020 R1 software are further analyzed using the CFD post Module of Ansys™ Workbench 2020 R1. The flow is visualized using vector plots to know the flow structure at any location in the upstream, converging or downstream pipe. This analysis is carried out in order to check any case of flow separation inside the converging section. The vector plot inside a 0.05m converging section is presented in Figure 6.15.

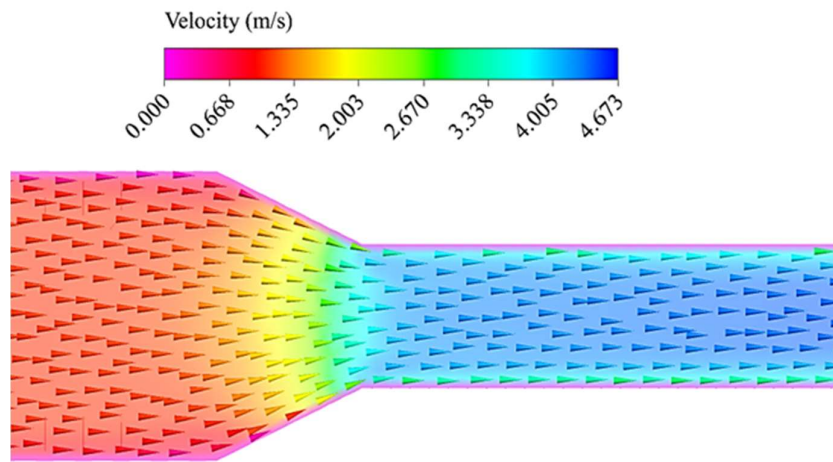


Figure 6.15 Flow structure visualized at downstream Re of 18336. No sign of flow separation is observed inside the converging section.

The flow structure presented in Figure 6.15 indicates no sign of flow separation or flow recirculation. Due to the reduction in the flow area, the velocity is observed to magnify and the downstream velocity is almost 4 times the upstream velocity. Similar results have been obtained in the cases of all other geometries of the converging sections.

The turbulence in the flow is visualized with the help of contour plot of the turbulence kinetic energy inside and downstream of the converging section. The turbulence kinetic energy is observed to be quite low in the upstream pipe. Upon entry into the converging section, the turbulence kinetic energy is observed to increase slightly in the regions close to the wall. The turbulence kinetic energy is observed to increase gradually in the streamwise direction in the downstream pipe. The nature of the turbulent wake is expanding in nature and covers more and more region of the downstream pipe as it proceeds in the downstream region. Similar observations have been made in the other geometries of the converging sections.

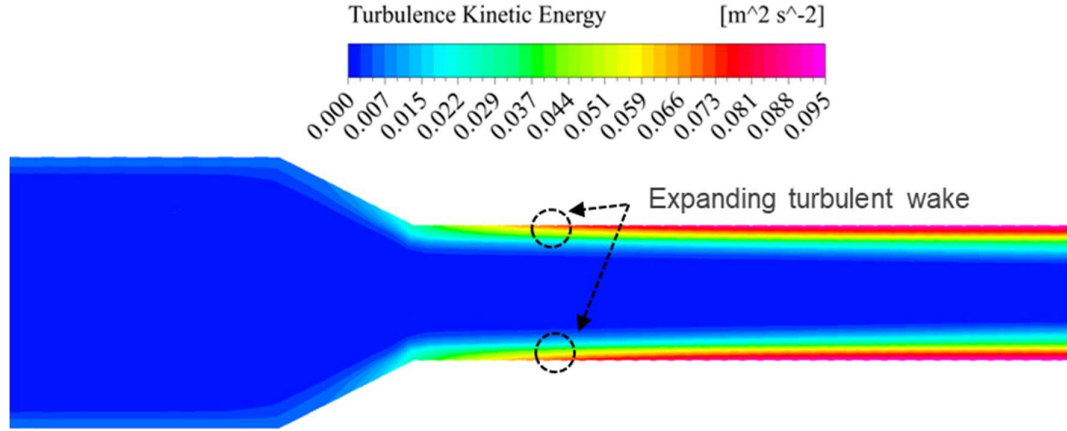


Figure 6.16 Turbulence in the flow

6.4.3 Performance characteristics of the converging section:

It is evident from the flow structure presented in Figure 6.16 that as the fluid stream from the upstream (0.1m dia) pipe enters the converging section, the first thing that happens is the conversion of the static pressure into the dynamic head or the kinetic energy of the fluid. The performance characteristics of the converging section refers to the fact that how efficiently the converging section is performing at this conversion process. To analyse, the performance of the converging section, the parameters, viz. head-loss across the converging section, pressure recovery coefficient and the friction factor are considered.

6.4.3.1 Pressure recovery coefficient:

The pressure recovery characteristics of the converging sections are analysed based on the pressure recovery coefficient ' C_P '. As per the Bernoulli's theorem, the reduction in the static pressure should reflect into the increase in the dynamic pressure downstream the converging section.

But in actual, the flow is also accompanied by some frictional losses, that lead to inefficiencies. Theoretically, the ' C_P ' is defined as the rise in the dynamic pressure at the outlet of the section to the available static head at the inlet of the section. The expression for the ' C_P ' is presented in equation. 6.48

$$C_P = \frac{\rho(\bar{V}_2^2 - \bar{V}_1^2)}{2 \cdot P_1} \quad 6.48$$

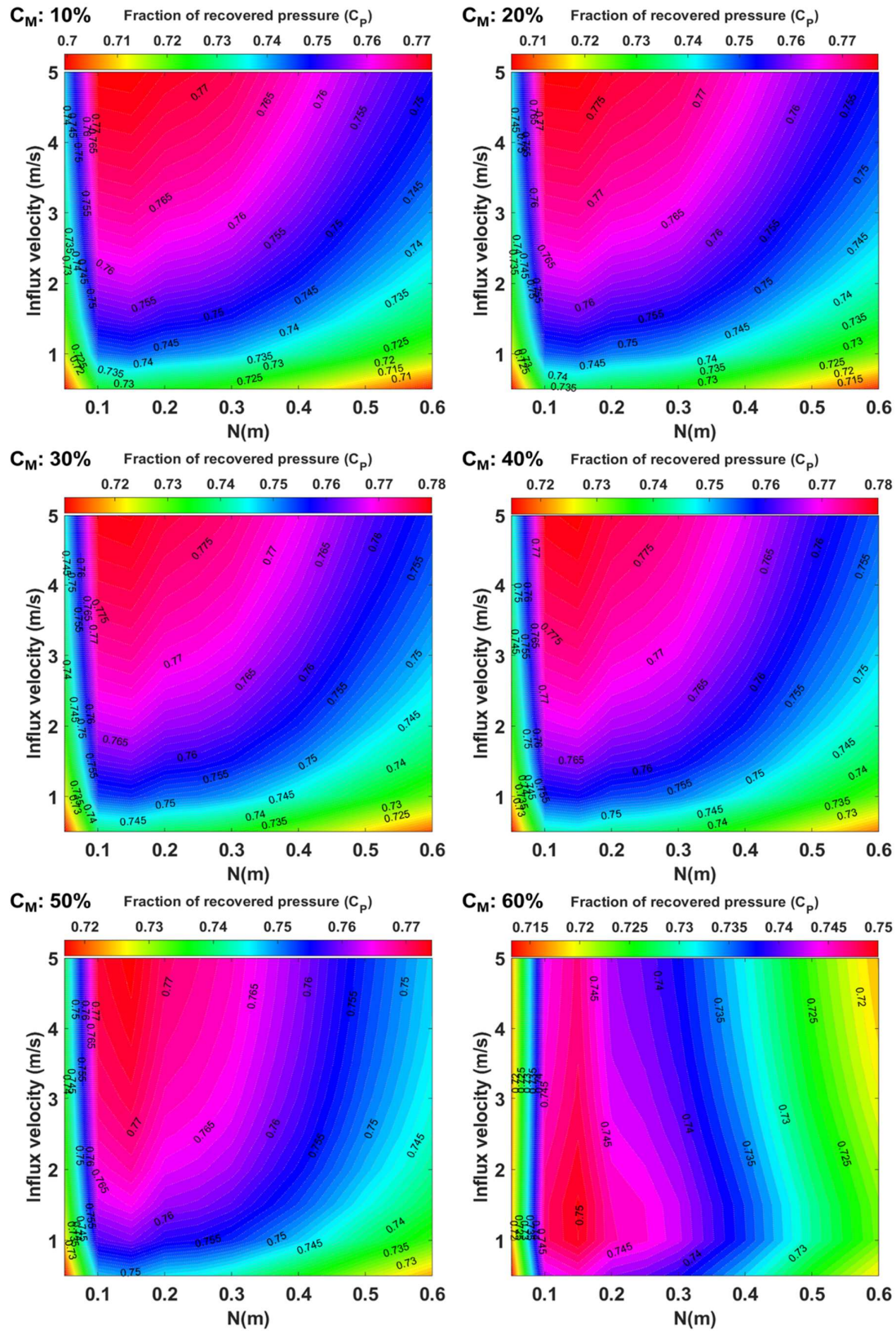


Figure 6.17 The pressure recovery coefficient for different converging sections at different influx velocities and influx concentrations of coal.

The pressure recovery coefficients obtained for various geometries of converging sections and slurries of different concentrations are presented in Figure 6.17. The pressure recovery coefficient is found to increase with influx velocity in all the geometries and for all the slurries (of different solid concentrations). Out of different geometries of the converging sections, the 0.05m long section is observed to perform poorly, possibly due to high pressure gradient inside the section. The further increase in the section length leads to an abrupt increase in the pressure recovery coefficient. This increase is also observed in the case of 0.15m long converging section. The further increase in the section length leads to reduction in pressure recovery coefficient. The 0.6 long section has pressure recovery coefficient slightly higher than the 0.05m long sections.

So, the shortest and the longest sections are observed to have the lowest pressure recovery coefficients. While the 0.15m section has the highest pressure recovery coefficient. The pressure recovery coefficient is observed to increase with concentration of coal up to 40% concentration and is observed fall at higher concentrations.

6.4.3.2 Head-loss across the converging section:

The headloss across the converging section is calculated as the loss of the velocity head due to the frictional drag. The Bernoulli's theorem for a real case can be written as:

$$\frac{P_1}{\rho \cdot g} + \frac{\bar{V}_1^2}{2 \cdot g} = \frac{P_2}{\rho \cdot g} + \frac{\bar{V}_2^2}{2 \cdot g} + H_L \quad 6.49$$

Where P_1 and P_2 are static pressures at the inlet and outlet of converging section. $\bar{V}_1$ and $\bar{V}_2$ are average velocities at the inlet and outlet of the converging section. The term H_L is the headloss associated with converging section. The headloss obtained using equation 6.49 for different cases is presented in Figure 6.18.

The headloss is the greatest in the case of the shortest (0.05m) and the longest (0.6m) sections. The headloss is observed to decrease in the case of 0.1m long converging section. The headloss is observed to be the least in the case 0.15m long section. The further increase in the length of the converging section leads to the increase in the headloss. The headloss is observed to increase with the influx velocity. The headloss is also observed to increase with concentration of the coal. At lower concentrations, a marginal increase in the headloss is observed with increase in the concentration of the coal.

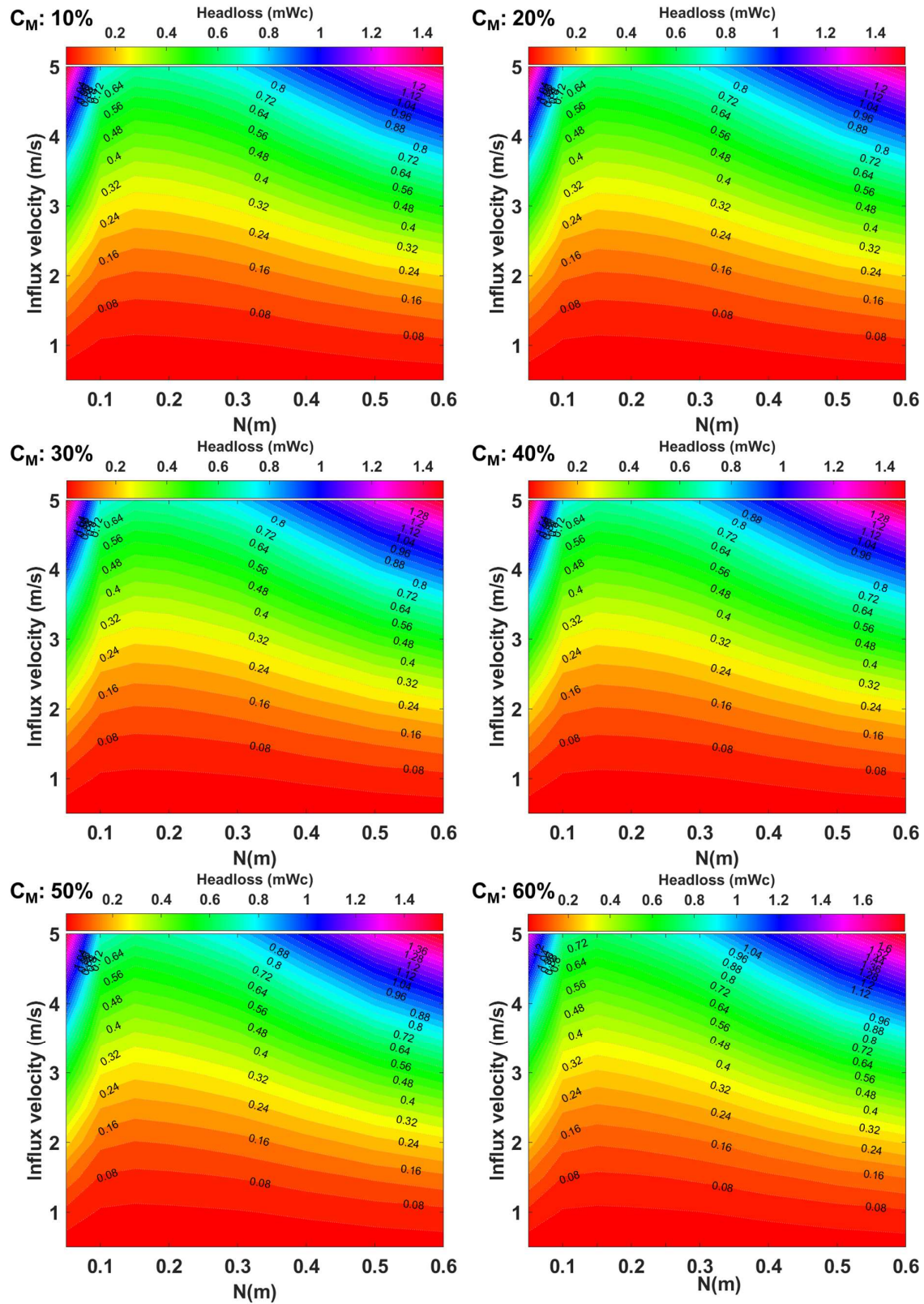


Figure 6.18 The headloss across different converging sections at different influx velocities and influx concentrations of coal.

However, a significant increase in the headloss is observed when the concentration of the coal is increased to 50%. The further increase in the concentration to 60% leads to an abrupt increase in headloss. The flow inside the converging section is associated to the headloss, which is dependent on two parameters. These are losses due to pressure gradient and losses due to skin friction. In the case of the short sections, like in the case of the 0.05 m section, very large pressure gradient is observed, but due to the short length of the section, the skin friction is very small. However, in the case of long sections, (0.6m) the pressure gradient is the least but due to greater surface area of the longer sections, the skin friction is significantly increased. The 0.15m section has a relatively lower pressure gradient as compared to the 0.05m section and lower skin friction as compared to the 0.6m section. It can be said that the 0.15m section is a design that balances both pressure gradient and the skin friction.

The increase in the influx velocity increases the shear rate in the flow and leads to increase in the wall shear stress. The high wall shear stress reflects in the form of high headloss at high influx velocities. Similarly, the increase in the concentration of coal in the slurry leads to increase in the shear stress at a given shear rate. In Chapter 4, the shear stress is observed to increase abruptly at 60% concentration of coal. Due to this, the headloss is also observed to increase significantly at 60% concentration.

6.4.3.3 Friction factor of the flow:

The headloss obtained in the previous case can be used have a deeper analysis of the problem in the form of friction factor of the flow. The following expressions and parameters are used to calculate the friction factor of the flow:

$$H_L = K_T \frac{\bar{V}_2^2}{2 \cdot g} \quad 6.50$$

Here K_T is the total pressure loss coefficient (based on the mean velocity in the minor diameter), which is further composed of local resistance coefficient and friction pressure loss coefficient.

$$K_T = K_F + K_L \quad 6.51$$

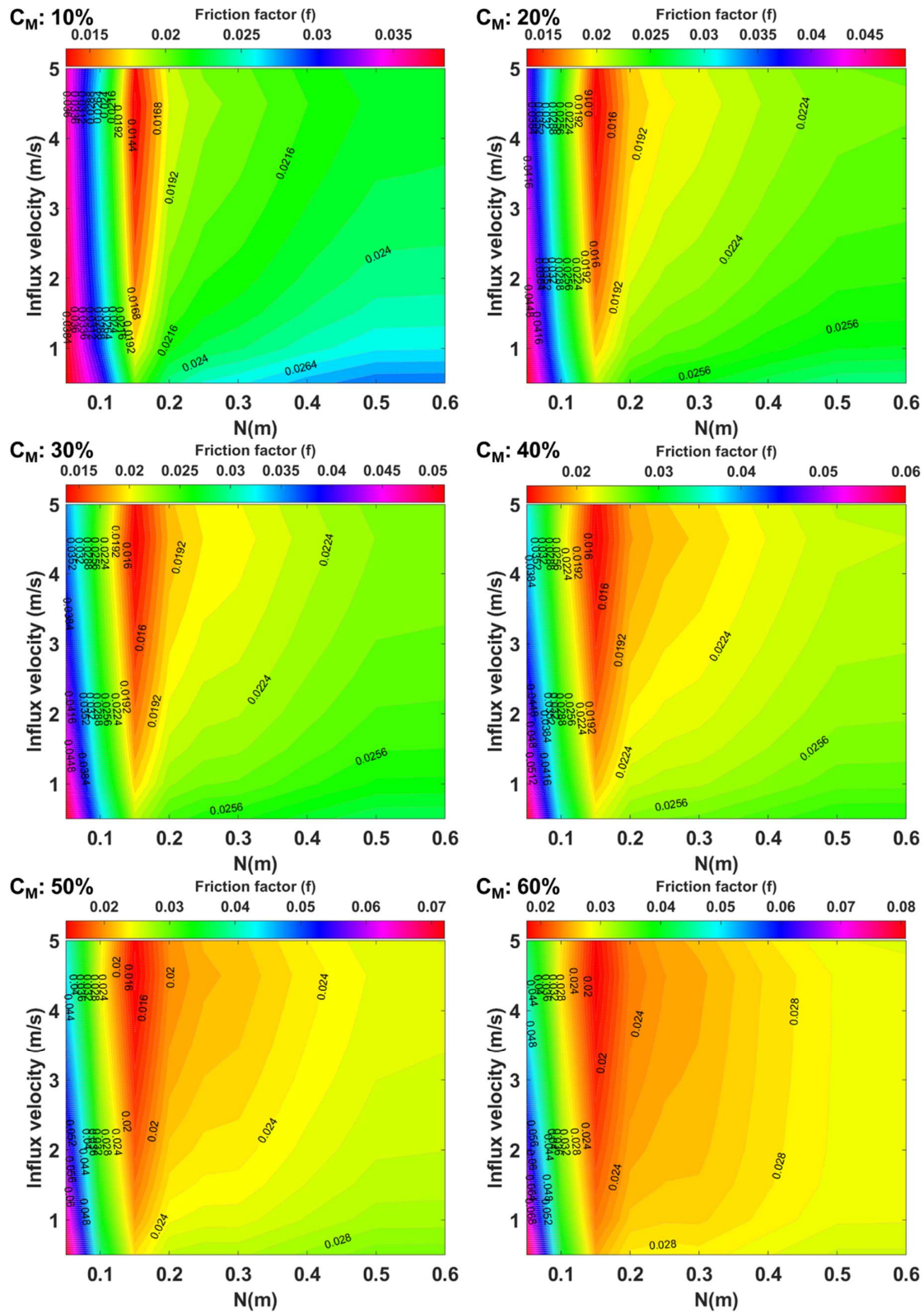


Figure 6.19 The headloss across different converging sections at different influx velocities and influx concentrations of coal.

The local resistance coefficient is dependent upon the shape/geometry of the section and evaluated as:

$$K_L = 0.0696 \cdot \sin\left(\frac{\alpha}{2}\right) \cdot (1 - \beta^5) \cdot \lambda^2 \cdot (\lambda - 1)^2 \quad 6.52$$

Here the α is the complete angle of the converging section, β is the ratio of the smaller to the larger diameter of the section and λ is the jet velocity ratio.

The jet velocity ratio is calculated as:

$$\lambda = 1 + 0.622 \cdot \left(\frac{\alpha}{180}\right)^{4/5} \cdot (1 - 0.215\beta^2 - 0.785\beta^5) \quad 6.53$$

The friction pressure loss coefficient is based on the friction factor and calculated as:

$$K_F = \frac{f_f \cdot (1 - \beta^4)}{8 \cdot \sin\left(\frac{\alpha}{2}\right)} \quad 6.54$$

f_f is the friction factor of the flow. The friction factors obtained for the different geometries of the converging sections are presented in Figure 6.19. The friction factor is observed to be the highest for the 0.05m section. There is decrease in the friction factor for the 0.1m long section. The lowest friction factor is observed in the case of the 0.15m section. The further increase in the section length leads to a gradual increase in the friction factor. The friction factor is observed to decrease with influx velocity. A gradual increase in the friction factor is observed with coal concentration up to 40% and a sudden increase is observed at 50 and 60% concentration of coal.

So, as per the results obtained in the present investigation, the 0.15m converging section is observed to be optimum design. This design is observed to have the highest pressure recovery, and least headloss, irrespective of the influx velocity and influx concentration.

6.5 Flow modelling through a diverging section:

6.5.1 Modelling of the flow domain:

6.5.1.1 Geometry:

The geometries of the diverging sections are modelled in the Design Modeler module of Ansys™ Workbench 2020 R1. Figure 6.20 shows the geometries of the diverging sections tested in the present study. The section consists of two cylindrical pipe sections of diameters 50mm and 100mm respectively and a diverging section. The 50mm diameter pipe section is

the upstream section and the 100mm diameter pipe section, located after the diverging section, is the downstream section. The horizontal length, N of the diverging section is varied from 0.05m to 0.6m, ($N = 0.05, 0.1, 0.15, 0.2, 0.25, 0.3, 0.4, 0.5, 0.6$ m).

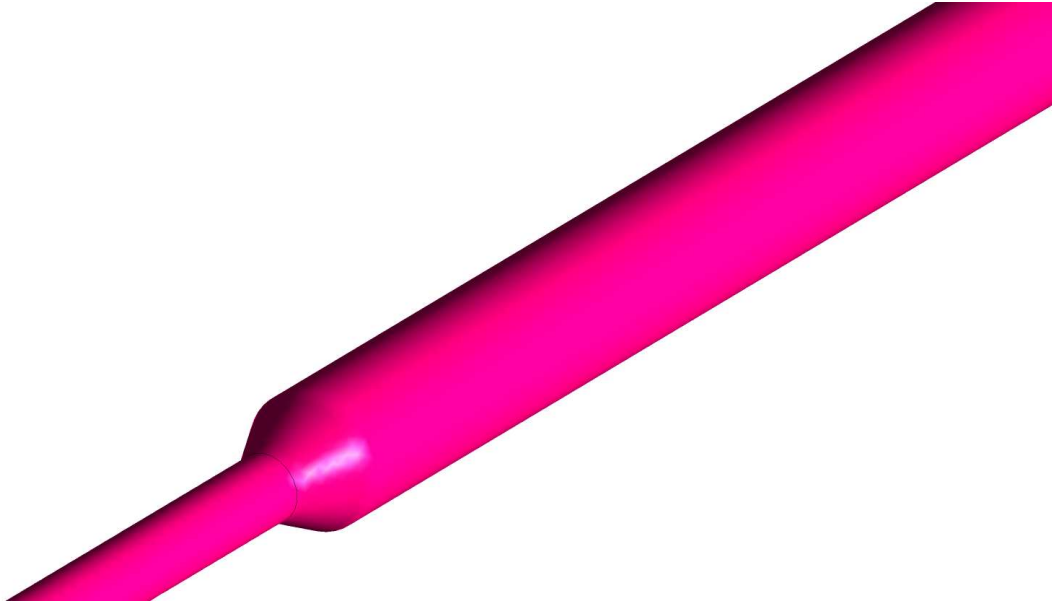


Figure 6.20 Modelled fluid domain for the 0.05m diverging section.

6.5.1.2 Meshing

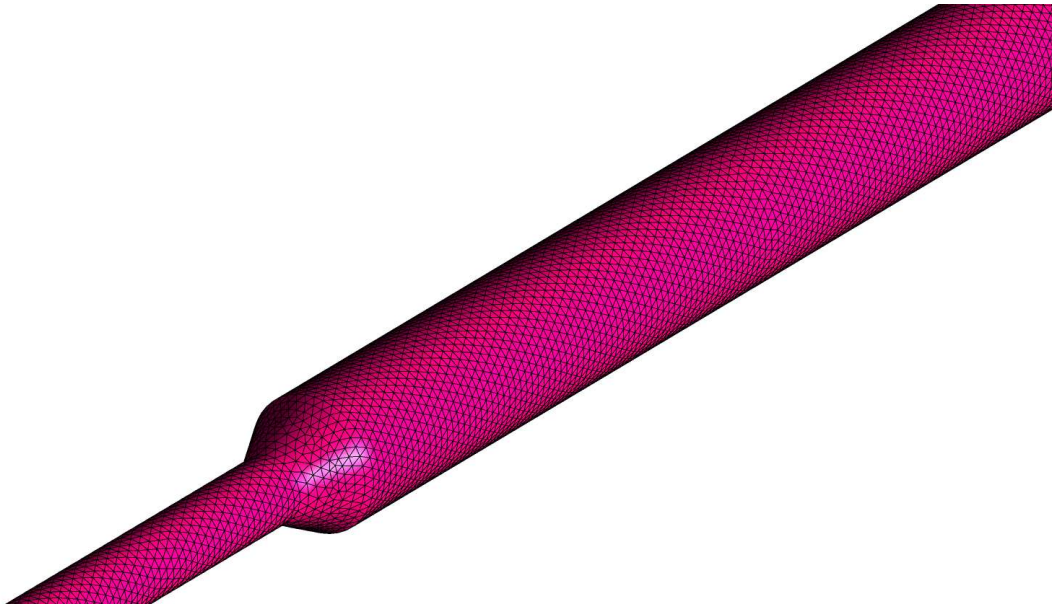


Figure 6.21 Meshed fluid domain for the 0.05m diverging section

The Meshing module of Ansys™ Workbench 2020 R1 is used to discretize the fluid domain in the present case. The meshing systems of the pipe fitted with a converging section are represented in Figure 6.21 Tetrahedron shaped cells are used in the present case, using the patch conforming method. The inflation layers also help in achieving the y^+ values of ~ 30 -35. The size of each cell is decided based on the results produced by different sized cells and represented in Figure 6.3.

6.5.1.3 Boundary conditions:

The fluid domain is resolved into three boundaries viz. inlet, wall and outlet. The pictorial representation of the domains is shown in Figure 6.22. The boundary conditions at these boundaries are given in Table 6.5.

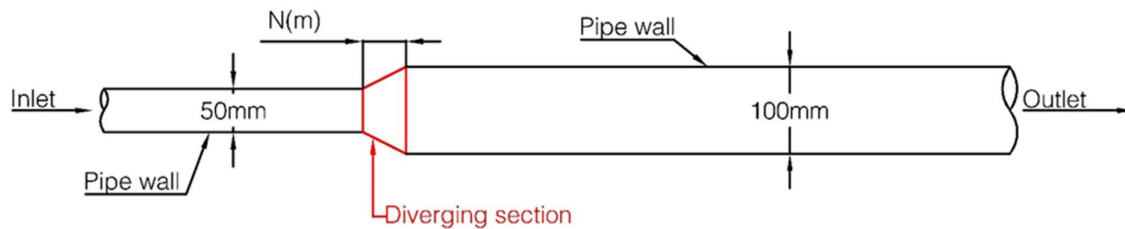


Figure 6.22 Schematic of the diverging pipe section.

Table 6.5 Boundary conditions

Boundary	Inlet	Outlet	Wall
Boundary type	Velocity inlet	Pressure outlet	Wall
Description	Cross-section of $D_1 = 50\text{mm}$	Cross-section of $D_2 = 100\text{mm}$	Cylindrical walls of 50mm and 100mm diameter pipes and conical wall of the diverging pipe section
Velocity (m/s)	Variable; 0.5-5	Not to be specified	0, due to no slip condition
Pressure (Pa)	Operating pressure of 101325	Operating pressure of 101325	Operating pressure of 101325

Turbulence (medium intensity)	Specified as Turbulence intensity of 4%, and hydraulic diameter of 50mm	Specified as Turbulence intensity of 4%, and hydraulic diameter of 50mm	Not to be specified
Volume fraction	Variable; 0.0741- 0.5193	Variable; 0.0741- 0.5193	Not to be specified
Roughness (m)	Not to be specified	Not to be specified	6.3×10^{-6}

Results

6.5.2 Visualization of separated and unseparated flow:

The results generated by the Fluent module of Ansys™ Workbench 2020 R1 software are further analyzed using the CFD post Module of Ansys™ Workbench 2020 R1. As per the previous reports published in [187], it was learned that there might be a chance of flow separation at the entry or the intermediate section of the diffuser. So, the flow behavior at the highest Reynolds number; Re (Re is the Reynolds number of the slurry/mixture relative to the pipe wall) of 19750 (at influx velocity, 5 m/s and 10% solids (w/w)) was visualized with a vector plot at the central vertical plane of the N = 0.05m diverging section. The vector plot is shown in Figure 6.23.

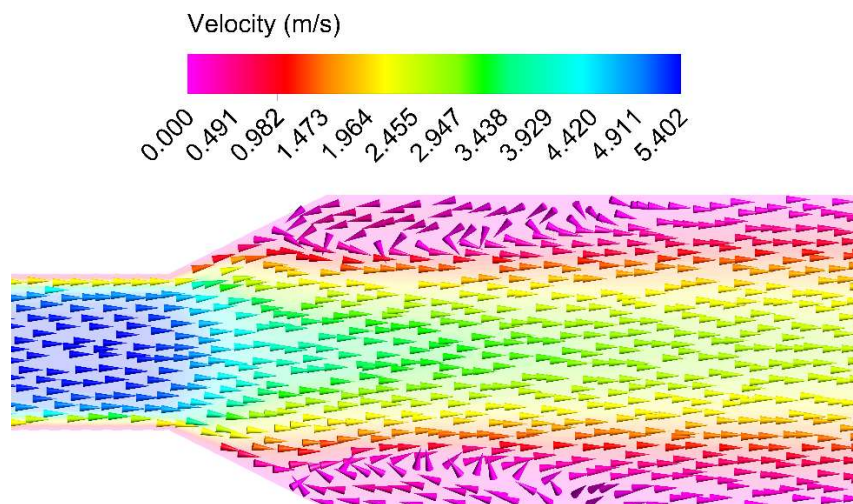


Figure 6.23 Flow separation visualized at the upstream Re =19750. The vectors facing in the direction opposite to the streamwise direction belong to the separated flow.

The magenta-colored vectors represent the separated flow. The transition of magenta to red color indicates the unseparated and the reattached flow. From the vector plot, it is observed that the separated flow follows a circulating motion and the separated region in the upper half section is larger as compared to the lower half-section. The highest Re was chosen as, for the first time, Sparrow et al. [188] report that the separation phenomenon is a strong inverse function of Re and grows stronger and adverse with the decrease in Re . If the flow separates at the high Re , then it will surely separate at lower Re as well. The flow separation was also visualized by plotting the velocity profile near the pipe wall in the separated region. The velocity profile at 10% solids concentration and varying influx velocity is represented in Figure 6.24.

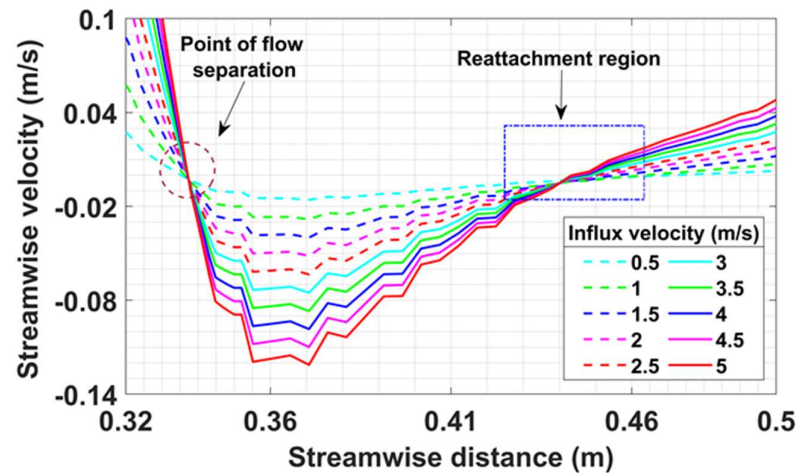


Figure 6.24 Velocity profiles in separated region near the upper wall of the pipe at 10% solids concentration.

It is learned that the flow separation takes place at a distance of 0.03793m from the start of the diverging section and the separation occurs at the same point irrespective of the influx velocity. The flow remains separated for some length of the pipe section and is again found to reattach to the mainstream. Unlike the separation point, the reattachment point is observed to be dependent on the flow velocity. The flows with higher velocities are found to attach earlier in comparison to the low-velocity flows. In further visualization of the results of the other cases, similar projections were obtained with a slight to significant variation in the separation length, ' L_s '. The ' L_s ' values for different flow velocities and solids concentration are compared in Figure 6.25.

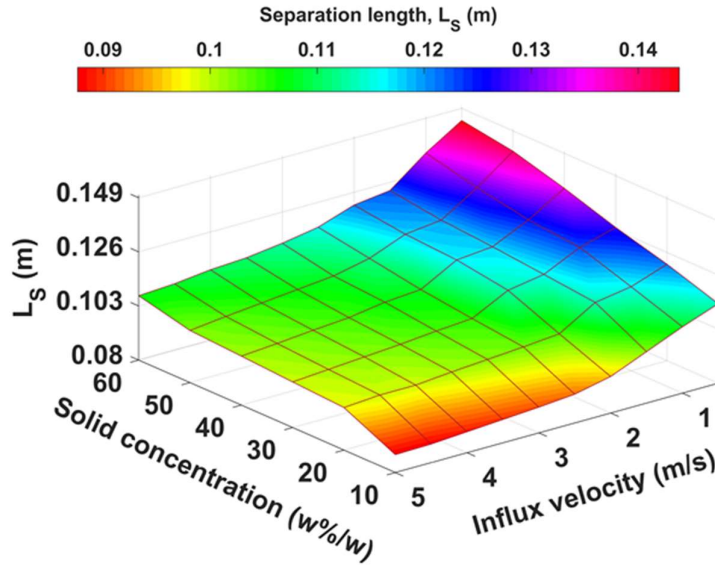


Figure 6.25 Dependence of separation length on the solid concentration and influx velocity.

The results show that the separation length shows a decrease with an increase in the influx velocity and increases with an increase in the solid concentration. So, the flow of 10% solids at 5 m/s shows the shortest separation length and the flow of 60% solids at 0.5 m/s shows the longest separation length. The other geometries ($N > 0.05\text{m}$) were also visualized at all the Re values, but no sign of separation was spotted in any of the cases.

The flow of the coal water slurry was found to separate in only one geometry of the diverging section viz. the 0.05m long diverging section. The separation length is found to be a strong inverse function of Reynolds number. The Re is calculated using the Equation 6.55 as suggested by Torrance (1963) [189] for a pseudoplastic fluid.

$$Re = 8\rho_w V_m^2 K^{-1} \left(8V_m/D\right)^{-n} = \frac{8n^n \rho_w V_m^{2-n} D^n}{2^n (3n+1)^n K} \quad 6.55$$

It is observed that, increase in Reynolds number decreases the separation length. This can be explained as in the case of the diverging section the increase volume of the section is occupied by a stagnant volume of slurry. The layer of fluid which is in contact with the pipe wall experiences a no slip condition. When this layer comes in contact with the stagnant layer, the contact becomes a free slip. Due to this, both the layers experience high shear, that generates turbulence in the flow. At this moment, the fluid is regarded as separated and the point is called as separation point (Location of the point being decided by the pressure gradient). The turbulent wake shows an extending and expanding phenomenon, which after travelling some distance

downstream is observed to impinge the bigger pipe's wall. Now, the flow is said to be re-attached and the location is called reattachment point. Downstream the reattachment point, the boundary layer starts reformation and continues till the flow is fully developed again as shown in Figure 6.26.

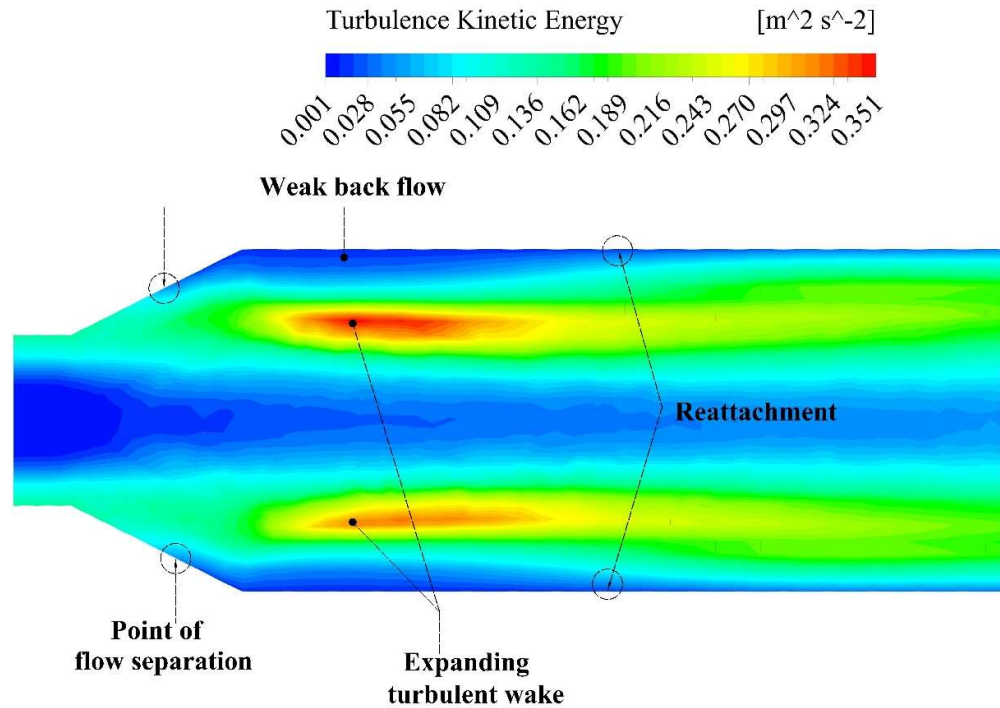


Figure 6.26 A representation of “flow separation”, “turbulent wake” and “flow reattachment” inside and downstream a diverging section

The strength and degree of expansion of the turbulent wake is found to be increasing with increase in the Reynolds number. With increase in the Reynolds number, the turbulent wake is observed to impinge earlier on the expanded pipe wall. So, the flow is found to reattach earlier in the case of the higher Reynolds numbers. This explains the trends obtained in Figure 6.25 and the separation length can be referred as an inverse function of the Reynolds number. Similar flow patterns of the turbulence kinetic energy can be seen in the research reported by [190]

A careful observation of the flow patterns in the other sections showed no sign of flow separation. This happened because the pressure gradient in the all other diverging section is not adverse to separate the flow. So, the flow remains attached throughout the pipe section.

6.5.3 Performance characteristics of the diverging section

It is evident that when the fluid stream enters a diverging section, the first thing that happens is the conversion of the velocity head or the kinetic energy of the fluid into the pressure head or the pressure energy. The performance characteristics of the diverging section refers to the fact that how efficiently the diverging section is performing at this conversion process. To analyse the performance of the diverging section, the parameters, viz. head-loss across the diverging section, pressure recovery coefficient, ' C_P ' and the volumetric efficiency of the diverging section are considered.

6.5.3.1 Pressure recovery characteristics

The pressure recovery characteristics of the diverging section are analyzed based on the pressure recovery coefficient ' C_P '. Ideally, if no frictional drag exists, the reduction in the velocity head is supposed to reflect into the increase in the static pressure head. Mathematically, ' C_P ' is defined as the ratio of rise in the static pressure to the dynamic pressure at the inlet of the diverging section as represented in equation 6.56.

In an ideal case, the ideal pressure recovery coefficient is expressed as:

$$C_{Pi} = \left(\frac{P_2 - P_1}{\rho \bar{V}_1^2 / 2} \right)_i = \frac{\bar{V}_1^2 - \bar{V}_2^2}{\bar{V}_1^2} = \left(1 - \frac{1}{AR^2} \right) \quad 6.56$$

Where P_1 and P_2 are the static pressures at the inlet and outlet of the diverging section and $\bar{V}_1$ and $\bar{V}_2$ is average velocities of the mixture stream at the inlet and outlet of the diverging section.

AR is the area ratio and is defined as:

$$AR = \left(\frac{D_2}{D_1} \right)^2 \quad 6.57$$

From the above expressions, it is evident that the ideal pressure recovery coefficient depends only on the area ratio and the length of the diverging section plays no role in the pressure recovery. But in an actual case, there is a loss in the static pressure due to the friction or drag associated with the flow in a closed pipe. The actual pressure recovery coefficient is analyzed on the basis of the actual static pressures taking into account the frictional losses and losses due to flow separation (in case of poor section design).

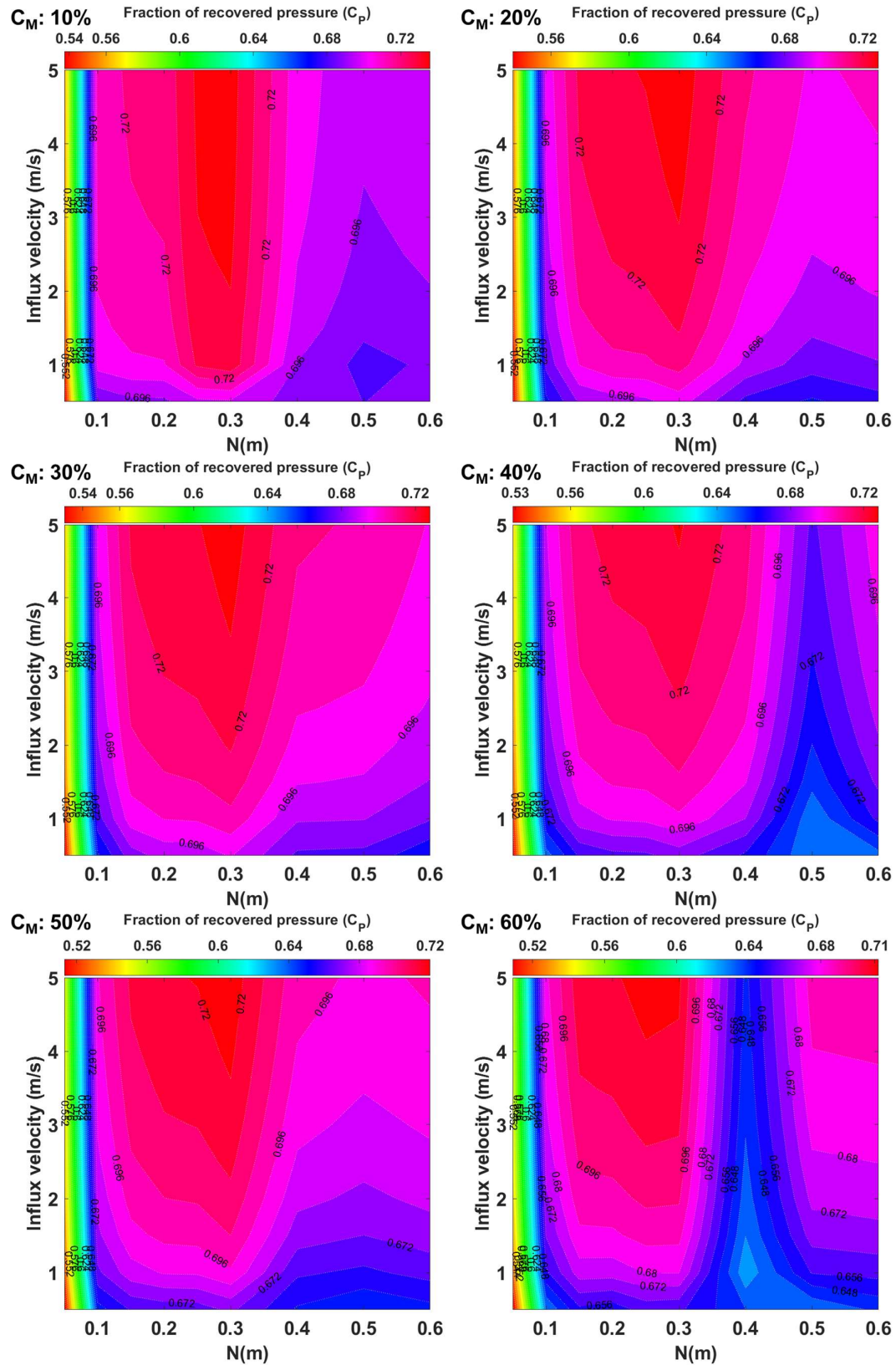


Figure 6.27 Pressure recovery coefficient of the different diverging sections at different influx velocities and solid concentration.

The actual pressure recovery coefficient is defined as:

$$C_P = \frac{P_2 - P_1}{\rho \bar{V}_1^2 / 2} \quad 6.58$$

The results obtained using the mathematical expression in equation 6.58 are presented in Figure 6.27. The results of 'C_P' are projected w.r.t. influx velocity and the length of the diverging section using surface plots. The 'C_P' values are also calculated for different slurries of solid concentration varying in the range of 10-60% and are presented in each subplot (a-f). The 'C_P' is observed to be minimum for the 0.05m long diverging section irrespective of the solid concentration. Out of different diverging section lengths, 0.3m length of the diverging section shows the greatest pressure recovery for the velocity range of 0.5-5m/s and the solid concentration range of 10-60%. The lowest pressure recovery was observed in the case of the 0.05m long diverging section. This is possibly due to the losses associated with the separation of the flow in the poorly designed diverging section.

As the section length is increased to 0.1m, an abrupt increase in 'C_P' is observed. This increase in the pressure recovery is due to the elimination of the separated region, which accounts for the losses due to eddies. The 'C_P' is observed to increase till the length of 0.3m of the diverging section and is observed to fall thereafter. As the length of the diverging section is increased, two significant parameters are observed to change their nature. These are the velocity gradient and the wall friction. As the length of the diverging section increases, the velocity gradient shows a decrease whereas a notable increase in the wall friction can be expected due to the proportional increase in the wall surface area. The increase of 'C_P' till N=0.3 is due to the velocity gradient being dominant and for N>0.3, the wall friction plays its role, thereby decreasing the 'C_P'. So, the 0.3m section appears to be a balance between the two parameters and shows the greatest pressure recovery.

6.5.3.2 Headloss across the diverging section:

The head loss across the diverging section is accounted as the loss of the velocity head due to flow separation or eddies or frictional drag or combination of these, which otherwise was to be added to the pressure head in an ideal situation. The head loss is calculated indirectly from the 'C_P' values using the following expression:

$$H_L = \frac{(C_{Pi} - C_P)}{2g} \bar{V}_1^2 \quad 6.59$$

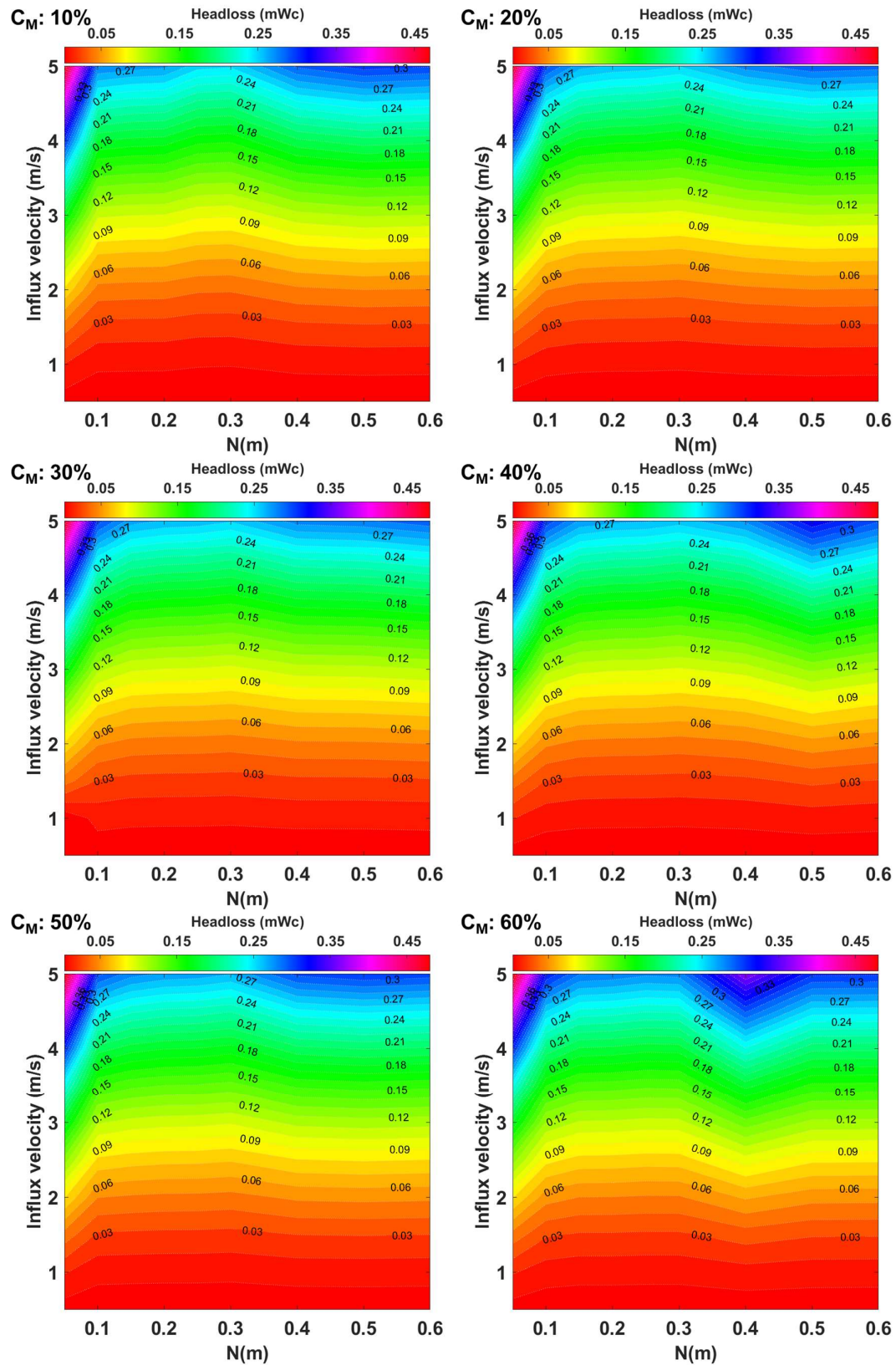


Figure 6.28 Head-loss across different diverging sections at different influx velocities and solid concentration.

The head-loss, H_L obtained from equation 6.59 is in terms of length of the slurry column. For the sake of uniformity, the H_L is converted in terms of length of the water column. The results of head-loss characteristics are presented in Figure 6.28.

The trends of head-loss are the inverse of those obtained in the previous section. The results show that the maximum head-loss is reported in the 0.05m long diverging section, due to the separation of the flow in the section. The head-loss is then observed to diminish until the $N=0.3\text{m}$ due to the decrease in the velocity gradient being greater than the wall friction. But the head loss is observed to increase for $N>0.3\text{m}$ due to a stronger effect of wall friction in comparison to the velocity gradient.

The head loss is found to be maximum across the 0.05m long diverging section. This is possibly due to the loss of the fluid's dynamic head during the flow separation and formation of eddies. Based on the Darcy-Weisbach equation, the head loss is found to increase with increase in the flow velocity. Similar trend is also observed with the solid concentration. The increase in the solid concentration leads to a linear increase in the slurry viscosity up to a concentration of 30% by mass. Beyond 30% solid concentration, the viscosity follows the power law, a substantial growth in the viscosity is observed. This leads to the increase in the head-loss with increase in the solid concentration. Irrespective of the flow velocity and the solid concentration, the head-loss follows a decreasing trend with the section length up to 0.3m and starts increasing for greater section lengths. This can be described as, the increase in the section length leads reduction in the velocity gradient but simultaneous increase in the skin friction. Up to the 0.3m length of the section, out of the two, effect of the velocity gradient is dominant and a reduction in the head-loss is witnessed. The sections of the greater length have relatively greater surface area, that is expected to contribute substantially to the skin friction. Now, the skin friction being dominant, leads to increase in the head loss. So, the 0.3m is optimum length for lowest possible head loss in the present situation(s).

6.5.3.3 Volumetric efficiency of the diverging sections:

As discussed earlier, different parameters play their key role in the design of a diverging section. Volumetric efficiency is another important parameter, which defines the efficiency of the diverging section in delivering the required volume flow rate of the fluid or slurry in a given case. While defining the volumetric efficiency, a term known as 'Idealized equivalent diverging section', is introduced, that will have a pressure recovery coefficient equal to that of a real diffuser, but the flow inside it will be frictionless. The schematics to all three cases are

presented in Figure 6.29.

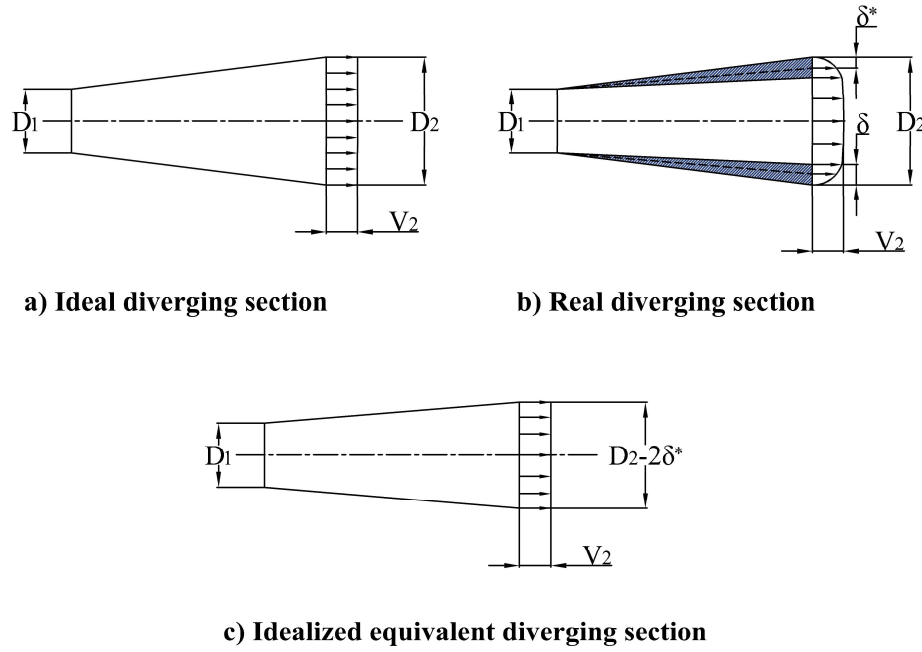


Figure 6.29 Schematics and velocity profiles of a) ideal, b) real and c) idealized equivalent diverging sections.

Here Figure 6.29 a) shows an ideal diverging section, in which the fluid flow is frictionless and the fluid flow takes place through the whole vicinity of the diverging section. In case b), a real diverging section is shown, with a formation of a boundary layer of thickness ‘ δ ’ including a displacement layer of thickness ‘ δ^* ’. It can be seen that the fluid inside the displacement layer is almost stagnant, which is supposed to be contributing to the actual loss of volume of the fluid finally delivered at the outlet. So, the real diverging section of case b) is now simplified to an idealized equivalent diverging section, which is frictionless and now the flow takes place only in the ‘ $D-2\delta^*$ ’ region. Where ‘ D ’ and δ^* are diameter and displacement thickness at a location inside the diverging section such that $D_1 < D < D_2$. So, in a real diverging section, the flow is taking place through the ‘ $D-2\delta^*$ ’ cross-section only.

From the above discussion, the following can be defined:

$$C_p = C_{pie} = \left(1 - \frac{1}{AR_e^2}\right) \quad 6.60$$

Here C_p & C_{pie} is the pressure recovery coefficients of real and idealized equivalent diverging sections. AR_e is the area ratio of the idealized equivalent diverging section. The equation 44 can be rearranged to equation 6.61:

$$AR_e = \sqrt{\frac{1}{1 - C_p}} \quad 6.61$$

If Q and Q_i are the actual/idealized equivalent and ideal volume flow rates of the slurry through the diverging section, then, volumetric efficiency η_V can be defined as:

$$\eta_V = \frac{Q}{Q_i} = \frac{V_2 A_{2e}}{V_2 A_2} \quad 6.62$$

$$\text{So,} \quad \eta_V = \frac{AR_e}{AR} \quad 6.63$$

Substituting AR_e from 6.61 in 6.63, and rearranging gives:

$$\text{So,} \quad \eta_V = \frac{1}{AR\sqrt{1 - C_p}} \quad 6.64$$

The equation 6.64 shows the relationship between the volumetric efficiency, area ratio and pressure recovery coefficient. As in the present study, the area ratio is fixed at 4, so the volumetric efficiency is entirely dependent on the pressure recovery coefficient of the diverging pipe section. The volumetric efficiencies of different diverging sections at the velocity range of 0.5 to 5 m/s and solid concentration of 10 to 60 % have been plotted on 2D contours and represented in Figure 6.30.

It is observed that the 0.05m long diverging section is the least efficient in delivering the required volume of slurry, at all influx velocities and solid concentrations. The trends of the volumetric efficiency are quite analogous to those observed in section 6.5.3.1. At all influx velocities and solid concentrations, the volumetric efficiency is found to be increasing with 'N' till $N=0.3$ and found to be decreasing for $N>0.3$. For the sections $0.05 < N \leq 0.3$, the velocity gradient is decreasing, which indicates a proportional decrease in the boundary layer thickness, ' δ ' and displacement layer thickness ' δ^* '. But, for the cases, where $N>0.3$, the increased frictional drag leads to the increase in both ' δ ' and ' δ^* '.

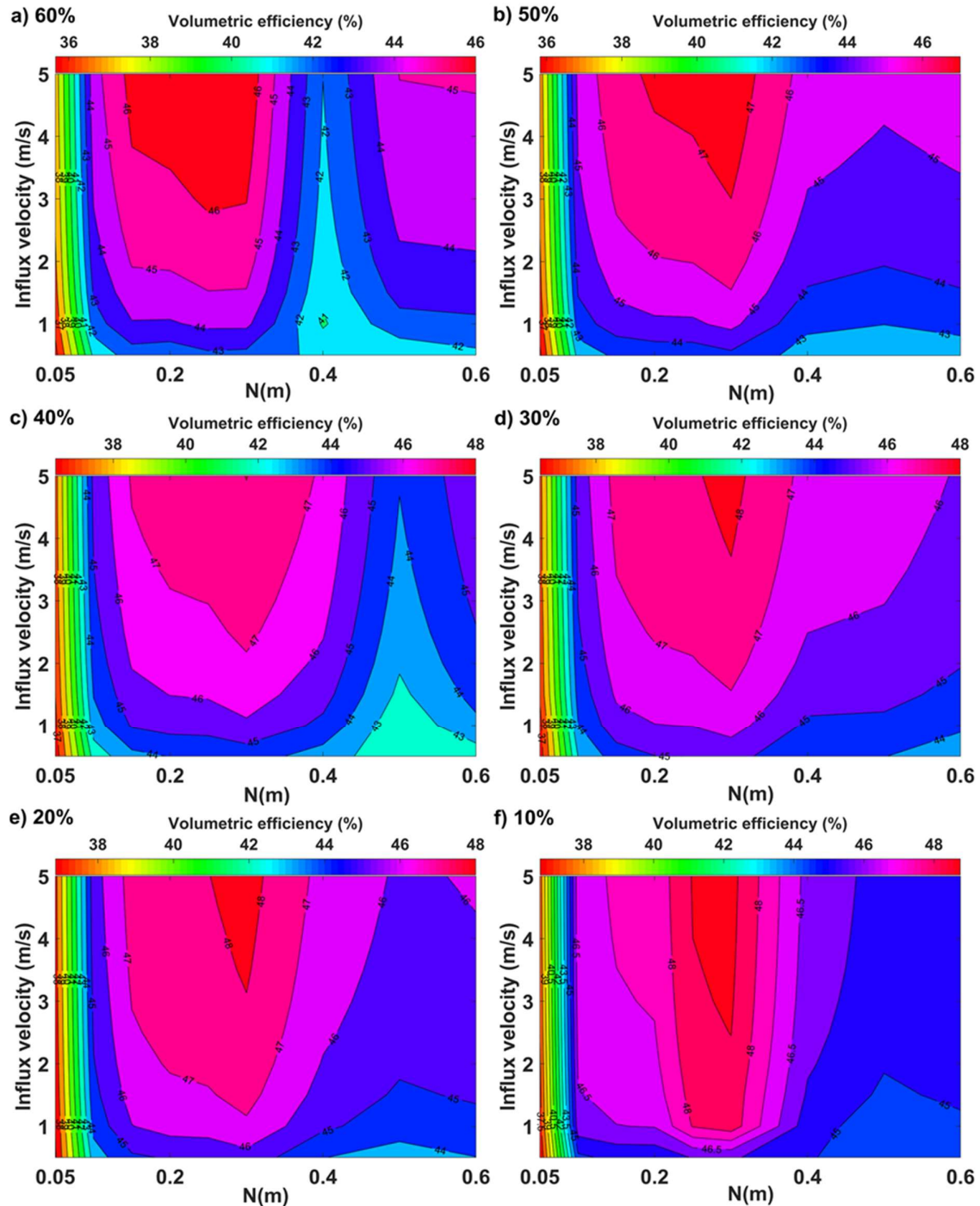


Figure 6.30 Volumetric efficiency across different diverging sections at different influx velocities at a) 60%, b) 50%, c) 40%, d) 30%, e) 20% and f) 10% solid concentration

The term volumetric efficiency is introduced to quantify the volume fraction of the stagnant and flowing fluid (slurry). In the present study the volumetric efficiency refers to the fraction/percentage of pipe/section volume occupied by the flowing fluid. In the case of 0.05m diverging section, the region of developed (flowing) fluid flow is very small, as almost all the

section experiences flow separation (stagnant state). As the separated region contributes only to secondary flow or eddies generation, this leads to very poor performance in terms of volumetric efficiency. Earlier in case of longer sections, the fluid flow is found to be attached throughout the section, so these sections have much greater volumetric efficiency as compared to the 0.05m section. The sections of up to 0.3m length, due decreasing velocity gradient, experience decrease in the displacement thickness, ' δ^* ' (actually composed of almost stagnant fluid due to excessive shear stress at the wall) and lead to increase in the volumetric efficiency of the section. The sections of length greater than 0.3m length, due to substantially increasing skin friction, experience thickening displacement layer. So, the volumetric efficiency of these sections starts decreasing. The 0.3m section shows greater volumetric efficiency as compared to all other sections.

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6.6 A solution to address the issue of particles' settling:

6.6.1 Introduction:

As the issue of settling is critical, so the scientific community has provided many solutions to mitigate the settling in heterogenous slurries. Previously authors have used variety of charged chemicals, dispersing agents and stabilizers. The studies indicate the improvement in the stability by all the above-mentioned synthetic substances. The use of dispersants and stabilizers have also proved to be beneficial in other bulk solids such as fly ash, bottom ash, sand, etc., which being hydrophilic are more susceptible to settling. Some thermal treatment techniques such as thermal heating in a furnace up to sub-pyrolysis temperature [191],

hydrothermal treatment [192–194], mechanical thermal expression [195,196] and microwave thermal treatment [197] have been proposed and successfully used.

All the above-mentioned solutions aim at increasing the time for which the solid particles remain suspended in the carrier medium. The chemical dispersant-based solutions have versatile applications and can be used for variety of solids. The action of the dispersant is based on the magnitude and the polarity of the charge present on the surface of the solid particle, that vary across different types of solids. So, a particular dispersant/surfactant may not be used for variety of solids. In the solid's disposal applications such as disposal of fly-ash into ash pond, these chemicals being non-biodegradable, can stay in the ash ponds for a long time, thereby polluting the surroundings. The thermal treatment of the coal has produced results with significant improvement; however, such procedures cannot be implemented on the other inorganic solids such as fly ash, bottom ash, sand, mineral tailings, etc.

In some applications, the solids to be transported are required in their original form (without any modification in the physico-chemical or surface properties), in such cases the present available solutions appear to be non-beneficial. In such case, the only solution is to increase the turbulence of the flow. Turbulence can be increased in many ways. These can be:

- Increasing the flow velocity; that it is expected to increase the headloss by many folds.
- Placing baffles close to the wall to enhance turbulence in the flow; this could create certain adverse pressure zones, where the flow can easily separate and create eddies leading significant increase in the headloss.

The above two solutions appear to be energy intensive, as these are associated to significant energy losses. In such a situation, the use of a twisted pipe sections is proposed. Twisted pipe sections are the pipes having helical impressions on the wall, that help in generating vorticity with axis aligned to the pipe axis, without any external energy source. The twisted pipes have been effectively used in heat transfer application, where these pipes facilitate the mixing of the fluid so that uniform temperature and increase in heat transfer rate could be achieved [198]. Previously, such twisted pipes have been used by Wood et al. 2001, to mitigate erosion in bends when particle laden fluids are transported through the pipe lines. Their study reported scattering of particles in the downstream region of the twisted pipe insert, that prevented particles from focusing the same region of the bend, thereby resulting in the

mitigation of the erosion [199]. It is expected that the use of twisted pipe sections can be beneficial for re-dispersion of the settled solid particles.

The previous applications of the twisted pipes in the enhancement of heat transfer in heat exchanger pipes and mitigation of erosion wear in pipe bends indicate the ability of twisted pipes in generating turbulence in the flow. However previously these pipes have never been used to disperse the already settled particles to prevent choking of the pipelines and to deliver a pseudo-homogenous slurry. So, the present study pursues with an aim to design twisted pipe sections, that are effective in re-dispersing the settled solid particles, and efficient enough to make the proposed solution a sustainable solution. The study is expected to provide the detailed information on the flow dynamics of the slurry through the twisted section(s) and the downstream untwisted pipe.

6.6.2 Flow domain modelling:

6.6.2.1 Geometry:

The geometries of the horizontal and the twisted pipe sections are modelled in the Design Modeler module of Ansys™ Workbench 2020 R1. The key parameters for the geometrical variation considered in the present study are shown in Table 6.6. The geometries of the twisted pipe sections used in the present research are presented in Figure 6.31. To ensure a smooth transition from the untwisted to the twisted section and vice-versa, small sections of 10 mm each have been used. The geometries of these sections are shown in Figure 6.31. The front sectional view of the twisted section is also shown in Figure 6.31.

The flow areas of the untwisted circular pipe and the all the twisted sections have been kept exactly same. In the preliminary studies, the authors found a sudden acceleration of the flow inside the twisted section. Simultaneously, flow separation is also observed downstream the section, due to significant deceleration of the flow. This is an indication of poor section design as the flow separation is usually associated to a significant pressure loss, that is not desirable in the present study.

It is evident from the front sectional view of the twisted sections that on increasing the number of lobes, the twisted section tends to approach a circular cross-section. As a result, the geometries with greater number of lobes are expected to produce less turbulence. Figure 6.31 shows the geometries of the 200mm long twisted sections. The other geometries are similar to these and have longer twisted sections.

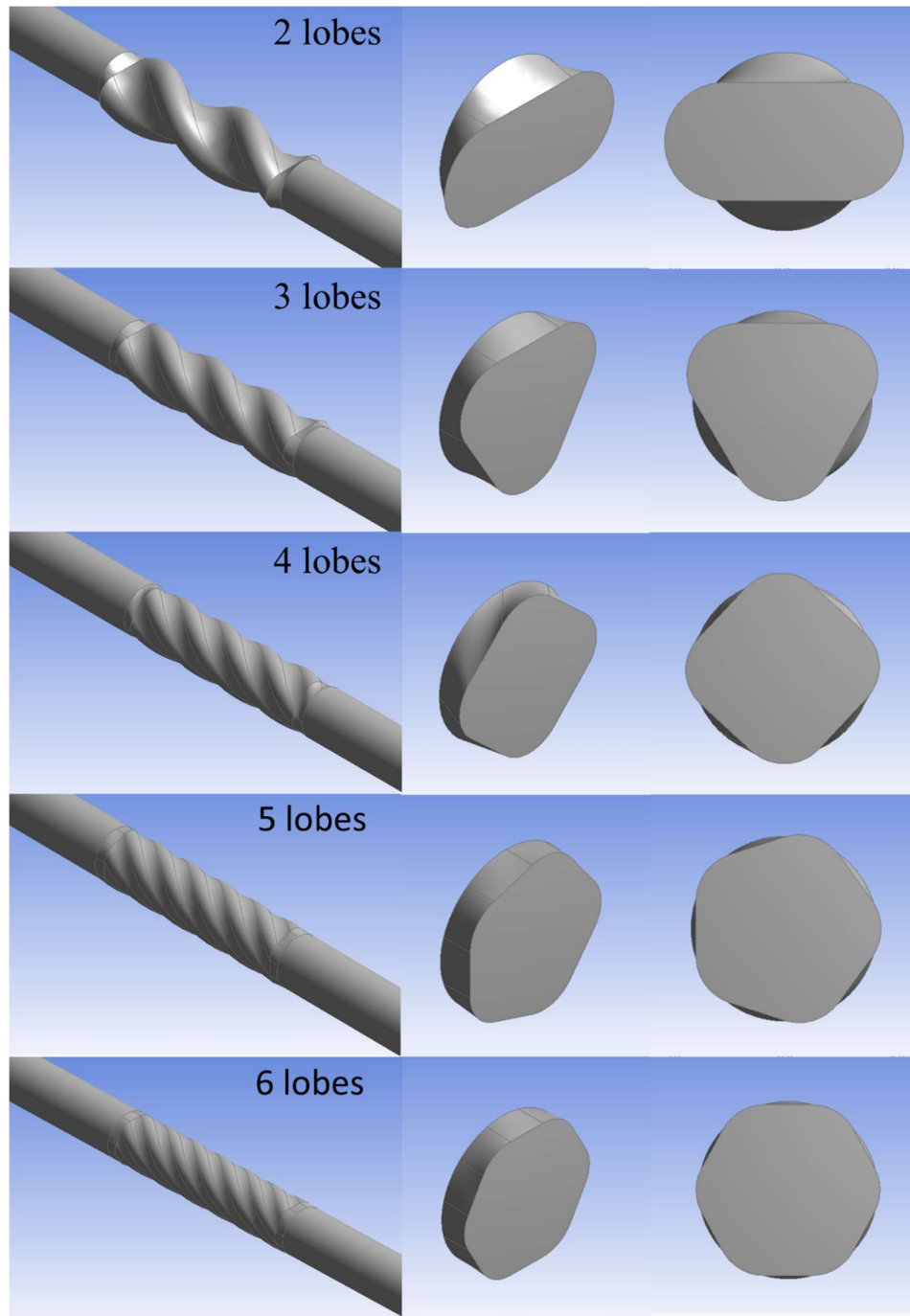


Figure 6.31 Conceptual drawings of 2, 3, 4, 5 and 6 lobes geometries along with the representation of the small 10mm long sections for transition from circular cross section to multilobes cross section.

Table 6.6 Design parameters

Twisted section	Key parameter	
	No. of Lobes	Section length (mm)
	2	200
	3	400
	4	600
	5	800
	6	

6.6.2.2 Meshing:

The discretization task for all the above mentioned 21 geometries (1 conventional untwisted pipe & 20 pipes with a twisted section) of the pipes is performed using the inbuilt meshing module of AnsysTM Workbench 2020 R1. The key parameters of the meshing system are presented in Table 6.7.

The meshing system of the all the geometries of section length 200mm and the untwisted pipe have been shown in Figure 6.32a and Figure 6.32b. The patch confirming method is used to convert the flow domain into tetrahedron shaped cells. The growth rate of 1.15 ensures a thin inflation layer close the pipe wall, that increase while moving radially inwards by a factor of 15%.

Table 6.7 Details of the meshing system

Parameter	Untwisted pipe	Twisted section
Cell size (mm)	5	5
Cell type	Tetrahedrons	Tetrahedrons
No. of Inflation layers	8	8
Inflation transition ratio	0.237	0.237
Inflation growth rate	1.15	1.15

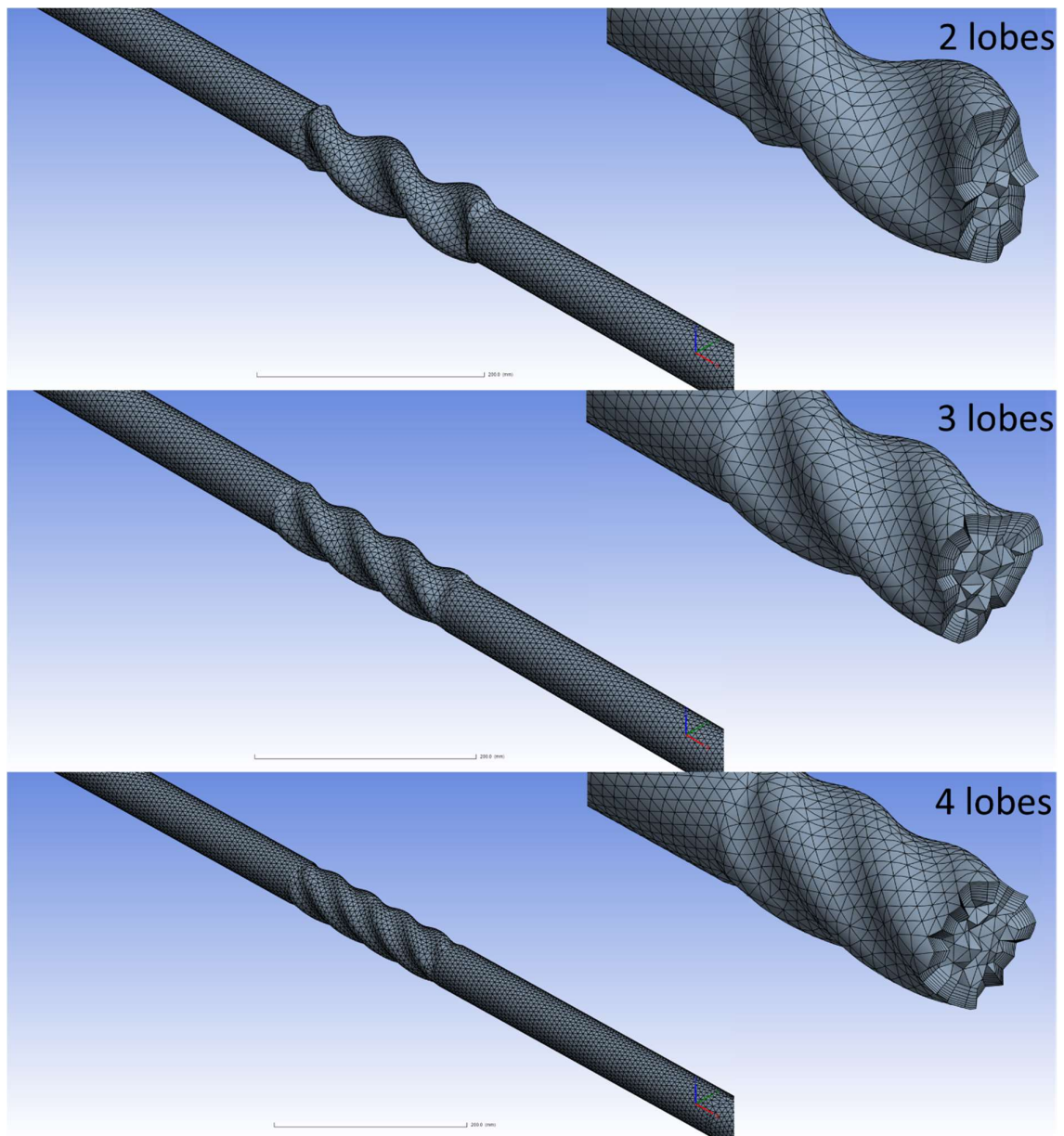


Figure 6.32a Complete mesh of untwisted and twisted pipes and sectioned mesh of the twisted geometry, showing the structure of inflation layers of 2, 3 and 4 lobes geometry.

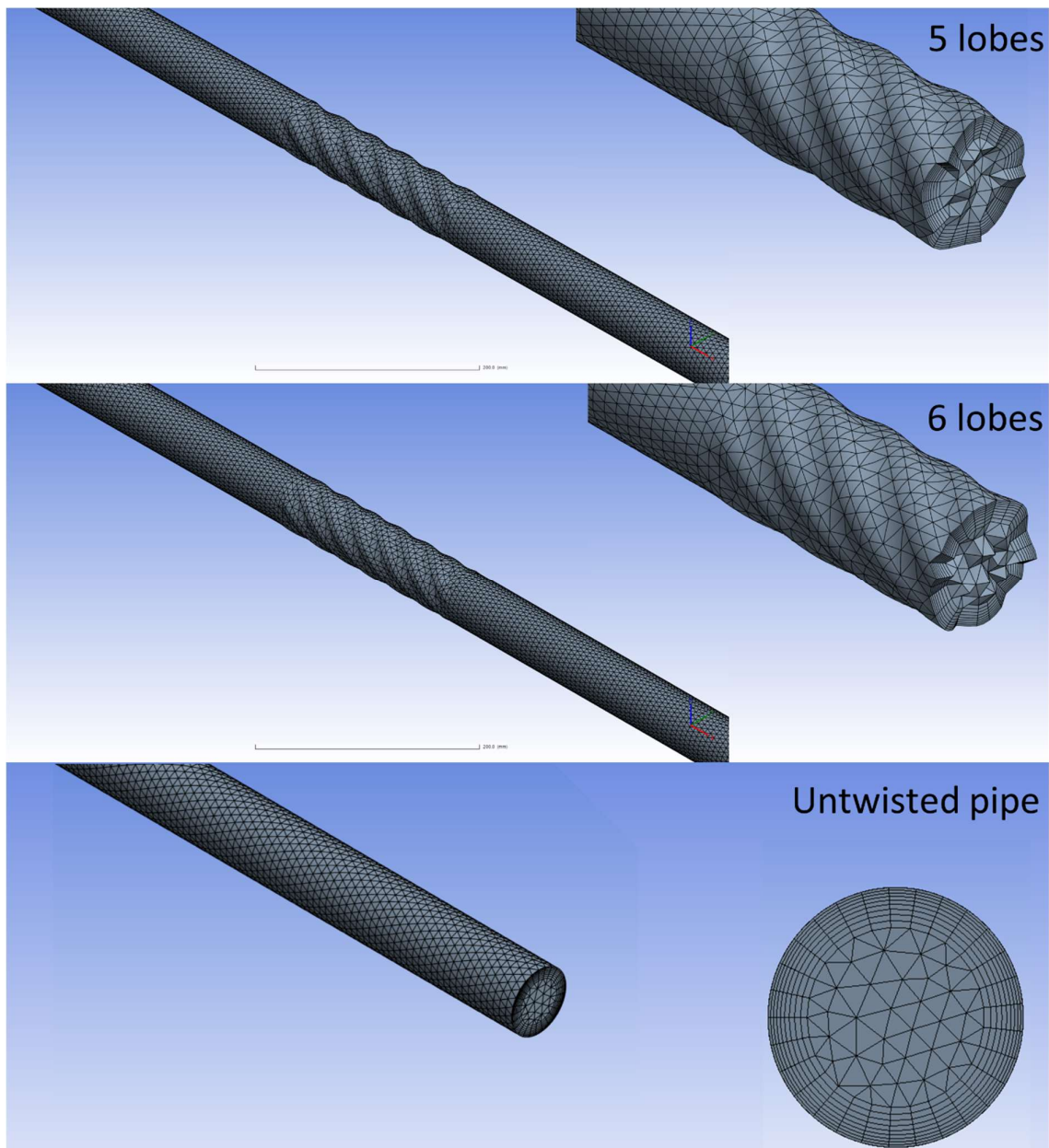


Figure 6.32b Complete mesh of untwisted and twisted pipes and sectioned mesh of the twisted geometry, showing the structure of inflation layers of 5 and 6 lobes geometry and untwisted pipe.

6.6.2.3 Boundary conditions:

The flow domain is assigned three different boundaries viz. Inlet, outlet and wall. These boundary conditions are presented in Table 6.8.

Table 6.8 Boundary conditions

Boundary	Inlet	Outlet	Wall
Boundary type	Velocity inlet	Pressure outlet	Wall
Velocity (m/s)	0.5	Not to be specified	0, due to no slip condition
Pressure (Pa)	Operating pressure of 101325	Operating pressure of 101325	Operating pressure of 101325
Turbulence (medium intensity)	Specified as Turbulence intensity of 4%, and hydraulic diameter of 50mm	Specified as Turbulence intensity of 4%, and hydraulic diameter of 50mm	Not to be specified
Volume fraction	Variable; 0.3244-0.5193	Not to be specified	Not to be specified
Roughness (m)	Not to be specified	Not to be specified	6.3×10^{-5}

The primary phase in the present study is water and the secondary phase is coal, that is assumed to be granular. The density of water is 999.9 kg/m^3 and is considered a Newtonian fluid having a viscosity of 0.001003 Pa.s . The coal with a median diameter, d_{50} of $49.5 \mu\text{m}$ and having a density of 1388 kg/m^3 is used in coal-water system. The coal-water system is modelled as a power law fluid (non-Newtonian fluid). The parameters viz. flow index; n , consistency index; K , maximum viscosity and minimum viscosity are obtained by fitting curves to the rheological characteristics obtained using RheolabQC rheometer.

Results:**6.6.3 Flow structure visualization:**

The flow structure of the untwisted and twisted pipes is presented in Figures 6.33 and Figure 6.34.

6.6.3.1 Streamlines of the flow:

Figure 6.33 shows the streamline projections on the trajectories followed the flow in different designs of the pipe. Due to significant alteration in the flow path, the maximum local acceleration is observed in the case of two lobe geometry. This results in the greater turbulence kinetic energy in the vicinity. In the 2 lobes geometry, as soon as the flow enters the twisted region, it starts rotating along with the twisted geometry. As the slurry is forced towards the pipe wall, as a result the flow closer to the wall is found to retard, however due to flow continuity the flow at the pipe center is found to accelerate and the maximum flow velocity of 0.72 m/s is observed at the pipe center in the two lobes geometry. In the case of the 3 lobes geometry, the flow continues to flow in the axial direction with some disturbances, however in the first 33% length of the twisted section, the flow starts rotating about the streamwise direction. Unlike the previous case, there is lesser deceleration of the near wall flow as a result the maximum velocity of 0.62 m/s is observed in the case of the 3 lobes twisted pipe geometry.

In the case of 4 lobes geometry, the flow takes even more time to align itself with the twisted geometry and in this case, the flow rotation starts at around the 50% of the section length. However, in this case, unlike previous cases, the secondary flow generated by the twisted section is significantly weak as a result the rotation is observed to diminish/damp in the downstream region. In the case of 5 and 6 lobes geometries, there is hardly any rotation in the flow. Instead of rotation, the flow is found to be slightly inclined to the streamwise direction (only inside the twisted geometry) and is found to align itself with the flow direction while exiting from the twisted geometry. In the further downstream regions, the flow is found to re-incline in a direction opposite to the earlier inclination. This appears to be a counter-balance effect to the earlier momentum imparted to the flow by the twisted section. So, in the 5 and 6 lobes geometries, there is no net rotation in the flow.

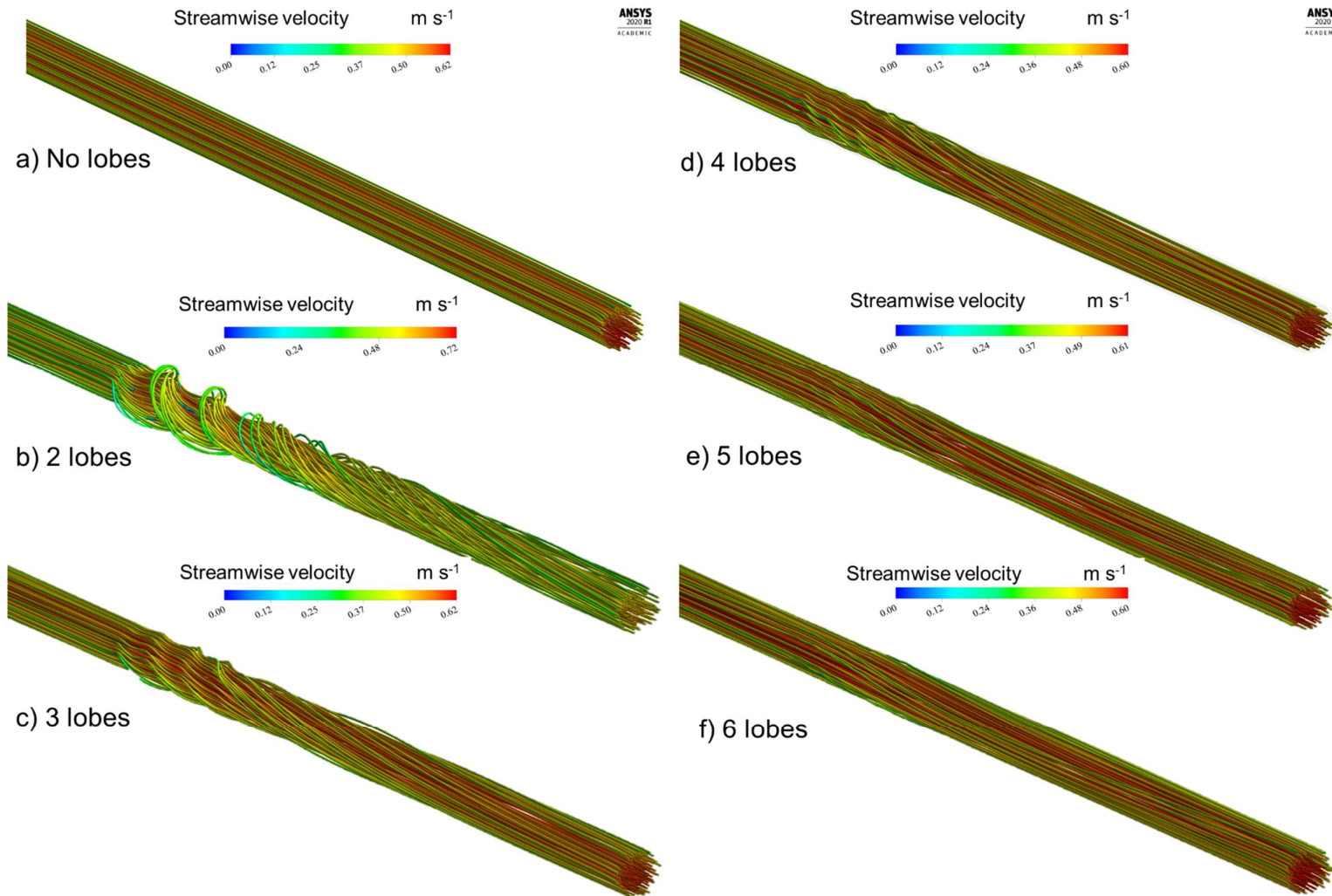


Figure 6.33 Effect of twisted geometries on the flow structure presented with the streamlines.

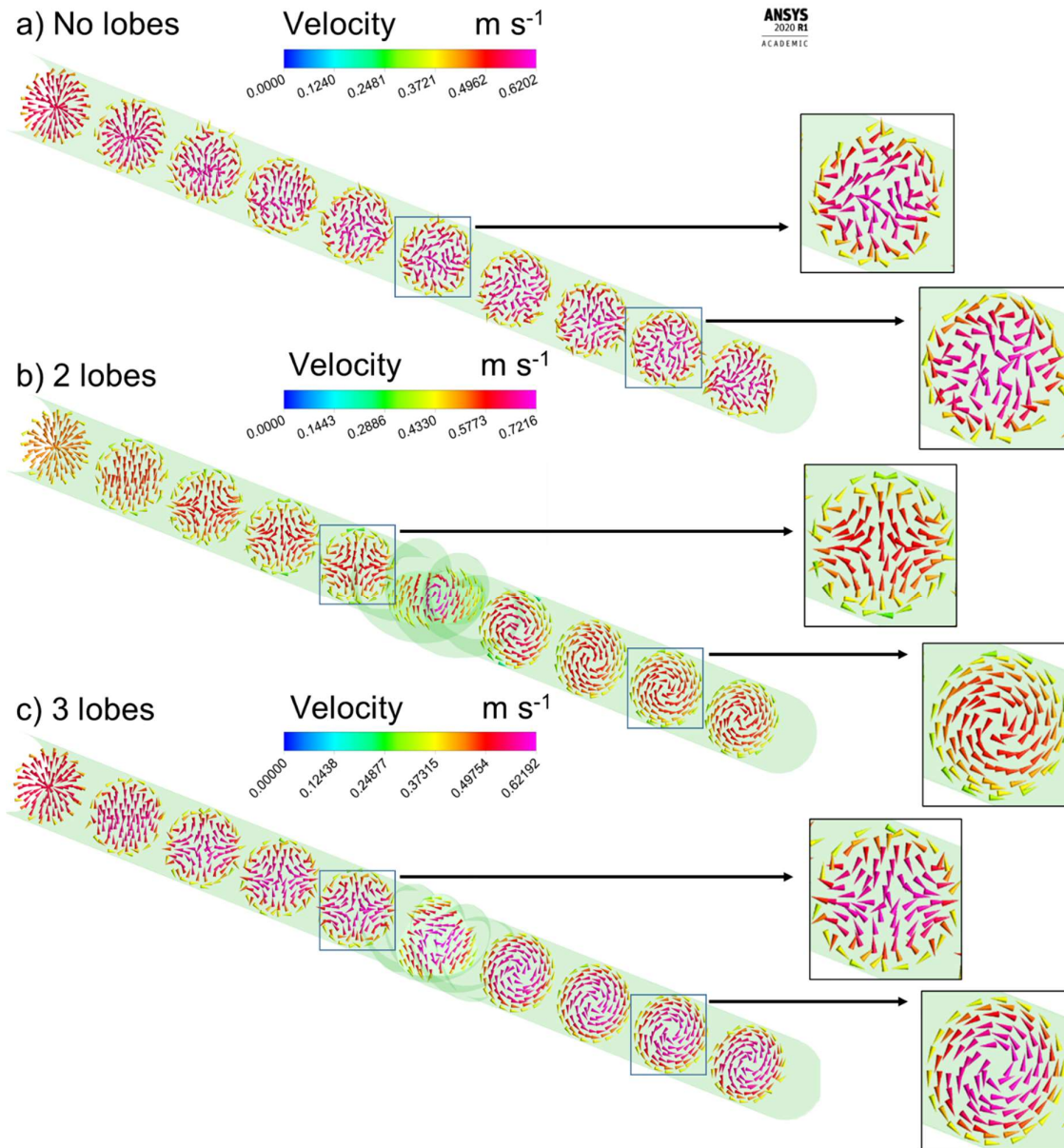


Figure 6.34a Secondary flow structure at a series of vertical planes represented by vector plots for a) untwisted (0 lobes), b) 2 lobes and c) 3 lobes twisted geometries.

Figure 6.34 (a and b) represents the secondary flow on a series of equally spaced vertical planes placed perpendicular to the streamwise direction. The vector plots in the case of the untwisted pipe show random orientation of the vectors at all the planes (in the developed region). As a result, the flow in untwisted pipe is found to be irrotational (no net rotation). In the case of the 2-lobe geometry, at the plane located just the upstream of the twisted section, an irrotational flow is observed as 4 self-cancelling vortices are formed. The flow gets a well-defined rotation inside the twisted pipe section and maintains its rotation in the far downstream regions also.

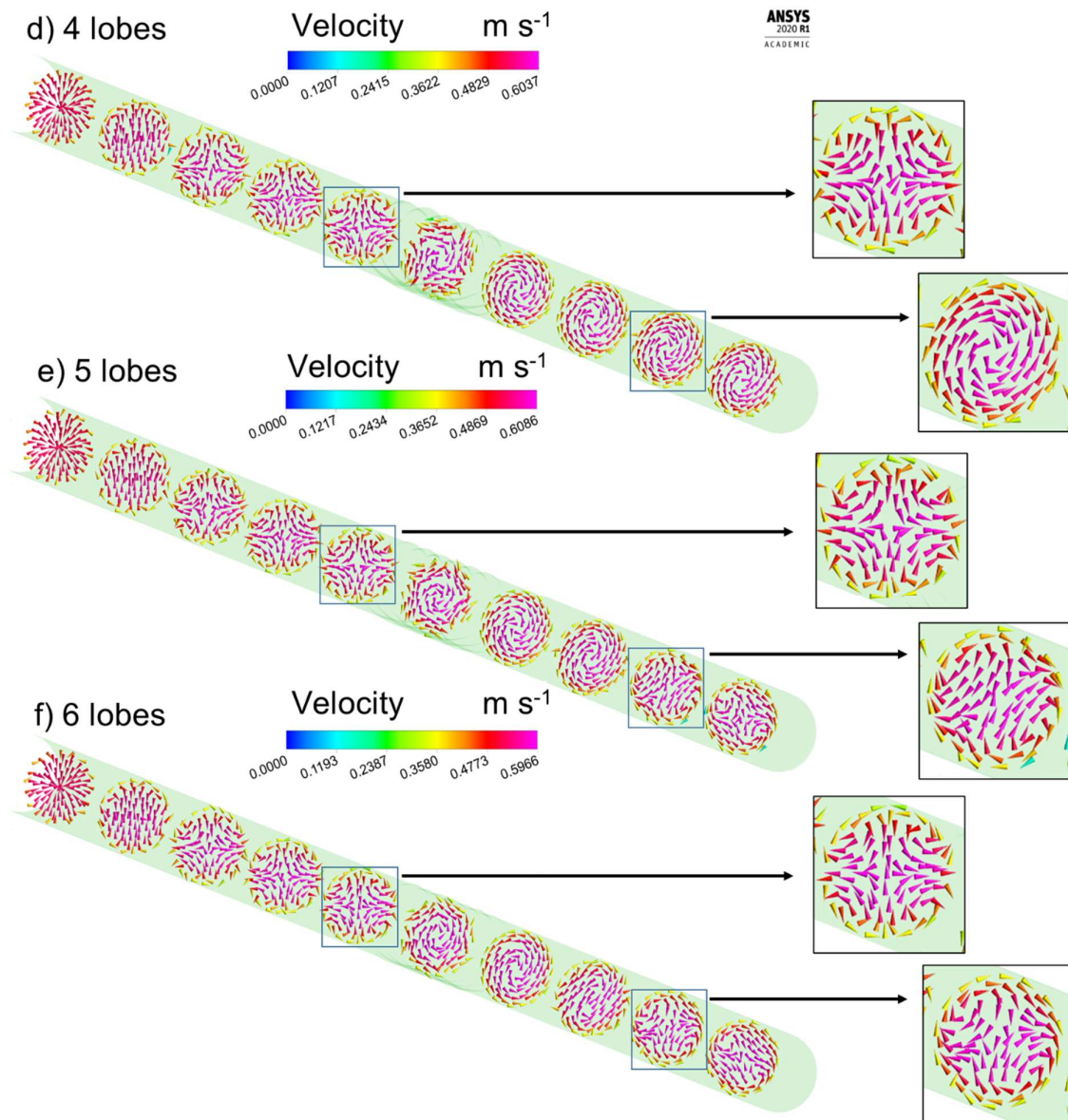


Figure 6.34b Secondary flow structure at a series of vertical planes represented by vector plots for a) untwisted (0 lobes), b) 2 lobes and c) 3 lobes twisted geometries.

Similar observations have been made in the case of 3 and 4 lobe geometries. In the case of 5 and 6 lobes geometries, the flow is observed to follow the twisted geometry for a very short length of the pipe. The flow in the downstream region is observed to nullify the induced rotation due to the generation of vortices in a direction opposite to the induced rotation. This is because the strength of the rotation is very small in the case of 5 and 6 lobes geometries.

6.6.3.2 *Turbulence in the flow:*

It is evident from the previous figures that the introduction of the twisted section results in an increase in the turbulence in the flow. To account for such changes in the turbulence, the volume averaged turbulence kinetic energy is analysed for the different pipe designs.

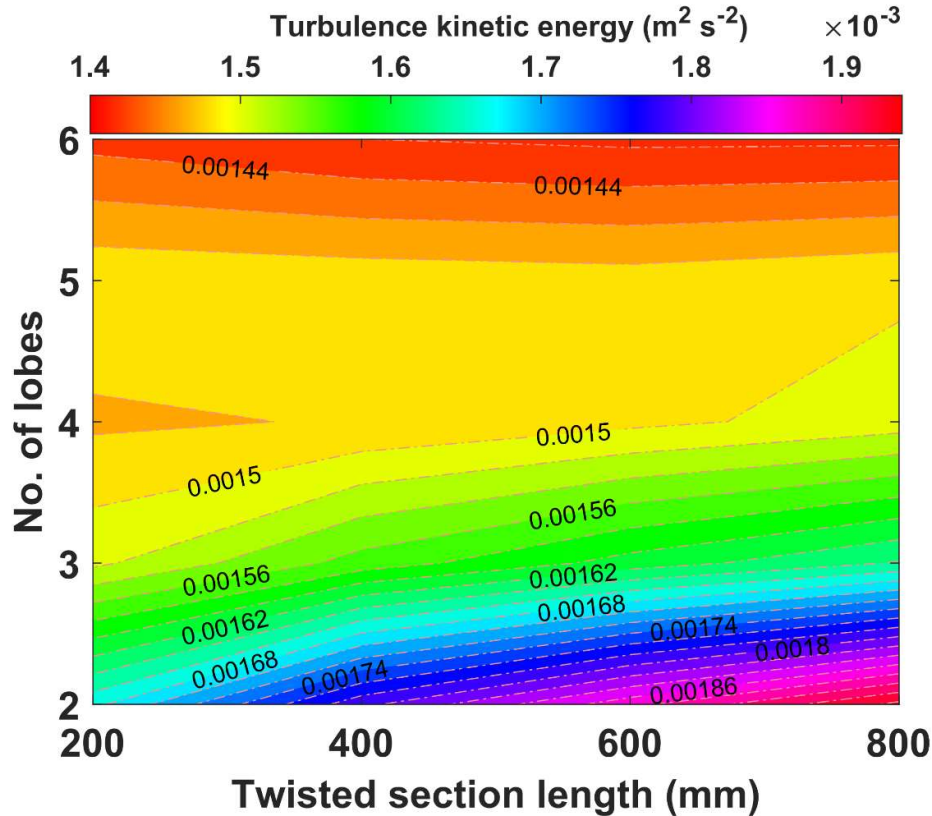


Figure 6.35 Solids' distribution along the flow stream of a) Untwisted, b) two lobes, c) three lobes, d) four lobes, e) five lobes, f) six lobes twisted sections of 800mm length.

Figure 6.35 shows the variation of turbulence kinetic energy of the flow with twisted section length and the number of lobes of the twisted geometry. The maximum turbulence is noted in the case of 2 lobe geometries. In the case of 2, 3 and 4 lobe geometries, the turbulence is found to increase with increase in the length of the section. However in the case of 5 and 6 lobes geometries, the turbulence remains almost constant irrespective of the length of the section.

In the present study, the turbulence induction is main principle to redisperse the already settled particles. Depending upon the design of the twisted geometry, the turbulence can be excessive, sufficient or insufficient. As the 3 lobes geometry is found to have the maximum mixing index so, the turbulence in the 3 lobe 200mm long twisted section can be called a the

favorable turbulence. The turbulence in the 2 lobe geometries being higher than the favorable values, can be regarded as excessive and in all other cases can be said to be insufficient.

6.6.3.3 Solids' distribution along the flow stream:

The CFD post Module of Ansys™ Workbench 2020 R1 is used to further analyse the results generated by Fluent. Contours of coal volume fraction have been plotted on a central streamwise vertical plane. Figure 6.36 shows the solids distribution of 200mm long twisted section at 40% solid concentration by mass. The solid phase distribution in untwisted pipe shows that the particles are very prone to settling and settle close to the very beginning of the pipe. However, the pipes equipped with twisted sections experience delayed particle settling in the upstream region. The downstream regions of the all the pipes indicate the redistribution of the particles into the fluid stream with different effectiveness.

The delay in the particles' settling can be due to the disturbance caused by the twisted section. The introduction of the twisted section, generates a significant turbulence in the downstream region. A part of this turbulence is also transferred to the upstream region, which leads to the delay in the settling of the particles. This can be regarded as a favorable outcome of the twisted sections.

The preliminary analysis of the solids' distribution in different pipes, represented in Figure 6.36 shows that the two-lobe geometry appears to be the most effective design for the redispersion of the settled particles. The effectiveness of the twisted sections appears to be diminishing with increase in the number of lobes as the downstream regions of the two-lobe geometry appear almost homogenous. Similar solid distribution can also be seen in the case of the three and four lobe geometries. In the case of five and six lobe geometries, the particle settling starts at a distance of ~300-400mm downstream the twisted section. As the number of lobes increase, the twisted cross-section approaches a circular cross-section, that lacks the capability of generating turbulence in the pipeline. Thus, the five and six lobe geometries are very ineffective in re-dispersing the settled particles. Similar trends can be seen in the cases of higher solids concentration of 50% and 60%.

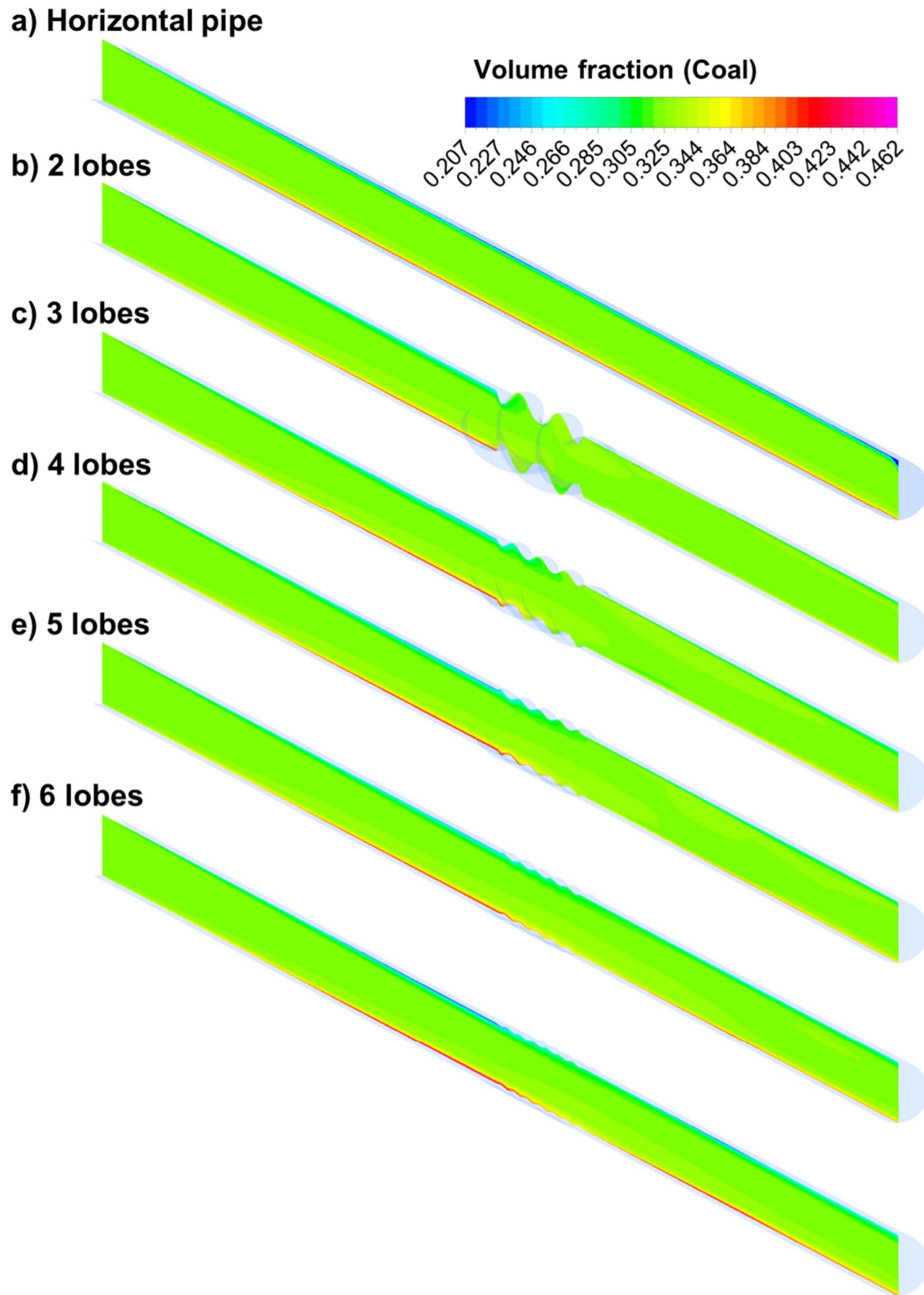


Figure 6.36 Solids' distribution along the flow stream of a) Untwisted/horizontal pipe, b) two lobes, c) three lobes, d) four lobes, e) five lobes, f) six lobes twisted sections of 200mm length.

6.6.4 Performance of the twisted sections:

6.6.4.1 Mixing index characteristics:

To get a better and clear idea about the effectiveness of the twisted sections, a parameter, “mixing index” has been introduced. The mixing index is the measure of homogenous nature of the slurry at a given plane perpendicular to the flowing stream and located in the downstream region. Mixing index calculations are based on the deviation of the in-situ coal volume fraction from the average, C_m . The mixing index can be calculated using equation 6.65:

$$MI = 1 - \left[\frac{1}{C_{va}} \left[\frac{1}{j-1} \sum_{l=1}^j (C_l - C_{va})^2 \right] \right]^{1/2} \quad 6.65$$

C_{va} : Average volumetric concentration at the plane.

C_l : Volumetric concentration at l^{th} grid point.

j : number of grid points at the plane.

l : Serial no. of the grid point.

In all the present cases, the MI is calculated by taking into account the values of the volumetric solid concentration C_l at 500-650 node points of the plane of interest. The results obtained from the Equation 6.49 are presented in the Figures 6.37a-b. Figure 6.37a and Figure 6.37b show the mixing index of all the twisted sections at 40, 50 and 60% solid concentration by mass.

The mixing index is calculated a fixed distance of 600mm from the twisted section in all the cases. The mixing index is observed to be almost independent of influx solid concentration as similar results have been obtained in Figures 37a-b. However, there is slight increase in the mixing index with increase in the influx solid concentration. This increase is relative and can be seen in all the cases, so overall trend remains the same. The mixing indices (at different influx concentration) in the case of the untwisted pipe, investigated at the same location are presented in Table 6.9.

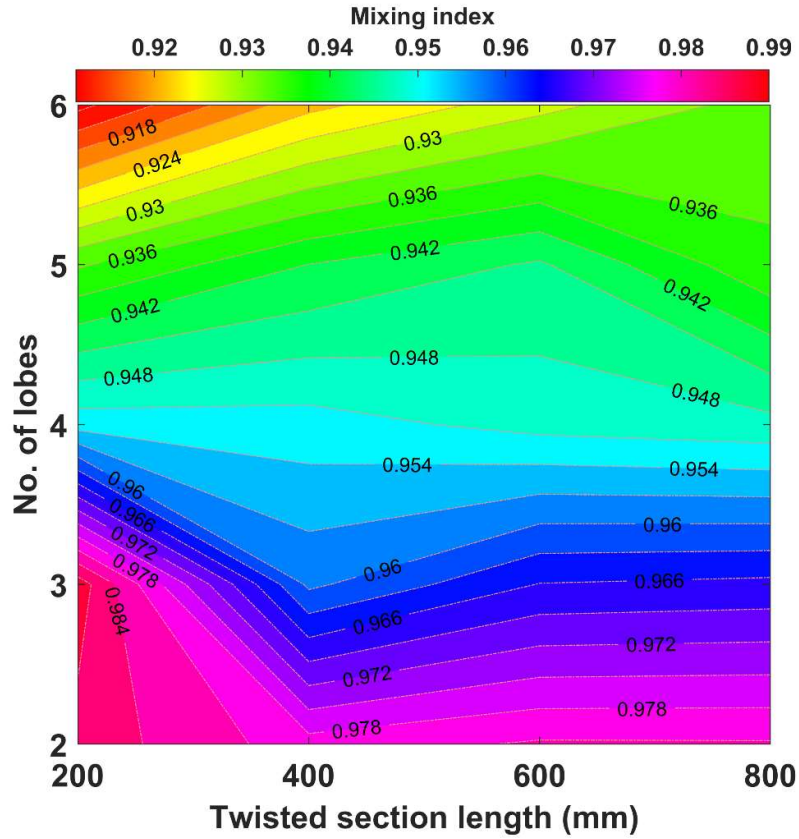


Figure 6.37a Variation of mixing index with no. of lobes and length of the twisted sections at 40% influx concentration

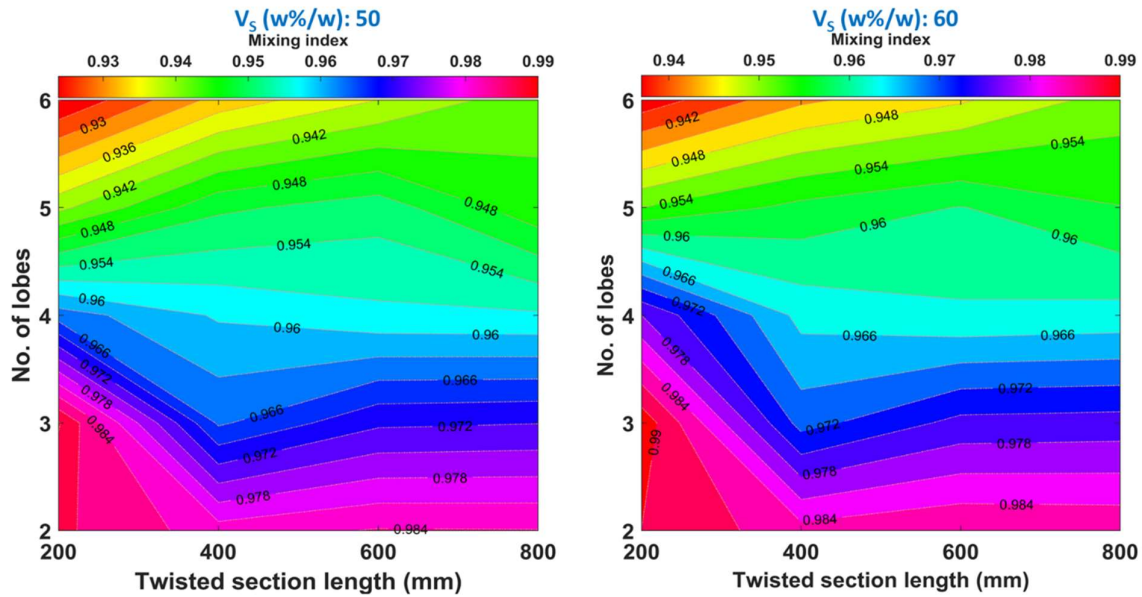


Figure 6.37b Variation of mixing index with no. of lobes and length of the twisted sections at 50 and 60% influx concentrations

Table 6.9 Mixing indices of the 50 mm untwisted pipe

Influx solid concentration (w%/w)	Mixing index
40	0.861
50	0.875
60	0.891

It is evident from Table 6.9 that due to the settling of the particles in the downstream region and location of analysis, the mixing index is found to be significantly diverting from the homogenous nature. As interpreted from Figures 6.33-6.36, the 2 lobe and 3 lobe geometries appeared almost equally effective. However detailed analysis of the twisted sections shows that the 3-lobe geometry is more effective than the 2-lobe geometry. Irrespective of solid concentration, the highest mixing index is obtained in the case of 200mm long 3 lobe geometry. The further increase in the no. of lobes results into the reduction of the mixing index. Similarly, the increase in the section length also leads decrease in the mixing index. However, this is not the case with the 6-lobe geometry, where the mixing index can be seen increasing with increase in the section length.

The above trends can be explained with the help of basic principle behind this process. In the flowing stream, the particles can be seen settled at some distance from the pipe inlet, depending on the influx conditions and downstream flow disturbances if any. Now to re-disperse the particles into the flowing stream, a particular range and amount of turbulence is required, that can be called as favorable turbulence. The turbulence lesser or greater than this range shall fall in the unfavorable range. It is certain that the turbulence generated by a twisted section is an inverse function of its no. of lobes. The 2-lobe geometry is observed to produce greatest turbulence inside the section and in the downstream region. However, compared to the 3-lobe geometry, this turbulence is on the higher side and lies in the unfavorable range. Due to the excessive turbulence, and higher centrifugal force, the re-dispersed particles are forced towards the pipe wall. In such a situation, slightly concentrated slurry can be found near the wall, that results into lesser increase in the mixing index.

The 3-lobe geometry generates turbulence, that is just sufficient to re-disperse the settled particles. For this reason, the mixing index of the 3-lobe geometry is the highest. Further

increase in the no. of lobes to 4, 5 and 6, leads to decrease in the turbulence, that appears to be insufficient for a homogenous efflux.

The increase in the length of the 2 and 3-lobe geometries by 200mm, adds to the turbulence produced by the shorter sections, thus forcing the particles towards the pipe wall. So, there is significant decline in the mixing index. The further increase in the section length shows slight improvement in the mixing index. This can be due to the decay of the turbulence in longer sections as a result of increased skin friction. Similarly, in the case of 4-lobe geometry, the mixing index reduces with increase in section length, but in the consecutive longer section, the skin friction being dominant, leads to substantial decline in the turbulence of the flow and the mixing index of the section. The case with 5 and 6-lobe geometries is slightly different, where the skin friction appears to be less dominant and there is no reduction in the turbulence. So, the longer sections show improved mixing characteristics.

6.6.4.2 Headloss and skin friction characteristics:

The flow of the slurry through twisted section becomes significantly turbulent, as compared to the fully developed flow through the untwisted pipeline. So, it is necessary to investigate the changes in the pressure drop characteristics of the pipeline with the introduction of twisted pipe sections.

The head loss characteristics of the twisted sections at solid concentration of 40, 50, and 60% are presented in Figures 6.38a-b respectively. The head loss characteristics are slightly influenced by the solid concentration and show a slight increase with increasing solid concentration. However, the trend across the different twisted sections remains the same. In all the geometries, the head loss is found to decrease with no. of lobes and increases with section length. So, the 2-lobe geometries experience the highest head loss.

The head loss is observed to magnify by many folds in the longer sections of the 2-lobe geometry. But this is not the case with the other geometries, where the head loss is observed to increase almost linearly with section length. The 3-lobe geometries experience substantially lesser head loss as compared to the 2-lobe geometries. Upon further increase in the number of lobes, the head loss moderately decreases irrespective of the section length. The 5 and 6 lobe geometries experience almost similar head loss.

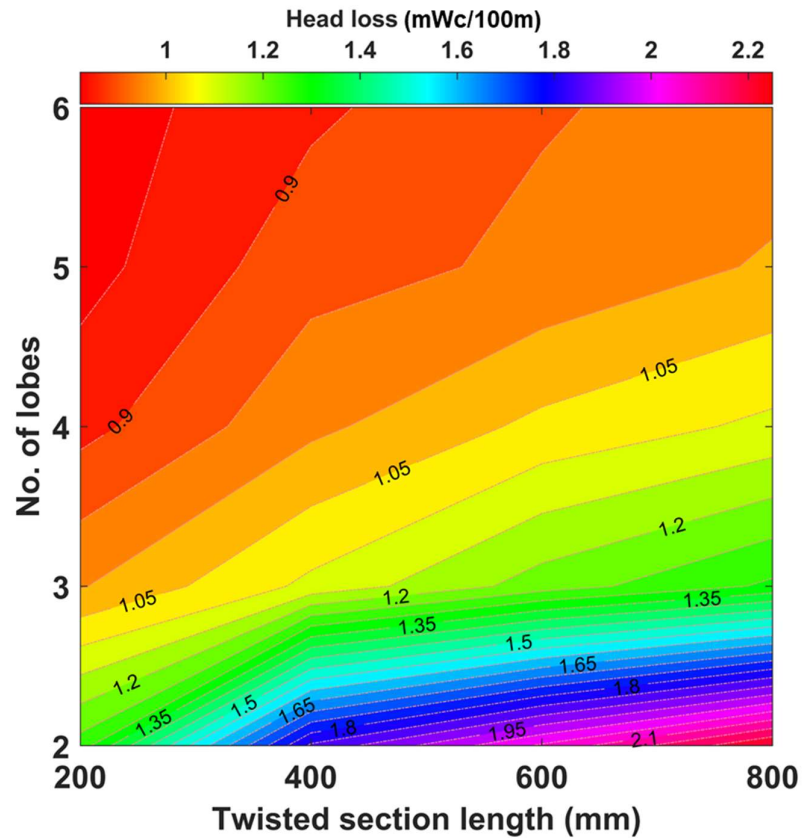


Figure 6.38a Head loss characteristics of different twisted sections of variable length at 40% influx concentration

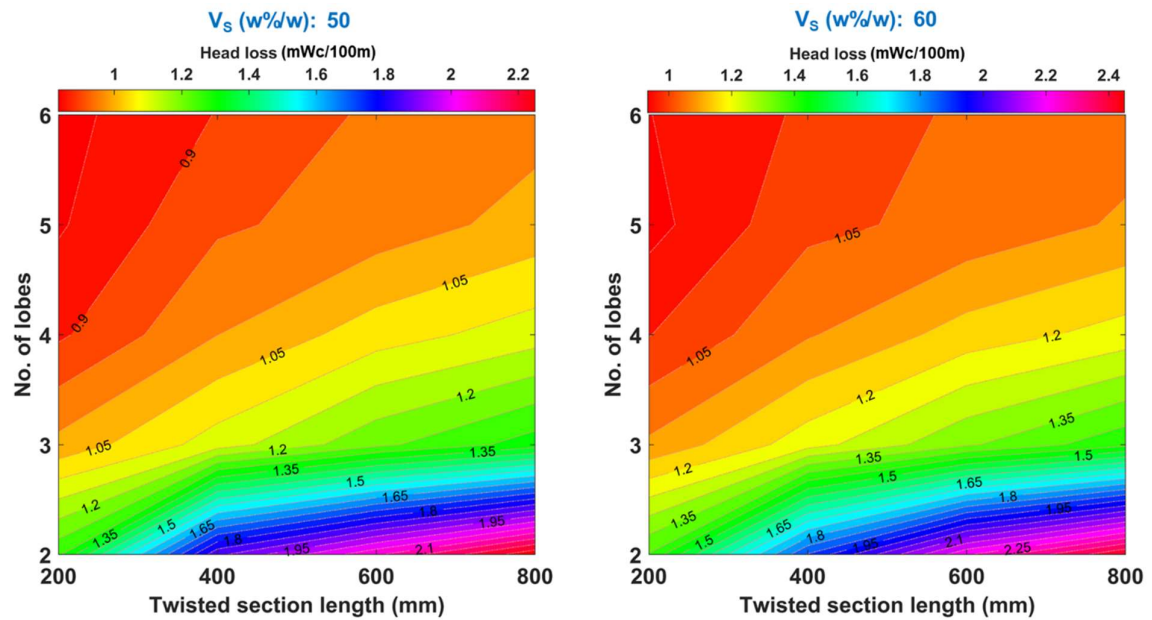


Figure 6.38b Head loss characteristics of different twisted sections of variable length at 50 and 60% influx concentration

Table 6.10 Head-loss across the 50mm untwisted pipe

Influx solid concentration (w%/w)	Head loss (mWc/100m)
40	0.807
50	0.875
60	0.954

The head loss across the untwisted pipe at 40, 50, and 60% solid concentration is presented in Table 6.10. It can be seen that the introduction of the twisted section leads to increase in the head loss significantly. This is possibly due to the increase in the skin friction due to the near wall motion of the slurry. The head loss characteristics of the twisted section are explained in relation to the skin friction characteristics of the slurry. These characteristics are presented in Figures 6.40-6.43 for 200, 400, 600 and 800mm long twisted sections respectively at 40% influx solids' concentration. Similar characteristics are also observed in the cases of 50 and 60% solids' concentration. The 200mm long twisted section experience the relatively lower head loss as compared to their respective longer sections.

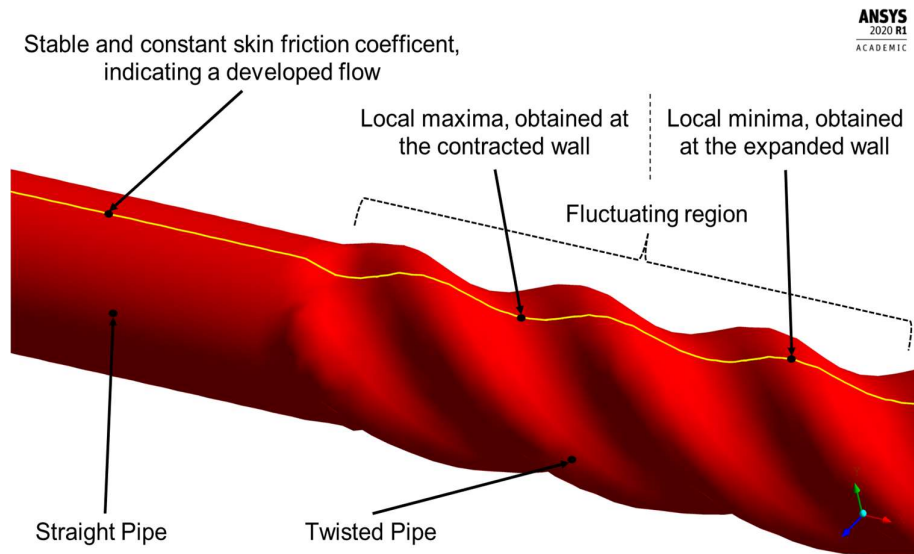


Figure 6.39 Representation of the regions of the pipe based on the skin friction characteristics. The upstream region of the twisted pipe shows a developed flow and skin friction coefficient has a constant value. The skin friction starts fluctuating in the twisted section and hence named as fluctuating region. The crest and trough of the fluctuation are obtained at the contracted and the expanded walls respectively.

Among the 200mm long twisted section, the 2 lobes geometry experience the highest head loss as the skin friction coefficient of this geometry is the highest. Due to excessive turbulence and centrifugal force, the particles-liquid mixture is forced towards the wall, where they experience heavy frictional drag, that contributes to the head loss. The skin friction values appear to fluctuate in the twisted section as presented in Figures 6.40-6.43. The reason for this fluctuation can be related to the conceptual representation in Figure 6.39. The skin friction coefficient shows significant rise at the contracted wall due high wall shear that is mainly due to local acceleration of the flow in the region.

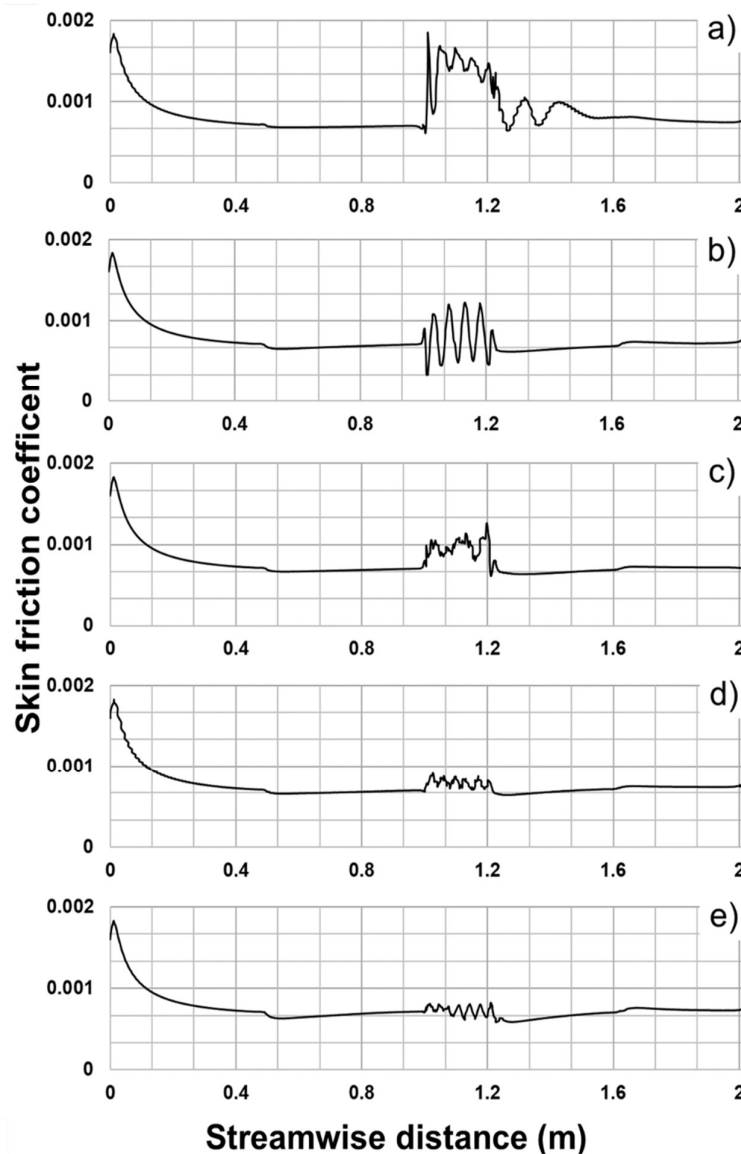


Figure 6.40 Skin friction coefficient of a) 2 lobes, b) 3 lobes, c), 4 lobes, d) 5 lobes and e) 6 lobes geometry and 200m length at 40% influx solid concentration by mass.

Similarly, a sudden decrease in the skin friction coefficient is noted at the expanded walls. This is due to reduction in the wall shear as the flow is highly decelerated in these regions. In the case of 2 lobes geometry, the values fluctuate around a mean value, that is significantly higher than the skin friction of a fully developed flow. While in all other geometries, the fluctuation can be seen around a mean value close to the skin friction of a developed flow. The 200mm long 3 lobes geometry also exhibits high frictional drag characteristics but it is significantly lower than the 2 lobes geometry. The increase in number of lobes makes the twisted profile smoother and leads to reduction in the amplitude of fluctuation of skin friction values.

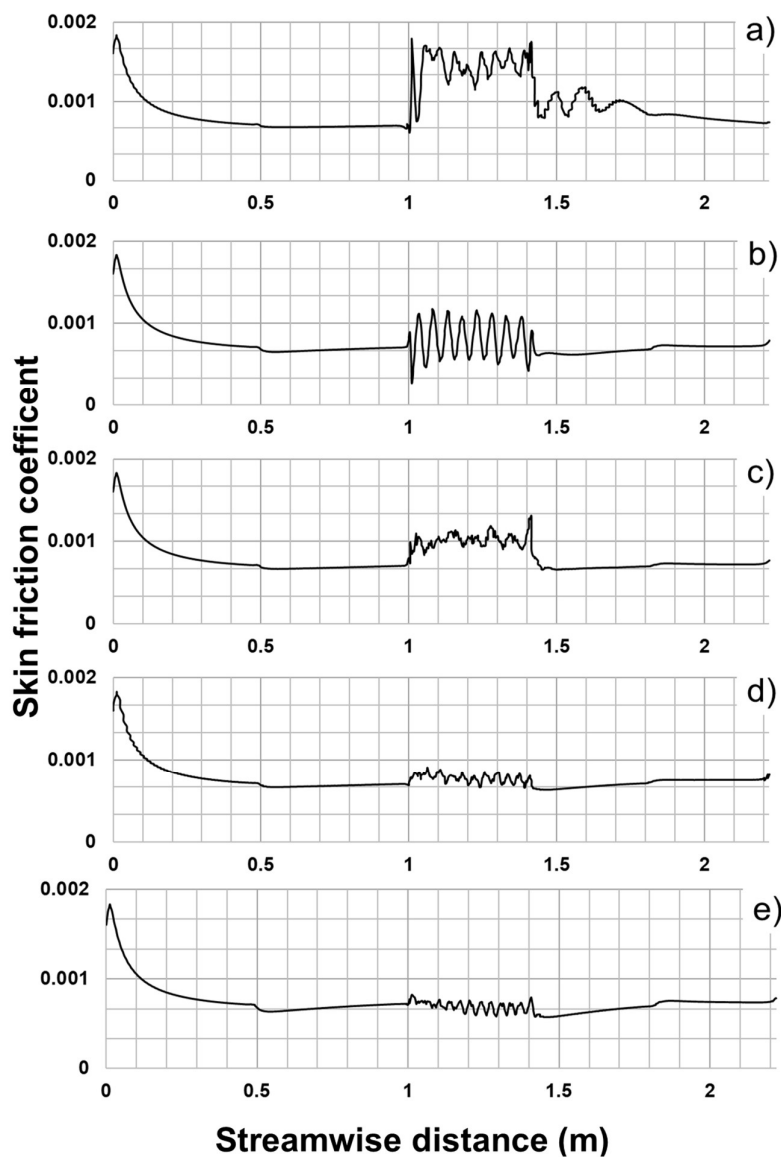


Figure 6.41 Skin friction coefficient of a) 2 lobes, b) 3 lobes, c), 4 lobes, d) 5 lobes and e) 6 lobes geometry and 400mm length at 40% influx solid concentration by mass.

The skin friction characteristics of the 400mm long twisted sections are presented in Figure 6.41. The results are quite similar to the 200mm twisted section. The amplitude of fluctuation of the skin friction is almost comparable to the 200 mm twisted section and remains almost same throughout the twisted section. The fluctuation can be seen on a longer length of the pipe.

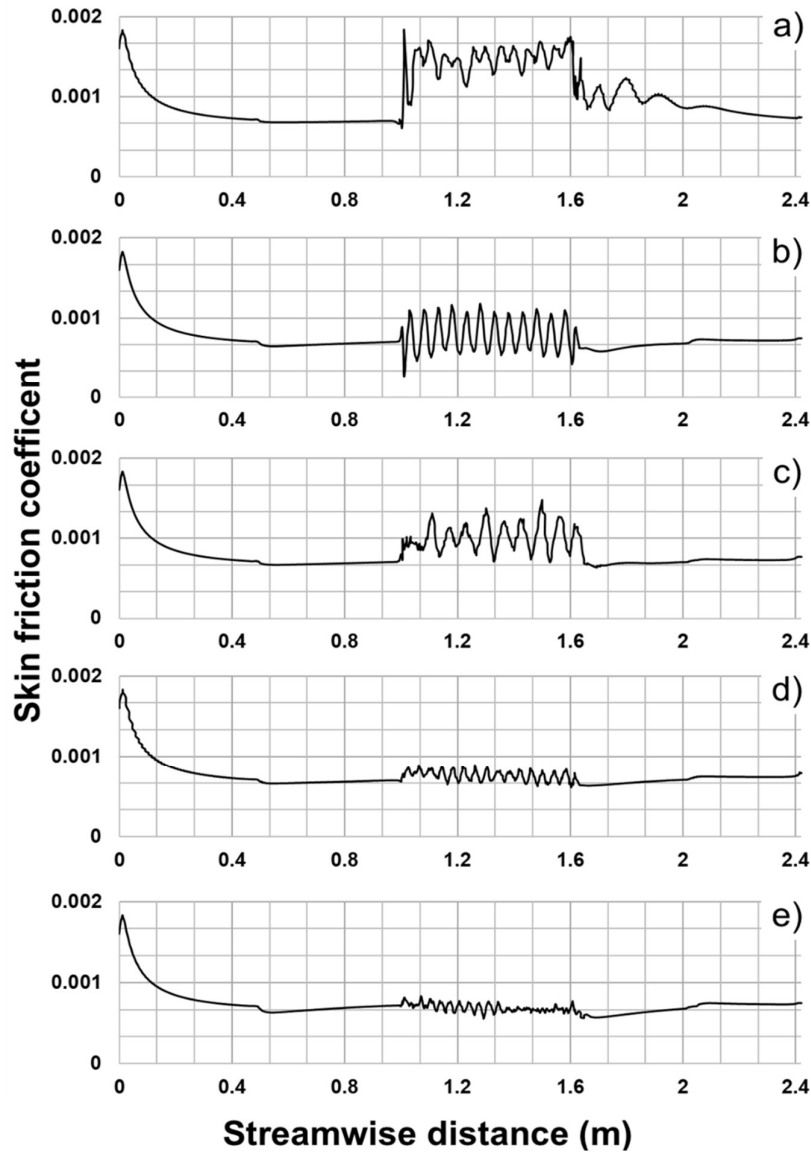


Figure 6.42 Skin friction coefficient of a) 2 lobes, b) 3 lobes, c) 4 lobes, d) 5 lobes and e) 6 lobes geometry and 600mm length at 40% influx solid concentration by mass.

Figure 6.42 represents the skin friction characteristics of the 600mm long twisted sections. All the twisted sections show same characteristics as the shorter sections show except the 6 lobes geometry, where the skin friction is observed to diminish in the downstream region,

indicating the partial development of flow and boundary layer in the region. However, in other geometries, there is no sign of such boundary layer development.

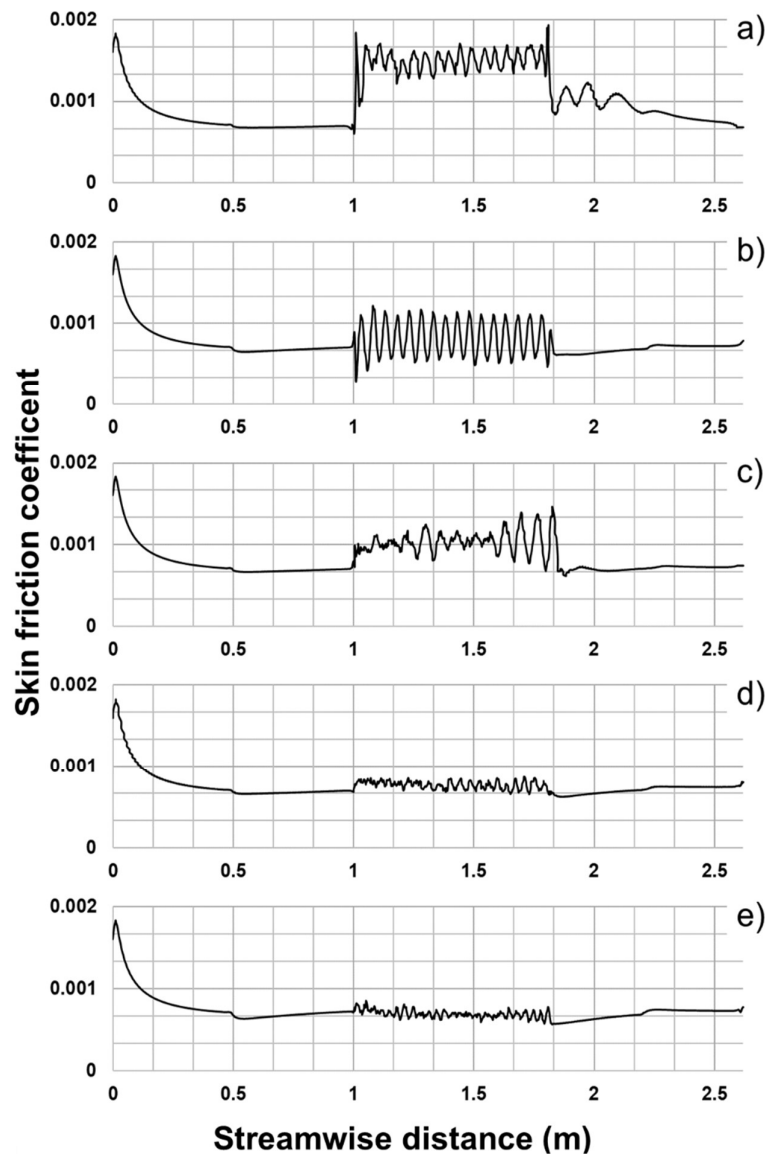


Figure 6.43 Skin friction coefficient of a) 2 lobes, b) 3 lobes, c), 4 lobes, d) 5 lobes and e) 6 lobes geometry and 800mm length at 40% influx solid concentration by mass.

In Figure 6.43, the skin friction characteristics of the 800mm long twisted sections are presented. The results indicate no change in the skin friction throughout the 2-lobes and 3-lobes twisted sections. But an increase in the skin friction is observed in the downstream region of the 4, 5, and 6 lobes geometries. This is mostly due to the increase in centrifugal force as the slurry tries to follow the twisted path. As a result, the particles experience constant contact with the wall.

6.6.4.3 Specific energy consumption:

The specific energy consumption is a dimensionless parameter that has a great practical significance. Specific energy consumption is the measure of energy required to transport a unit weight of bulk solids by a unit distance. In the present study, the specific energy consumption is calculated using equation 6.66 as published in [179]:

$$SEC_w = \left| \frac{\Delta p}{\Delta l} \right| \times \frac{1}{\rho_s V_{sd} g} \quad \text{Units} = \text{J/N-m} \quad 6.66$$

In practice, the mass is used to quantify the solids instead of the weight. To make the analysis more practical and easier to understand, the equation 6.66 has been modified to equation 6.67. The equation 6.67 presents the SEC as the measure of energy required to transport “a unit mass” of bulk solids by a unit distance. It can be seen that the SEC in terms of mass is “g” times the SEC in terms of weight.

$$SEC_m = \left| \frac{\Delta p}{\Delta l} \right| \times \frac{1}{\rho_s V_{sd}} \quad \text{Units} = \text{J/kg-m} \quad 6.67$$

In equations 6.66 and 6.67, $\frac{\Delta p}{\Delta l}$ is the pressure loss per unit length and is measured in Pa/m, ρ_s is the density of solids/secondary phase in kg/m³, V_{sd} is the volumetric concentration of the solids, that is actually delivered. The V_{sd} is calculated using the homogeneity/mixing index as presented in equation 6.68:

$$V_{sd} = MI \times V_s \quad 6.68$$

Here V_s is the influx volumetric concentration of the solids. The SEC characteristics of the untwisted pipe are presented in Table 6.11

Table 6.11 Specific energy consumption in untwisted pipe

Influx solid concentration (w%/w)	SEC (J/kg-m)
40	0.204
50	0.169
60	0.146

The SEC characteristics of the different twisted pipes of 200, 400, 600 and 800mm length are presented in Figures 6.44a and Figures 6.44b respectively.

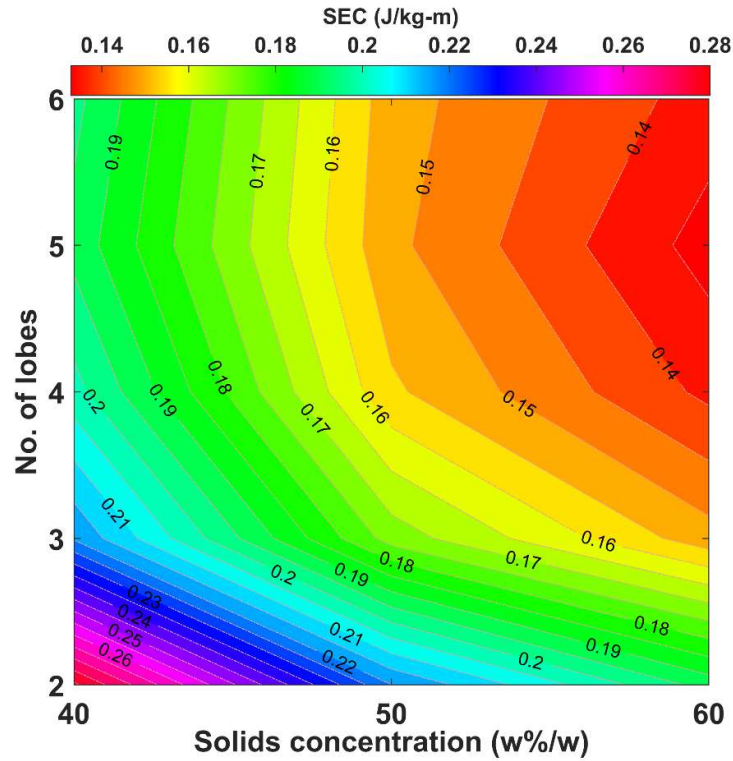


Figure 6.44 a) Specific energy consumption for the transportation of solids at influx concentrations of 40-60% through different twisted sections of 200 mm length.

The SEC characteristics of the all the twisted pipes indicate the reduction in the SEC at higher solids concentration and the lowest values are obtained at 60% solids concentration (w%/w). So, despite the higher headloss at 60% solids concentration, the SEC is lower. The SEC followed a decreasing trend with increasing number of lobes. In some cases, the 5 lobes geometry showed the lowest specific energy consumption. The headloss characteristics of the 5 and 6 lobes are almost comparable in most of the cases, but the mixing indices of the 5 lobes geometries are higher than 6 lobes geometries, so the specific energy consumption of 5 lobes geometries is lower than the 6 lobes geometries in some cases. In all the cases, the 5 and 6 lobes geometries show the lowest specific energy consumption due to significantly lower headloss, with small decrease in the mixing index.

Comparing the SEC across different lengths of the twisted pipe section, indicates a significant increase in the SEC in the 2 lobes geometries. This is due to the substantial increase in the headloss in the longer 2 lobes geometries. In all other geometries, there is small increase in the SEC and this is also due to the higher headloss in the longer pipe sections. So, the 200mm long twisted pipe sections are observed to have lowest specific energy consumption.

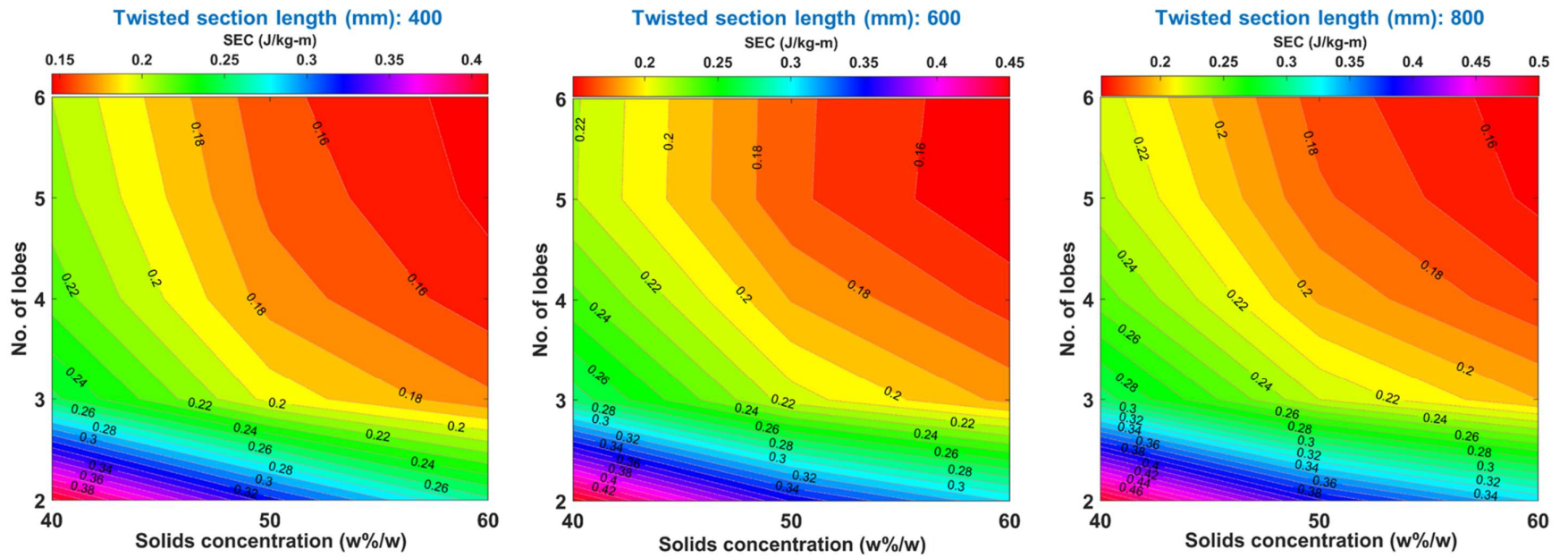


Figure 6.44 b) Specific energy consumption for the transportation of solids at influx concentrations of 40-60% through different twisted sections of 400 mm length.

6.7 Concluding remarks:

In the present chapter, the commercial CFD software ANSYS Fluent is used to simulate the solid-liquid flow inside pipe fittings like an elbow, a converging section and a diverging section. The results generated by different turbulence models are compared with the experimental results obtained in the present study. The following conclusions are drawn from the results obtained:

1. The SST $k - \omega$ model shows the closest resonance to the experimental results because it uses both the $k - \epsilon$ and $k - \omega$ models for the accurate freestream and near wall flow respectively
2. The headloss across the elbow is observed to increase by around 10% with concentration of coal. The headloss at 5 m/s becomes 105 times of the headloss at 0.5 m/s. So, the headloss increases non linearly with influx velocity. The bendloss associated with the elbow is observed to decrease with by 10-14% when velocity is increased from 0.5 m/s to 5 m/s. However, an increase of 6-7.5% in the bendloss is observed with increase in concentration of coal from 10 to 60%.
3. Headloss across the shortest and longest converging sections is around 116% higher than that across 0.15m long converging section. The 0.15m long converging section has pressure recovery 0.3 to 4.9% higher than any other design. The shortest and longest sections have friction factor 183% and 54% higher than the 0.15m long converging section.
4. The 0.3m long diverging section has headloss 1 to 20% lower than other geometries and 84% lower than the shortest geometry. The 0.3m long diverging section has pressure recovery 0.4 to 5.5 higher than other geometries and 22.3% higher than the shortest section. Similarly, 0.3m long diverging section has highest volumetric efficiency. So, this design of diverging section is optimum as it is able to balance the losses due to velocity gradient and skin friction efficiently.
5. The headloss across the twisted section is 5 to 155% higher than the straight untwisted pipe. The 3-lobes 200mm geometry was able to produce highest homogeneity of 99.2% in the slurry with a slight increase (4.23%) in the specific energy consumption.

Chapter 7

Conclusion and future scope

7.1 Conclusion:

In the present study, different methods for the improvement of transportation characteristics of coal water slurry are proposed. The study is addressed to the mining industries and the power generation systems. This study, specifically numerical modelling of turbulent flow, serves as motivation for a variety of industrial flows such as in chemical industries.

The experiments of the present study are conducted with an aim of improving the flow characteristics of the Indian low rank coal using physical and thermal treatment processes. The research supports the use of hydrothermal treatment and microwave irradiation for the upgrading of low-rank coals. The Indian coal was able to couple with the microwaves due to the presence of moisture and mineral matter composed of Fe, Mg and S. In hydrothermal treatment process, the low-rank coal was treated for 30, 45, 75, and 120 minutes. Similarly, the same coal was subjected to a fixed input microwave power but for different time intervals, namely 30, 60, 90 and 120 s. The coal samples showed an increase in the fines due to the destruction of pores by the locked moisture upon HT or MW treatment. The moisture and light volatiles escape the coal particle, making it more hydrophobic. The treated coals were stable for a long duration also the concentration of the settled bed was lower as compared to the untreated coal. The increase in the treatment times showed a linear improvement in the physico-chemical and rheological characteristics of coal and CWS, respectively. In the other part of the experimental study, it was observed that the particle size distribution strongly affects the stability and flowability of the coal water slurry. The study also describes an economical way to utilize the low rank coal by blending with MW treated coal. The specific conclusions that have been drawn from the experiments of the present study are:

1. The present coal sample shows average settling characteristics due to high ash (34.73%) and high volatile matter (18.78%). The maximum settled concentration of the hydrothermal and microwave treated samples are reduced by 22.2 and 23.8%. Similarly, the maximum settled concentration for the fine sample blended with MWT coal is reduced by 27.6%. The 30% proportion of -75+53 μm coal or 20% proportion of -106+75 μm coal are observed to be optimum for increasing the stability of the suspended particles. The addition coarse particles beyond the above values lead to setting of the coarse particles.

2. The thermal treatment using the HTT and MWT processes helped in the removal of oxygen by 34.43% and 38.65% respectively from the coal. The median diameters of the untreated, HTT120 and MWT120 coals are 51.7 μm , 32.2 μm , and 22.4 μm respectively. Thus, these processes also caused the breakage of the particles, thereby destructing the pore structure of the coal.
3. The CWS behaves like a Newtonian fluid up to a solid concentration of 30% by mass and the apparent viscosity is observed to increase with solid concentration. However, the increase in the solid concentration beyond 30%, leads to change in the behavior of the slurry and a pseudoplastic behavior is exhibited. Rise in the apparent viscosity is observed at 50 and 60% concentration of the coal as the hydrophilic matter absorbs most of the water present in the slurry. The water deficiency leads to increase in the viscosity of the slurry.
4. The thermal treatment of the coal is helpful in modifying the surface morphology of the coal by eliminating the hydrophilic oxygen functional groups. So, the viscosity of the coal water slurry is improved by the thermal treatment of the coal. The apparent viscosity of the slurry is observed to reduce by 75.41 and 77.51 when treated via HTT and MWT for 120 minutes and 120 s respectively. The maximum reduction in the apparent viscosity is observed to be 43.2% and 41.4% in the case of addition of -75+53 μm and -106+75 μm sized coarse particles respectively.
5. The addition of coal to the water makes it a complex fluid, as a result the headloss of the flow increases to 2.7 to 3.6 times of the headloss for water. The addition of -75+53 μm particles in 30% proportion to the fine particulate slurry leads to reduction in the headloss by a maximum of 55.9%. Similarly, the addition of -106+75 μm in 20% proportion leads to reduction in the headloss by a maximum of 55.7%. The SEC is observed to decrease by around 78%, when the concentration of coal in the slurry is increased from 10 to 60%.

The part-II of the study is based on the use of the computational fluid dynamics approach for the prediction and optimization of slurry flow through common pipe fittings viz. a 90° elbow or miter bend, converging section and diverging section. These pipe fittings are an integral part of the slurry pipeline. Proper knowledge about the flow characteristics through these pipe fittings can save a huge capital cost and prevent any minor or major system break-down. The specific conclusions drawn from the study are:

6. The SST $k - \omega$ model shows the closest resonance to the experimental results because it uses both the $k - \epsilon$ and $k - \omega$ models for the accurate freestream and near wall flow respectively.
7. The headloss across the elbow is observed to increase by around 10% with concentration of coal. The headloss at 5 m/s becomes 105 times of the headloss at 0.5 m/s. So, the headloss increases non linearly with influx velocity. The bendloss is a function of friction factor, that is inversely related to the Reynold's number. The bendloss associated with the elbow is observed to decrease with by 10-14% when velocity is increased from 0.5 m/s to 5 m/s. However, an increase of 6-7.5% in the bendloss is observed with increase in concentration of coal from 10 to 60%.
8. The 0.05m and 0.6m long converging sections have the lowest performance due to increased skin friction. The 150mm long converging section is observed to show the lowest headloss at all influx concentrations and velocities. This design of the converging section has the pressure recovery of 78%, highest than any other geometry.
9. The 200mm 3 lobes geometry shows highest effectiveness in re-dispersing the settled solids, with slight increase of 0.365mWc in head loss per 100m of pipe length compared to the original head loss of 0.954mWc per 100m of untwisted pipe. However, the specific energy consumption in this pipe design is quite comparable to the untwisted pipe and there is a small increase of 4.23% in the SEC. At the cost of this small increase, a critical issue of the particle settling can be effectively addressed.

The practical application of this research will significantly reduce the hydrotransportation losses due to the improvement of the rheology of the slurry. The volumetric efficiency of the system is expected to be higher as the treated coals have higher dispersion stability and there will be lesser chances of the particles settling at the pipe bottom. The optimum designs of converging and diverging sections have been successfully identified, that assure lowest possible loss of head and highest possible pressure recovery. The twisted pipes can certainly be used for the transportation of hydrophilic materials such coal ash, biomass ash, sand and mineral tailings at very low influx velocities with least possible chances of particles settling.

7.2 Future scope:

The hydrotransport of the industrial solids is an emerging area of science, that needs some additional work to be carried out for the addition of knowledge and information to this field. Some of the prospective works in this study can be:

1. Study of real-time distribution of the particles inside the flow domain using EIT (Electrical Impedance Tomography)
2. Investigation of erosion in pipelines equipped with twisted sections
3. Study of erosion in bends equipped with twisted section.

Appendix A

Headloss of the coal water slurry

Table A.1 Headloss (mWc/100m) for the flow of coal water slurry through a pipe of 100mm diameter

Influx Velocity (m/s)	Concentration of coal (-53 μ m)						
	0	10	20	30	40	50	60
0.25	0.0763	0.33	0.385	0.402	0.425	0.472	0.638
0.5	0.260	0.728	0.750	0.773	0.819	0.864	1.102
0.75	0.537	1.54	1.57	1.591	1.638	1.68	1.992
1	0.9	2.26	2.3	2.393	2.588	2.869	3.02
1.25	1.30	3.428	3.530	3.671	3.903	4.136	4.490

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